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JULY, 1916



# Bulletin of the American Institute of Mining Engineers

## PUBLISHED MONTHLY

### SECTION I



### SECTION II.—PAPERS AND DISCUSSIONS

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.



# Stoping in the Calumet and Arizona Mines, Bisbee, Ariz.

BY PHILIP D. WILSON,* B. S., E. M., WARREN, ARIZ.

(Arizona Meeting, September, 1916)

THE mines of the Calumet & Arizona Mining Co. are situated in the Warren Mining District, Cochise County, Arizona, between Bisbee and Warren and adjoin those of the Copper Queen Consolidated Mining Co. and the Shattuck Arizona Copper Co. Paleozoic limestones of considerable thickness have been intruded by granite porphyry. A great stock of the intrusive outcrops boldly near the city of Bisbee and is known as Sacramento Hill. From this central core the porphyry has tongued out into the limestone for many thousands of feet as dikes and sills of irregular shape and variable dimensions. Churn drilling has recently developed a large tonnage of secondarily enriched copper ore in the main porphyry mass of Sacramento Hill. While some ore has been found in porphyry in other parts of the district it is relatively unimportant compared with that which occurs in the Paleozoic limestone as irregular lenses replacing favorable beds near porphyry contacts or associated with well-marked fracture zones often many hundreds of feet away from the intrusive.

In a zone extending from Sacramento Hill for a distance of from 1,500 to 3,000 ft. the limestone has been extensively metamorphosed and in this contact metamorphic zone the ore occurs through a known vertical thickness of upward of 900 ft. and individual orebodies often show extension in a vertical direction of over 200 ft. At a greater distance from the intrusive stock the ore occurs as lenses of greater horizontal than vertical dimensions with an average thickness of approximately 30 ft. An exception to this rule is found in the occasional steeply dipping mineralized fractures, almost true tabular veins, which are found long distances away from the zone of contact metamorphism.

The copper was probably introduced during or slightly later than the porphyry intrusion in the form of chalcopyrite and bornite. These primary copper minerals are always associated with pyrite and often occur as rich lenses in a huge mass or shell of pyrite carrying an insignificant percentage of copper. In many instances the copper-bearing solutions

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were preceded by siliceous solutions and these latter, often persisting during the period of mineralization, have thoroughly indurated the orebody and the surrounding country rock. Oxidation took place during two distinct periods separated by a period of submergence in the Cretaceous sea and subsequent elevation, distortion and tilting of the sediments. This has resulted in a most irregular zone of oxidation. In some cases enriched chalcocite ore and even thoroughly oxidized ore are found stratigraphically far below primary orebodies. Where oxidation has been complete the carbonates and oxides of copper occur beneath or entirely surrounded by a mass of oxides of iron and aluminum, often of great extent. While there has been considerable enrichment with development of chalcocite in the district, opinion, in view of recent developments of the primary ores, is tending toward the conclusion that migration of values has been for the most part local and that both the oxidized and the chalcocite ores depend for their present copper content rather on rich primary sulphides than on any extensive leaching and subsequent enrichment of low-grade material. The large masses of iron and aluminum oxides found in connection with the oxidized ores are then the oxidation products of the shells of lean pyrite and pyritic limestone which surround the primary copper sulphides rather than gossans from which appreciable copper has been leached.

## Conditions Determining Stoping Methods

The great variety of ore occurrences in the district and the equally variable character of the country rock in the vicinity of the orebodies render any uniformity of stoping method impossible where the factors of economy and efficiency are considered. In the early days of Bisbee all of the ore mined was taken from the irregular high-grade oxidized bodies found within a comparatively short distance of the surface. Square-set stoping with subsequent filling was used with much success and even today this time-tried method of Philip Deidesheimer is recognized as the system most widely applicable to the Bisbee ores. It was not until comparatively recent years that the natural reluctance to try new methods while the old could still be used was overcome, in the Calumet and Arizona mines under the superintendence of W. B. Gohring, with resultant reduction of costs in some cases to less than one-half of those formerly obtained.

The considerations determining the stoping method to be used may be summarized as follows:

1. Safety and efficiency of working conditions for the men.
   (a) Physical characteristics of ore and country rock.
   (b) Ventilation.
2. Maximum ultimate profit.
   (a) Cost of mining.

(b) Ore which may be economically sacrificed.
(c) Relative economy of mining and of sorting out included waste.
(d) Rapidity with which it is advisable to mine, or daily tonnage required.

3. Shape and dimensions of orebody.
4. Safety of mine.
(a) Cost of timber.
(b) Availability of filling.
(c) Fire risk.

## Square-set Stoping

The square-set system is easily the most flexible and where the orebody is very irregular and has large included blocks of waste it is the most satisfactory. It is a simple matter to leave the waste behind as a portion of the filling, and where the mining of too large a section at once is not attempted and filling is kept within a reasonable distance of the back it is as safe or safer than any of the other methods in vogue. By this method the cost* of stoping per ton including labor, powder, timber, carbide and air, ranges from $0.80 in sulphide ore to $1 or even $1.30 in oxide ore. Where the ground is very heavy, additional timber as bulkheads and double-up sets increases the cost materially and in sulphide ore the forest of timber makes the risk of fire an important consideration. In a normal square-set stope the item of timber, which laid down at the mine costs about $17.50 per thousand, amounts to 25 per cent. of the total.

### *Recovering Square-set Stope Timbers*

The first attempt to reduce this expense was made some years ago by M. W. Mitchell, foreman of one division of the Calumet and Arizona mines, who devised a method for "robbing" a considerable portion of the timber from a square-set stope during the process of filling. Approximately 50 per cent. of the timber can be recovered where conditions will permit of this robbing process, and the actual stoping cost based on a series of stopes over a considerable period in which the timber has been recovered and used again is reduced by about 8 per cent. While applicable to a large majority of square-set stopes, in many instances the ground is too heavy to risk removing the timbers and in some cases the timber is so badly damaged that it is not worth extracting. The method finds its greatest economy where an orebody is being mined in successive sections so that the recovered timber may be left in open square sets at the edge of the stope and used when the adjacent section is being worked, as the cost of excessive handling soon reduces the saving to a minimum.

When an attempt to rob the timbers from a square set is anticipated the first section is carried up five sets wide, as shown in Fig. 1. The

* These costs obtain under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers. During the past year the sliding wage scale and the abnormally high cost of supplies have resulted in a material increase in all mining costs.

length of the section is determined by the character of the ground and varies from five to as many as ten sets. It is advantageous to carry the section as long as the ground will permit, since the longer the section the greater will be the ultimate timber recovery. It is usually not feasible to attempt to recover timber from a stope over 50 ft. in height. When an orebody, the thickness of which is greater than this, is to be mined and the timber recovered by this method the operation is carried on in successive lifts, as will be hereinafter described. In stoping the section a gangway is maintained in the central row of sets on the sill floor and chutes, *ee*, built in each alternate set on either side. Slides at appropriate places in

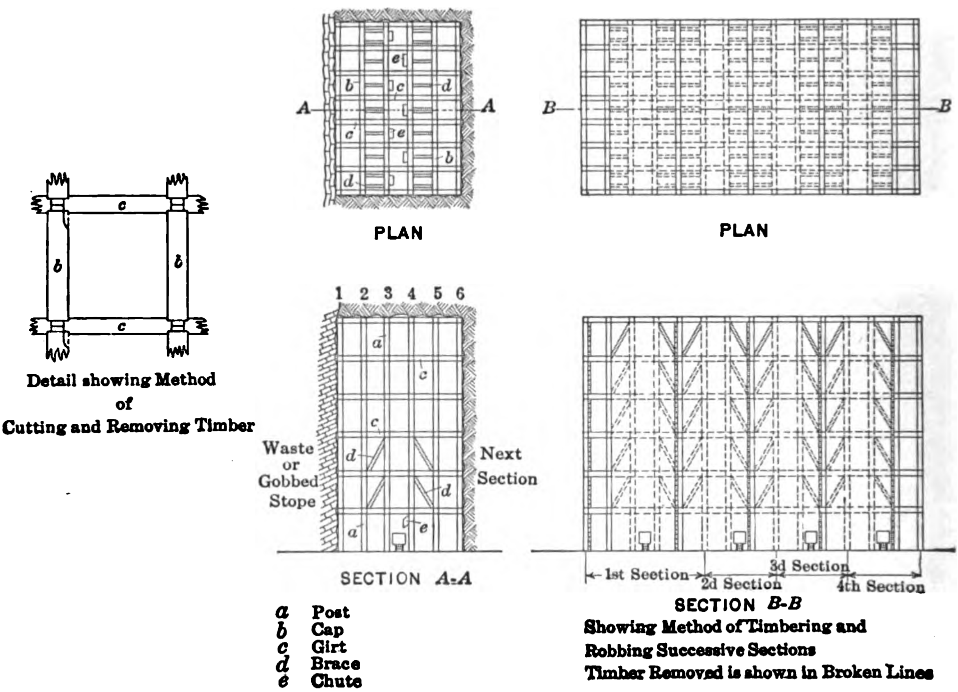



FIG. 1.—MITCHELL METHOD OF RECOVERING SQUARE-SET STOPE TIMBERS.

the stope will deliver the broken ore to these chutes as stoping proceeds, with a minimum of shoveling. The square sets are so erected that the caps, *bb*, lie parallel to the gangway and the girts, *cc*, consequently at right angles to it. When the ore has all been extracted and the stope is ready for robbing, 6 by 6-in. braces, *dd*, are placed between the two diagonally opposite caps in the second and fourth rows of sets on the two floors above the sill, as shown in Section *AA*, Fig. 1. If the stope is taking weight badly a brace may be necessary at each end of each of these caps, but in some cases a single brace in the middle of the cap is sufficient. It is usually the case where robbing is attempted that the plans include the

robbing of successive sections which will each be mined in turn after the preceding one has been robbed and filled. If such is the case, gob lagging is spiked to the rows of posts 1 and 5, Section *AA*, so that when filling is introduced the gob will lie tightly against the ground or previously gobbed stope on that side of the section, in this case the left, where no further mining is anticipated. One run of sets on the other side of the section will be left open when the stope is filled, so that the timber as extracted may be stored in these sets and used when the adjacent section is carried up.

After the braces have been put in place the caps in row 2 are cut, as shown in the detail, Fig. 1, to permit of slipping out the girts between rows 2 and 3. When these have been removed it is a simple matter to remove the caps and the sill and first floor posts in rows 3 and 4 and the girts between rows 3 and 5. Waste filling is then introduced until it reaches to within a few inches of the second floor above the sill, the caps cut and the timber on this floor between rows 2 and 5 removed as before. The braces, partially buried in the gob, may be drawn out by throwing one end of an ordinary mule skinner's tail-chain over the exposed end of a brace and fastening the other to the end of a piece of timber. Using this piece of timber as a lever and the cap still in place above as a fulcrum the brace is easily loosened and drawn from the filling. During each downward movement of the lever the chain automatically loosens and slips lower on the brace, so that one man can easily manage the operation. The braces as removed are placed diagonally between the caps at the top and bottom of the third floor above the sill. These braces may obviously be used again and again until weakness or damage impairs their usefulness. The process as described with alternate filling and drawing timbers a floor at a time is repeated until the top of the section is reached and the stope filled. The timber as removed is stored in the open run of square sets for use in the adjacent section. In all cases the posts and girts on each end of the stope are left behind and if the ground is unusually heavy it is safer to leave an entire run of sets to support the ends of the stope as the timber in the center is being removed. These are left in the gob so that the ultimate timber recovery is much reduced. The braces, as the timbers below are removed, take up the vertical and part of the lateral pressure, but the major portion of the latter is taken up by the gob. It is essential, therefore, that the filling be kept well up to the level of the floor from which the timber is being removed.

If the ground is unusually heavy or the orebody over 50 ft. in thickness the section must be carried up in successive horizontal lifts in order that there may be a minimum of ground open at any one time before filling is introduced. It is the usual practice to open the section three sets high, brace, remove the timbers and fill as before the lower two floors. With the stope thus securely reinforced, the section may be carried two

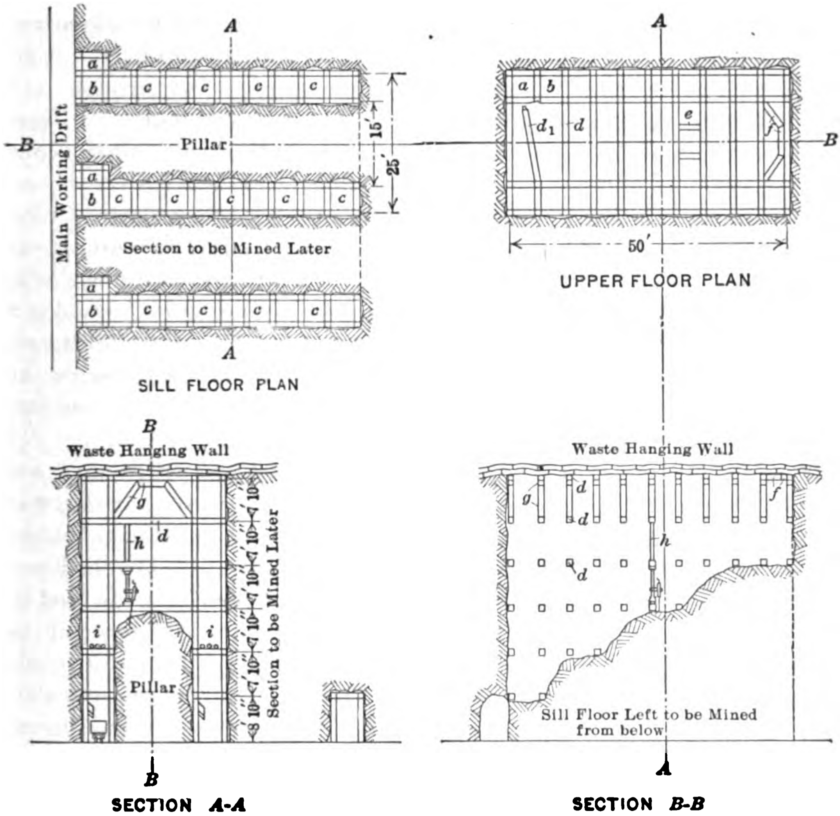
sets higher when the two lowest open floors are robbed and filled as before. If the ground is so heavy that additional precautions are necessary the timber is first drawn from only one end of the stope and filling is introduced into the robbed portion while the adjacent row of timber on the same floor is being extracted. Thus, as soon as the timber is removed, it is replaced by gob so that the walls of the stope are at all times amply supported. Using this method of mining and recovering the timbers in successive lifts, there is virtually no limit to the height to which a section may be carried safely in heavy ground.

The adjacent section and each subsequent one are carried only three sets wide, but the same length as before. The method of timbering and robbing successive sections is shown in Section *BB*, Fig. 1. In each case the stope is carried up alongside of the run of square sets left open in the previously mined section so that four sets of ground are actually open. The timber extracted from the preceding stope is used as far as it goes in the new section. That which is in the best condition is used where it can be recovered again and the partly damaged material where it will be left behind in the gob. When the robbing process begins in a new section the braces are placed in the run of sets left open in the previously mined section and the center run of the new one. The caps cut are those directly against the gob. In every case it is the caps to be left behind that are cut, leaving unmutilated all of the timber to be removed. As filling is introduced, gob lagging is spiked as shown to the outside of the center run of sets, leaving one run open as before for storing timber for the next section. It is evident from Section *BB*, Fig. 1, that in using this method in several successive stope sections only every third row of posts and caps is left behind in the gob while all the intermediate timbers are recovered. The secrets of success in using this method are the narrow stopes and care in keeping the filling well up to the floor from which the timber is being taken.

## Mitchell Slicing System

Mr. Mitchell is also responsible for the next improvement in stoping methods which has found wide application in the Calumet and Arizona mines during the past few years. This method, known as the Mitchell slicing system, has been described in an article by M. J. Elsing in the *Engineering and Mining Journal,* July 23, 1910, but as several changes and improvements have been made in it since that time it will bear further mention. It was arrived at more or less by accident. A large block of ore broke away from the back while a heavy sulphide stope was being worked and in order to recover it, long stringers were thrown across the top of the ore to support the back and the ore mined from above by underhand stoping. The method as finally developed from this simple beginning is applicable to orebodies in which the hanging wall is fairly

flat and regular and the lateral pressure not too great. It is flexible, however, by virtue of the fact that square sets may be used in conjunction with the method in mining irregular and outlying portions of the ore-bodies. Furthermore, there must be no large quantity of waste in the ore, for while it is possible to sort and leave some waste material behind in




*a* Manway
*b* Main Chute
*c* Auxiliary Chutes
*d* Stringer in Position
$d_1$ Stringer being Swung in Position
*e* Temporary Spread
*f* Diamond Brace
*g* Segment Set
*h* Temporary Brace
*i* Grizzly

FIG. 2.—MITCHELL SLICING METHOD.

the stope the method loses many of its advantages if careful sorting must be done.

After preliminary prospecting has roughly outlined the shape and dimensions of the oreshoot, two lead rows of sill square sets are driven, 20 ft. center to center, at right angles to the general strike of the body from the main working drift to the end of the section to be mined, as shown in the sill floor plan, Fig. 2. The length of the section varies from 25 to 50

ft., depending upon the character of the ground. Where it is possible to run cars directly into these two crosscuts a small sloping cut is made in every alternate set on the same side of each crosscut for a loading chute. Care must be taken to keep these cuts small and under no circumstances to take them on both sides of the same pillar, for if the base of the pillar is weakened trouble is sure to ensue. If it is not practicable to load the cars in these side crosscuts, as is often the case, the first set alone may serve as a chute and the cars be loaded directly in the main drift. A six-post raise is carried over the first two sets in each cross-cut through to the level above to be used as a timber and manway and later as a waste chute. On the sill floor a small manway, *a*, is cut out next to the chute set as shown, and on the floor above the ladderway is carried up over the chute while the other compartment of the raise is lined for a chute. When the raises have holed to the next level runs of square sets are carried directly over the lead rows of sill sets to the top of the ore, leaving a pillar of ore 15 ft. wide and the length of the section resting on its own base to be subsequently mined in horizontal slices from the top downward. Under normal conditions standard 10 by 10-in. square-set timbers are used in these preliminary operations, but if the ground is unusually heavy 12 by 12-in. timbers are advisable.

The first slice is taken, as shown in Sections *AA* and *BB*, Fig. 2, over the whole top of the pillar, cutting it loose from the waste hanging wall. Machines are first erected in the topmost square sets and holes pointed in such a manner that the ore passes with a minimum of shoveling onto grizzlies in the open run of square sets. Where only one chute is used the broken ore fills the sets until its normal angle of repose is reached, the remainder running from the chute by gravity. The 10 by 10-in. stringers, *dd*, 15 ft. long and framed with 2-in. tenons, are thrown across the open slice between the caps of the topmost run of square sets and directly under the hanging wall with such blocking and lagging as is necessary to keep the stope safe and the timbers secure. In placing a stringer in position the tenon on one end is inserted in the place between caps and post framed to receive it in the same manner as an ordinary girt. The opposite cap is cut 2 in. to allow the tenon on the other end of the stringer to be swung into position, as shown in the upper floor plan, Fig. 2, and a piece of lagging spiked to the cap to hold it in place. The topmost stringers are supported temporarily by stulls resting on ore in the pillar until enough ground has been broken to permit of laying similar stringers between the caps of the next lower run of square sets. When these stringers, resting on the unbroken ore in the pillar, have been placed in position the stulls are replaced by segment sets, *gg*, of 10 by 10-in. timber, erected, as shown in Section *AA*, Fig. 2, to support the upper stringers and the back of the stope. While 10 by 10-in. or 8 by 10-in. stringers are necessary directly under the hanging wall, it is the usual

practice to use 8 by 8-in. timber for those subsequently put in where the weight to be carried is much less. Segment sets are usually unnecessary except below the topmost stringers, although occasionally used below if the ground is very heavy.

The next slice is then taken across the top of the pillar. Either jackhamers or piston machines set up on columns between the stringers are used to drill down holes, pointed so that the ore is broken directly into the chutes. In subsequent slices, if a piston machine is used, it is usually necessary to put in temporary 6 by 6-in. braces, *hh*, between the stringers vertically above the drill column in order to make the machine secure while drilling. It is sometimes safer to put in temporary horizontal spreads, *ee*, between the stringers near the holes which are about to be blasted. If the ground at the end of the stope begins to swell or work 10 by 10-in. diamond braces, *ff*, are placed in position as shown in the upper floor plan, Fig. 2, to take up longitudinal pressure. The slices are taken to within one floor of the sill leaving one set of ore to be mined from below. A floor is then laid over the entire stope and 2-in. gob lagging is spiked vertically to the inside of both vertical runs of square sets, leaving the square sets open when the stope has been filled to be used as one set of chutes in each of the adjoining sections. If the ground is very heavy it has been found impracticable to leave these sets open, and in that case they are lagged on the outside and filled with the rest of the stope. The filling is introduced through the two raises from the level above and if the stope is not taking weight badly a considerable proportion of the stringers and segment sets, if the latter have been used, may be recovered as filling progresses. Upward of 50 per cent. of the stringers may be thus removed under favorable conditions. If it has been considered advisable to leave the square sets open the timber recovered may be stored in these sets and used when the adjoining section is taken. Where the timber cannot be used in a stope in the immediate vicinity it is usually better economy not to attempt to extract it, as much handling soon neutralizes any saving gained.

The maximum vertical height of a section which may be safely mined by this method has been found to be about 60 ft. When the ore extends from level to level, or 100 ft. thick, the stope is worked in two lifts of 50 ft. each, the upper lift first. In this case raises are driven through from level to level as before but the preliminary rows of square sets are driven 50 ft. above the sill and carried from there to the upper level. An attempt was made at first to carry these runs of sets the entire 100 ft., but it was found that by the time the upper lift was mined out and filled the timbers below were so badly crushed as to be useless. The upper lift is mined as has been described with the exception that the slices are taken for the entire 50 ft. before the floor is laid and filling commences. It is obvious, too, that all of the ore must perforce be drawn from only two

chutes instead of from the series of chutes as is optional in the method as originally devised.

Where waste material must be sorted from the ore and left behind, flooring may be laid on the stringers and a certain amount of waste piled on the floors. A recent adaptation of the method, which eliminates to a certain extent this makeshift device, depends on mining the pillars in benches as shown in Section *BB*, Fig. 2. The slices are taken out more rapidly in the end of the stope nearest the waste raises so that the lowest slice is mined out, the floor laid and filling brought in while mining is still progressing on the upper benches at the other end of the stope. When this stage is reached any waste encountered may be mixed with the filling in the worked-out end of the section. Mucking broken ore is further reduced by this improvement as the ore runs more readily into the chutes if the benches are properly taken.

In preparing to work a section adjoining a filled stope, a new lead row of sill sets is driven, as shown in the sill floor plan, Fig. 2, leaving a 15-ft. pillar as before, the vertical run of square sets carried to the hanging wall, and this run of sets used in conjunction with the one of the other stope which is still open, if it has been possible to so leave it, as chutes in working the new section. Thus but one run of square sets with the consequent narrow work is necessary for each section. If it has not been possible to leave the square sets open in the filled section it becomes necessary to take the slices at such an angle that the broken ore will run by gravity into the new run of sets. The bench method is particularly useful in this regard as it facilitates the gravity handling of the ore.

The saving in labor, timber, and powder by this method over the square-set stope is an obvious and substantial one, but the main advantage of the system is the rapidity with which the ore may be mined if necessary and the increased tonnage yield per man in the stope. While 5 or 6 tons per man is considered creditable in a square-set stope, it is not considered extraordinary in mining the pillars in the Mitchell system to reach three or four times those figures and in auger ground a yield as high as 50 tons per man per shift has been attained. The actual saving in stoping cost as shown in the appended table amounts to from 20 to 30 per cent. of that of square-set stoping.

## Mitchell Top-Slice Caving System

It was decided about two years ago that the top-slice caving method as used on the Minnesota iron range, in Cananea, and elsewhere, might be applied to a large body of oxide ore in which heavy and swelling ground made the cost of square-set stoping excessively high. The fact that there was no working level above this ore from which the stopes might be filled also contributed to the choice of the top-slice system. Although

the method as usually practised worked with some degree of success and reduced the cost of stoping substantially, Mr. Mitchell, who was in charge of the work, was not satisfied with the results, and from it he evolved an inclined top-slice caving method which reduces the large amount of handling of ore in the stope, incidental to ordinary top-slicing, to a minimum.

The method is applicable to the same conditions as those under which

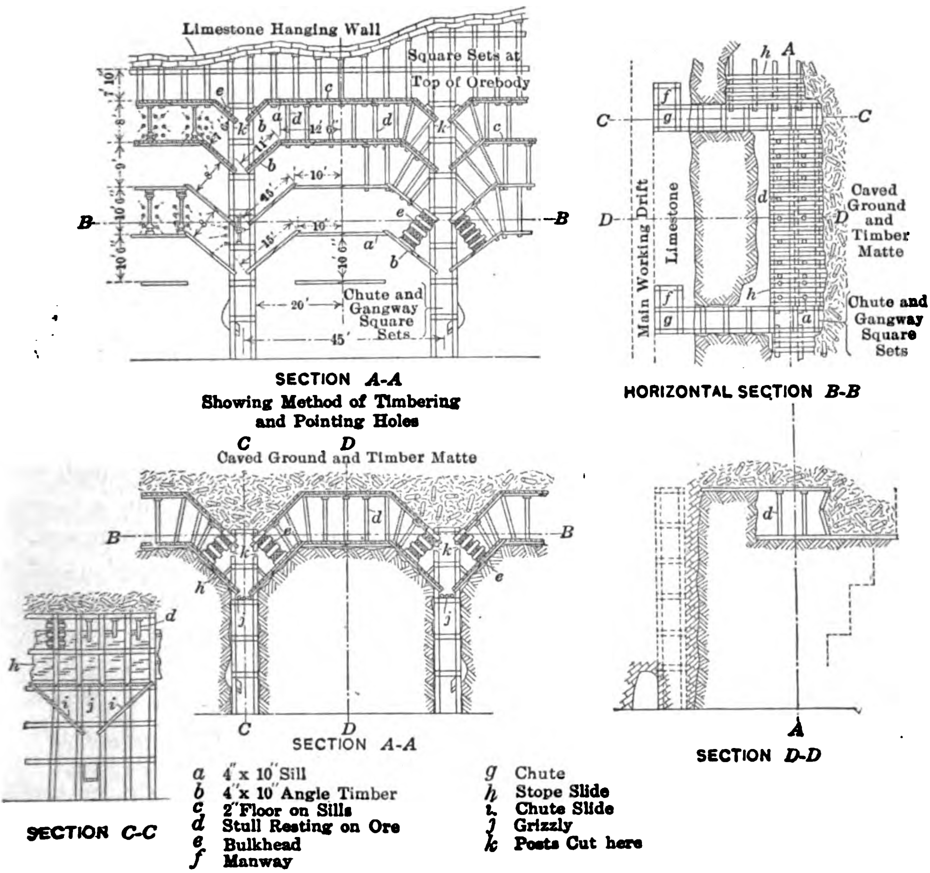



FIG. 3. MITCHELL TOP-SLICING CAVING SYSTEM.

ordinary top-slicing is successful, that is to say, uniform ground which is too heavy for economical square-set stoping and which will cave readily as required, but may be at all times kept under adequate control. It may often be used with increased economy in ground adapted to square-set stoping if the other conditions obtain. Before deciding upon this method there should be reasonable assurance that there is no ore left unextracted above the body to be caved, as the subsequent mining of this

ore would be attended by many difficulties after the country has been well broken up. In case development work is being done on a level above that of the stopes, this method, unlike square-setting, affords no means of disposing of the development waste, which consequently must be trammed to the shaft and hoisted. In common with ordinary top-slicing, ventilation is poor in the stopes worked by this system. There is sure to be some ore sacrificed in the process even under the most satisfactory conditions, which is an important consideration in the Warren District where a comparatively high-grade ore is mined.

In preparing for this method, the orebody is laid out in 45-ft. sections at right angles to its general strike. These sections may be mined the width of the ore to a maximum of about 80 ft. and 100 ft. is the economic limit of the height to which each section should be carried. Permanent six-post raises are driven, as shown in Sections *AA*, *BB* and *DD*, Fig. 3, preferably in waste at the edge of the ore, to the elevation of the top of the orebody. One compartment of each is used as a chute, *g*, the other as a timber and manway, *f*. From the chute compartment, parallel lead rows of square sets are carried the width of the ore, or of the section to be taken, on the three floors directly beneath the waste hanging wall. On each of the floors below these the square sets are carried a distance decreasing as the sill is approached leaving a foot wall of unbroken ground sloping in the direction of the main drift on which the broken ore will run directly into the chutes. This leaves a pillar 40 ft. wide which is mined by horizontal slices from the top downward, one-half from either side simultaneously, and the back allowed to cave as stoping progresses.

Before taking the first slice, the ground on either side of the top row of square sets is broken by holes drilled from a set-up in the square sets in such a way that 4 by 10-in. angle timbers, *bb*, 7 ft. 6 in. long, may be rested on solid ground at an angle of about 45°, as shown in Section *AA*, Fig. 3; 2-in. flooring is laid on these angles, making a slide, *h*, upon which the subsequently broken ore will run by gravity upon grizzlies, *jj*, in the square sets and thence into the chutes. The first cut is taken from both sides simultaneously across the pillar at the end of the ore or section farthest from the main drift and is timbered with standard square sets. The slice then proceeds from this cut toward the main drift, working from both sides of the section at once and supporting the waste back with square sets; 4 by 10-in. sills, *aa*, cut to lengths and with bevels as shown in Section *AA*, Fig. 3, are laid at the base of the posts, furnishing a support for the upper end of the angle timbers. Beneath the beveled joints of these sills pieces of 2-in. plank are spiked to render them rigid and to act as headboards for the stulls, *dd*, placed under them when the next slice is taken; 2-in. flooring is laid on top of these sills. As the slice retreats toward the main drift the rows of square-set posts first erected are shot out and the back caved, leaving at all times at least two open

sets between the working face and the caved ground. To prevent the vertical run of square sets from being crushed by the weight of the caved ground, small bulkheads, *ee*, are built under the lower caps of this floor resting upon the angle timbers. The posts are then cut at points, *kk*, directly under the angles, thus transferring the weight from the square sets below to the solid ground in the pillar.

When the upper slice has proceeded far enough from the extremity of the section so that the ground first caved is solid, preparations may be begun for the second slice. A machine is set up in the open run of square sets and ground broken so that 4 by 10-in. angle timbers 11 ft. long may be laid as before at a slope of about 45°. The angle timbers and sills above are caught up as shown with round stulls resting on the unbroken ore in the pillar, 6 to 8 in. in diameter as the ground may require. The preliminary cut is carried as before about 5 ft. wide and 9 ft. high across one-half the width of the section, or 20 ft., erecting stulls under the sills beneath the caved ground and timber matte above; 4 by 10-in. sills are so laid that they do not come vertically below the upper sills, for the vertical stulls supporting the upper sills should rest directly on solid ground. These sills are covered with 2-in. flooring as before. The erection of drill columns and the pointing of holes is shown in Section *AA*, Fig. 3. The breast holes shown are usually unnecessary, as the cut holes and lifters are in almost all cases sufficient to break the ground. After the preliminary cut has been taken the major portion of the ore is slabbed directly into the chute sets, leaving from one-third to one-fifth to be mucked. Stulls should be just strong enough so that the third stull back from the working face is commencing to crush as the face is being drilled, as shown in Section *DD*, Fig. 3.

As slicing proceeds downward the lower floors of the vertical runs of sets must be extended to maintain the slope of the solid ground to the extreme end of the section and permit the broken ore to run readily into the chutes. Care must be taken not to take out the vertical runs of sets too far ahead of operations, for the weight on them is considerable even if the greatest precautions are taken. If the orebody is over 80 or 100 ft. in width and consequently too wide to be mined in one section at right angles to its strike, it is not advisable to attempt to leave a vertical wall of ore against the caved ground and timber matte at the end of the section. In this case each new slice is started about 5 ft. closer to the working raises than the slice above, as shown in broken lines in Section *DD*, Fig. 3. Thus the end of the stope is left in steps and the danger of losing ore which may break from the wall and mix with the matte is eliminated. The ore so left may be recovered readily when the next section is taken. As mining approaches the sill floor, chutes may be put in the sill sets, as shown in Section *AA*, Fig. 3, at such intervals as will facilitate the rapid and efficient drawing of the broken ore. Bulkheads, with subsequent

cutting of posts to take the weight from the vertical run of sets, need not be erected oftener than conditions require. In normal operations bulkheads in every third slice afford sufficient protection to the square sets below. Enough partly crushed timber comes through from the timber matte above to build these bulkheads so that their cost is insignificant. It has been found that the maximum thickness of slice which can be safely carried under the conditions encountered in the Calumet and Arizona mines is 10 ft. 6 in. and slices of this thickness are not attempted until the fourth slice is taken and the caving matte is under perfect control. If waste is encountered in the ore it is quite feasible to sort it out and leave it behind to become incorporated with the caved ground and timber matte, but like the method last described the system loses many of its advantages if careful sorting must be done.

As shown in the appended table, a 10 per cent. reduction in stoping cost is gained by this method over ordinary top-slicing. The labor cost is cut about 15 per cent. and the item of powder is substantially reduced. The ore may be mined rapidly and safely in the heaviest kind of ground. An output of about 10 tons per man per shift or a total of about 125 tons per day can be maintained from each section.

## Gilman Cut-and-Fill System

When the large bodies of primary sulphides of great vertical as well as horizontal extent, from which about one-half of the total tonnage of the Calumet and Arizona mines is now being derived, were first opened up they were mined by ordinary shrinkage stopes of great size, the dimensions being determined solely by the limits of the orebody. When the stope was finished the shrinkage ore was drawn out and waste filling introduced. While confined to virgin ground this method was quite safe and satisfactory as the country rock was strong massive limestone and the walls sound. As more of the country was opened, however, these large open holes became dangerous and the method evolved itself into a combination of shrinkage and cut and fill with virtually no limit to the width of the stope or the amount of open ground above the gob. But increasing disturbance of the country rock as mining proceeded recently made it necessary to develop some safer method of mining these great sulphide lenses without too great a sacrifice of the very low costs and flexibility of the systems previously used. Consequently a cut-and-fill method was devised by Oscar Gilman, foreman of one division of the Calumet and Arizona mines, which has remedied the existing evils of the older systems and still permits of a minimum of timber, low costs, great flexibility and large daily tonnage. The method may be used wherever the ore and country rock are strong and solid and, while the width of each stope section taken is limited to 40 ft., the length and height of stopes are

limited only by the extent of the commercial ore, although for convenience sections are usually included between levels 100 ft. apart vertically.

Crosscuts are driven at 40-ft. intervals through the orebody from a strike drift, preferably in country rock in one wall of the orebody, at right angles to its general strike. A cutting-out stope is started, as shown in Section *BB*, Fig. 4, by enlarging the crosscut with stoping drills. When it has reached a height of about 8 or 9 ft. above the rail, hitches are cut in the walls and temporary so-called "Cousin Jack" stringers, *gg*, about 10 ft. long are thrown across the crosscut at 5-ft. intervals 5 ft. above the

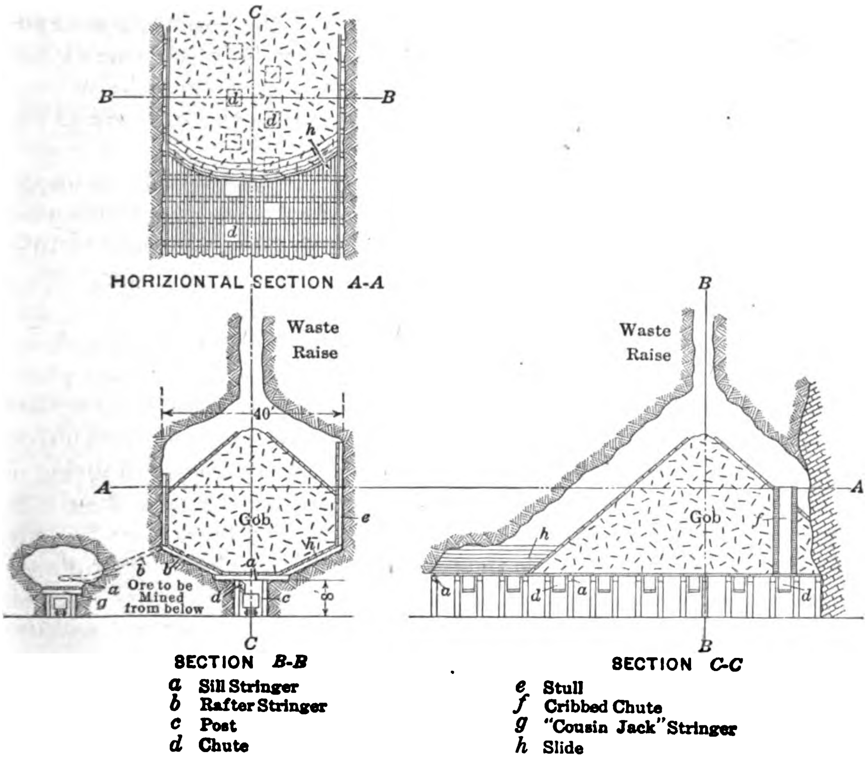



FIG. 4.—GILMAN CUT-AND-FILL SYSTEM.

rail; 4 by 10-in. timbers are used at each side of the crosscut to support these stringers where the ground requires them. Flooring is laid upon the stringers in a direction parallel to the crosscut so that ore subsequently broken may be loaded directly into cars run into the crosscut through the opening made by removing one of the center planks. This device is used only during the early stages of the operation to eliminate mucking from the floor of the crosscut before the stope has reached proportions large enough to permit of the erection of permanent sill timbering and chutes. The cutting-out stope is then enlarged by drilling deep holes

with stoping drills, maintaining an arched back and a slope of about 20° on the two sides of the crosscut to a width of 40 ft. Hitches are cut in the sides of the stope to receive 16-ft. by 12-in. by 12-in. stringers, *aa*, which are laid at 5-ft. intervals for the length of the section 8 ft. in the clear above the rail. The "Cousin Jack" stringers are then removed and if the ground requires it the 16-ft. stringers are supported by 10 by 10-in. posts, *cc*, 5 ft. in the clear. This leaves ample room for the 42-cu. ft. motor cars into which the ore is loaded in these crosscuts. As a rule, no braces are necessary between stringers but where required 4 by 10-in. timbers are used. Then 18-ft. by 10-in. by 10-in. rafter stringers, *bb*, beveled as shown in Section *BB*, Fig. 4, are laid on the sloping sides of the stope with one end resting on each 16-ft. stringer; 4-in. flooring is laid over the whole and chutes, *dd*, built on both sides of the drift in each alternate set. A 10 by 10-in. brace is used between the stringers at the side of the crosscut where the chute is to be built.

While this preliminary preparation has been going on, at about the center of each section a raise has been started which is ultimately connected with the level above to be used for timber and waste filling. If waste is encountered in driving this raise it is discontinued until filling is needed in the stope. To provide for absolute safety no more than 10 ft. of open ground is left at any one time between the floor and the back of the stope. The initial enlargement of the stope takes place, as shown in Sections *BB* and *CC*, Fig. 4, beneath the waste raise, and when the 10-ft. limit has been reached, waste is admitted and forms a cone extending out toward the edges of the stope. Temporary flooring is laid upon this gob at its normal angle of repose leaving only 2 or 3 ft. between the floor and the back of the stope. The back may be supported in any weak spots while the next slice is being taken by temporary stulls or bulkheads resting upon the waste. As the chutes under this gob pile are covered over they are abandoned and the ore as broken runs upon the sloping floor and is drawn from the chutes still open at either end of the waste.

As mining exposes the sides of the stope, 8 by 10-in. stulls, *ee*, from 8 to 16 ft. in length are placed vertically at 5-ft. intervals, resting on the rafter stringers. Where one stull is placed upon another as the stope progresses upward they are securely lashed together, on four sides if possible, and each tied into the gob with timber "dummies." It is advisable to use as long stulls as practicable for every joint is a point of weakness when the next section is taken; 2-in. gob lagging is spiked horizontally to these vertical stulls before filling is admitted. In the actual process of mining it is often found that a few long holes judiciously placed toward the sides of the stope will when blasted break the arch and break a much greater tonnage than would normally be expected. When the limits of the orebody approach the vertical it becomes neces-

sary to carry up a chute or two, *ff*, next the waste wall with 4 by 10-in. cribbing through the waste filling, to be used as ore chutes. The length of the stope often becomes so great as mining progresses upward that one raise will not serve for filling the whole stope. In this case another raise may be driven through to the level above, or the outlying portions of the stope filled by driving small inclined raises off into the waste walls. It is apparent, however, that under normal conditions waste raises may be a long distance apart.

*Comparative Stoping Costs*[1]

| Method and Conditions | Labor | Explosives | Timber | Candles or Carbide | Air | Other Supplies | Total |
|---|---|---|---|---|---|---|---|
| Square-set, oxide ore,[2] heavy ground | $0.73 | $0.06 | $0.34 | $0.01 | $0.12 | $0.04 | $1.30 |
| Square-set, oxide ore, average ground | 0.60 | 0.05 | 0.25 | 0.01 | 0.10 | 0.04 | 1.05 |
| Square-set, oxide ore, robbing timbers | 0.61 | 0.05 | 0.16 | 0.01 | 0.10 | 0.04 | 0.97 |
| Top-slice caving, old method, oxide ore, heavy ground | 0.63 | 0.07 | 0.20 | 0.01 | 0.10 | 0.03 | 1.04 |
| Mitchell top-slice caving, oxide ore, heavy ground | 0.54 | 0.04 | 0.21 | 0.01 | 0.10 | 0.03 | 0.93 |
| Mitchell slicing, oxide ore, average ground | 0.51 | 0.05 | 0.20 | 0.01 | 0.06 | 0.02 | 0.85 |
| Square-set, sulphide ore,[3] average ground | 0.49 | 0.04 | 0.19 | 0.01 | 0.04 | 0.03 | 0.80 |
| Mitchell slicing, sulphide ore, average ground | 0.43 | 0.03 | 0.15 | 0.01 | 0.03 | 0.01 | 0.66 |
| Cut-and-fill, Gilman method, sulphide ore, good ground | 0.34 | 0.04 | 0.07 | 0.01 | 0.04 | 0.01 | 0.51 |
| Cut-and-fill, old method, sulphide ore, good ground | 0.32 | 0.04 | 0.04 | 0.01 | 0.04 | 0.01 | 0.46 |

[1] Figured on the basis of the wet tons mined. These costs obtain under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers.

[2] Oxide ore in place will average from 12 to 16 cu. ft. to the ton.

[3] Sulphide ore will average from 9 to 10 cu. ft. to the ton.

If there is a great demand for ore, stoping may be commenced in the adjoining section while still mining in the original section. If this is contemplated it is better practice to drive the original waste raise nearer one end of the section so that there may be a considerable thickness of gob at that end while stoping is still in progress in the lower portion of the other end. Stoping in the new section is carried on in identically the same manner as before. The new waste raise is driven near the same end of the section as in the adjoining partially filled stope so that mining in the new section proceeds most rapidly adjacent to that part of the original stope in which the gob is thickest and most firmly consolidated. In mining alongside of this old filling care must be taken not to expose too

much of it at one time, and the stull joints must be watched. But if proper precautions are taken no difficulties are experienced in mining two or more parallel stopes simultaneously. It is inadvisable to leave a section with the idea of mining it after the stopes on either side have been finished and filled. Carrying a stope by this method between two gobbed sections is at best an uncertain operation and the risk may be entirely eliminated by taking the sections in succession as described. The ridge of ore 35 ft. wide and 14 ft. high between each stope on the sill floor is left to be mined from below. While as yet no stopes have been carried up to this point, no difficulty is anticipated as the two rafter stringers braced against each other will be virtually self-supporting and in any event it will prove a simple matter to catch them with stulls from below.

Phenomenally low costs for the Warren District have been obtained with this system of stoping. It will average about $0.50 per ton and a much lower figure has been reached under especially favorable conditions. Although slightly higher than the cost of the old shrinkage method the added consideration of safety more than counterbalances the small increase over the more hazardous system. The stope ventilation is good and the efficiency of the men high. An average yield of about 12 tons per man per shift can be maintained under normal working conditions. The daily tonnage output of each section will average 100 tons and may be crowded to 150 or even 200 tons per day.

## Summary

In comparing the various methods described, the considerations enumerated in the early part of the paper must be applied to each system. The cut-and-fill system is easily the cheapest and most satisfactory in every way where conditions will permit of its use. The item of timber, usually a large one, is much reduced and the risk of fire is virtually eliminated. While any large horse of waste encountered in the ore may be kept separate and left in the gob, any attempt at close sorting is inadvisable. As in the case of the Mitchell slicing system, if the ore is fairly clean the greatest ultimate economy will be gained by mining it all as it comes without attempting to sort. The consequent low stoping cost obtained by mining large tonnages rapidly will more than counterbalance the increased cost of handling and treating a slightly lower grade material. In neither system can the grade of ore be controlled as closely as in the square-set or top-slice caving methods. Where the ore is very irregular or badly mixed with waste the square-set method is still found to be the most satisfactory. If the stope is in good condition a considerable proportion of the timbers may be safely extracted as it is filled.

For a heavy uniform orebody the Mitchell inclined top-slice caving system is a good one. While it still retains some of the disadvantages of

the old-fashioned top slice it is in many ways a great improvement over the older method. Its principal advantages are the substantial saving in labor and powder, and the greater rapidity with which the ore may be mined. Its unfavorable features are those of all top-slice caving methods, the almost inevitable sacrifice of some ore, narrow work, poor ventilation in the stopes and the fact that the ground above is so badly broken as to render the expense of future work on upper levels so great as to be in many cases prohibitive. The Mitchell slicing system finds its chief application to fairly regular bodies in not too heavy ground. Under favorable conditions lower costs are obtained with this method than by any of the others in use, with the exception of the cut-and-fill method. Its saving in labor and timber is a very appreciable one over square-set stoping and the cycle of preparing, mining, filling and abandoning a section is normally a very short one. This feature, in addition to permitting a large daily output, reduces considerably the repair cost incidental to keeping a section of the country open over a long period. In short, the introduction of these methods has effected a tremendous saving to the Calumet & Arizona Mining Co. during the past few years without the slightest sacrifice of the safety of the men or of the mines in which they have been used.

In closing the author wishes to acknowledge his indebtedness to M. W. Mitchell and Oscar Gilman, mine division foremen of this company, for invaluable assistance in the accumulation of data for this paper.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

**DISCUSSION OF THIS PAPER IS INVITED.** It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Flotation of Minerals

BY ROBERT J. ANDERSON,* B. S., ROLLA, MO.

(Arizona Meeting, September, 1916)

## INTRODUCTION

DURING the past 5 years no subject has aroused more interest or received more attention among mill operators than flotation. One reason for this is, undoubtedly, the remarkable success of the process in Australia where that country has risen to the position of furnishing one-fifth of the world's supply of zinc. The plant of the Butte & Superior Copper Co., designed by James M. Hyde in 1912, was the first important large flotation plant in this country; other installations have been made since then with considerable frequency, particularly so in the last 2 years.

Without going into complete detail, it may be safely said that although flotation is a highly successful commercial process, it is more or less on an empirical basis. Upon scanning the literature, it is found that the flotation investigations have, in the main, dealt with a solution of the problems which accrue to practise; until recently no attempt was made to remove the difficulties in the way of the formulation of a consistent and harmonious theory. Of late this phase of the work has been receiving considerable attention, particularly by the United States Bureau of Mines at its Salt Lake City station, by the Mellon Institute at Pittsburgh, by the General Engineering Co. at Salt Lake City, by many of the mining schools, and private individuals almost *ad infinitum*. There is, at present, an enormous amount of laboratory work being carried on in flotation research, and probably the time is not far away when a generally acceptable explanation of flotation phenomena can be set forth. However, at this time, it seems as though the current technology is befogged with a superabundance of contradictory evidence; this confusion can only be dispelled as time proceeds and the knowledge of the subject becomes more complete.

Many phenomena are supposed to contribute to the flotation of minerals, whether in whole or in part is a mooted question. I shall only sketch roughly the present tendency of ideas and make no reference to the first early and crude notions which have little value other than histori-

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cal interest. Many questions have arisen in connection with flotation concerning such matters as surface and interfacial tensions, adsorption, absorption or occlusion, colloids, emulsions, electrolytic and electrostatic phenomena, etc. These phenomena are to be discussed in later paragraphs in order to ascertain what bearing they may have, if any, on froth flotation. No final solution of this very large problem is attempted in this paper, for in any discussion of flotation theory, at this time, one is confronted by so many obstacles that at best only a rather fragmentary presentation can be made. An attempt has been made in this writing to correlate the operative and fundamental principles and compare in a measure some of the theories so far advanced.

## Discussion of Certain Factors

### *Surface and Interfacial Tension*

Surface tension has been rather well defined in articles appearing in the *Journal of the American Chemical Society* during the years from 1908 to 1913. The theory of surface tension has been treated in particular by Laplace, Gaus, and more recently by Van der Waals, and by Willows and Hatschek.[1] As defined by Jones,[2] "potential energy, present at the surface of liquids, produces a tension which is known as surface tension." The phenomena which are invariably indicative of surface tension are commonly observed: Drops of a liquid not exposed to an external force, *i.e.*, either suspended in another liquid of the same specific gravity or freely falling, assume a spherical shape, the sphere being that form of body with the smallest surface per given volume; further, if water be placed in an open vessel its surface film will be a measurable quantity, and its thickness will vary with a number of factors of which temperature is one. Its thickness is observed as ranging from $4 \times 10^{-5}$ cm. to $4 \times 10^{-8}$ cm., and its density, when referred to the main bulk of the water below, will approximate 2.14. Surface tension is not affected by the surface area. It is numerical in value and expressed in dynes per centimeter. It is a variable factor dependent on temperature, increasing numerically with falling temperature, *e.g.*, water at 18° C. has a surface tension of 73 dynes per centimeter, and at 0° C. 75 dynes per centimeter. At the critical temperature of a liquid its surface tension becomes nil.

All liquids have a definite cohesion or tensile strength, which is ascribed to the well-known mutual attraction of their molecules. This then is comparable to a pressure existing within a liquid, which has been termed the "intrinsic" pressure. Naturally the value of the surface tension of solids is numerically a high one. The surface tension of a pure liquid

[1] Willows and Hatschek: *Surface Tension and Surface Energy*, 1915.

[2] Jones: *Elements of Physical Chemistry*, 1907.

against its vapor can be and is markedly affected by the addition of soluble contaminants. Some salts will raise the surface tension of water while others will lower it; the fact that the salts of weak acids will lower the surface tension of water is explained by the fact that free acid is liberated by hydrolysis. It is further known that all acids will lower the surface tension of water. The surface tension of water is decreased by the addition of oil, or better, oil will reduce the interfacial tension between the water-air phases. A phenomenon for which no explanation has been given is the one which shows that the addition of contaminants may either raise or lower the surface tension of water, but such addition, while it may decrease that tension greatly, can increase it only slightly. Any lowering of surface tension is more marked in a liquid which has a high surface tension, *e.g.*, water, than in liquids of low surface tension.

There can be, of course, no surface tension without adsorption, which produces, in the case of positive adsorption, an increased surface concentration resulting from a lowering of the surface tension by the contaminating and dissolved substance whatever it may be. The equation of Gibbs—$u = -c/Rt \cdot d\sigma/dc$—gives the relationship between surface tension and the distribution of the solute between the bulk of the liquid and the film interface. Here the notation is:

$u$ = excess of substance in the surface layer,
$c$ = concentration in the main body of the liquid,
$R$ = the gas constant,
$t$ = absolute temperature,
$\sigma$ = surface tension.

This shows that when the surface tension is reduced by the addition of a contaminant, the quantity $d\sigma/dc$ is negative and $u$ is positive (from algebraic consideration). The surface film then contains more of the contaminant than the main body of the solution. If the surface film contains less of the contaminant than the main body of the solution it is a case of negative adsorption.

As given in the foregoing, the surface of a liquid against its vapor is in tension—surface tension; the surface of liquid against another liquid, or a gas or solid, is also in a state of tension: this is termed interfacial tension. In the consideration of the bearing which surface and interfacial tensions have on froth flotation, this condition of affairs obtains in the flotation machine: Pulp consisting of ore of approximately 80-mesh, water in ratio of 3:1 of ore, and oil in disappearingly small amount, is being violently agitated. For the sake of a specific case, the air is being forced mechanically into the swirling pulp by beaters or stirrers. These phases then are present in flotation by the oil-froth process, viz., solid-liquid (ore-water), solid-liquid (ore-oil), solid-gas (ore-air), liquid-liquid (water-oil), liquid-gas (water-air), and liquid-gas (oil-air). Thus six ten-

sions are present, but if the oil is soluble in the water the tension number becomes three. It is known that pure water can not be made to maintain a persistent froth because its surface tension is too high. Acid, if present, will lower the surface tension of water, as will oil if it is soluble to any extent. Very many interesting and important, as well as speculative, consequences follow from a consideration of interfacial and surface tensions, and, in a later paragraph, we shall have occasion to return to the consideration of these complicated phenomena.

Certain metallic sulphides, *e.g.*, galena, have the power of floating on undisturbed water; they are not wetted and the curve of contact is convex. Some gangue minerals, *e.g.*, quartz, possess an adhesive force of attraction for water which exceeds the intrinsic pressure of the water; they are therefore wetted and sink to the bottom, being drawn through the surface film. Such properties of the minerals are affected by the presence of oil, acid, and other reagents. Oil has a greater adhesive attraction for sulphide minerals than for gangue minerals; and the addition of acid and oil (if it is soluble) acts as a contaminant which will lower the surface tension of the water and aid in the production of a persistent froth. Let us now look into the question of adsorption and see what part it plays in flotation, since it is so requisite to the production of a variable surface tension.

*Adsorption*

Generally speaking, adsorption deals with the unequal distribution of substances at the interface between dissimilar phases such as, solid-solid, solid-liquid, solid-gas, liquid-liquid, liquid-gas, and gas-gas. It is purely a physical effect. Commonly, adsorption[3] is construed to be the result of the condensation of a disperse phase upon the interfacial boundary solid-liquid. Returning for a moment to the Gibbs equation mentioned above, adsorption may occur if the interfacial tension solid-liquid is reduced—this being positive adsorption. If, however, such an interfacial tension is raised in value it is a case of negative adsorption as the solute or disperse phase will be rejected from the surface. Any condensation, strictly stated, of a solute or disperse phase in the interfacial boundary separating liquid-liquid or liquid-vapor is held to be a special case of adsorption. However, in the general sense, the phenomenon is looked upon as being the result of condensation of a disperse phase in the interface of two immiscible phases. Adsorption is shown very strikingly by colloid gels—the product obtained by the coagulation of sols—and certain cases of selective adsorption are very remarkable. Adsorption will naturally vary with the surface exposed. In Miss Benson's experiments with amyl alcohol in aqueous solution, amyl alcohol reduced the surface ten-

[3] Briggs: *Journal of Physical Chemistry*, vol. 19, No. 3, p. 210 (March, 1915).

sion of the water, and it was found by producing a voluminous froth that the alcoholic concentration in the froth exceeded that in the bulk of the aqueous solution by about 5 per cent. A froth has a very large surface, and it would be expected that the adsorption would be greater. Such experiments prove the value, qualitatively, of the Gibbs rule.

Recent work shows that all solids do condense certain amounts of gases on their surfaces and retain them there with very great tenacity. Liquids in like manner adsorb gases. Further, liquids and solids exhibit marked selective adsorption of gases. Although this selective adsorption obtains, no evidential proof has been submitted which says that the amount of gas adsorbed by one substance is largely different than the amount adsorbed by another substance. An electric charge on an adsorbed substance probably would considerably influence the amount adsorbed. The adsorption of air plays an important rôle in flotation, for as Breuer points out, the adsorbed air film is enormously responsible in preventing the coalescence of solid particles.

A comprehensive study of the adhesion of small particles of solid to the dineric interface (surface separating two liquid phases) has been made by Hofmann[4] based on the theory of Des Coudres.[5] From the standpoint of flotation this may be given as follows: If a solid particle, *e.g.*, quartz, is wetted much more strongly by water than by another liquid, *e.g.*, oil, the water will displace the oil, and a film of water will form about the quartz particle according to the relative forces of adhesion. Then the quartz particles will remain in the water phase if the water has a specific gravity greater than the oil, regardless of their size; but if now the oil has a greater specific gravity than the water, then the quartz particles will remain in the water phase until the size of the particles is such that the force of gravity will remove them from the water. Conversely, if a solid particle, *e.g.*, galena, is wetted more strongly by oil than by water, the oil will form a surface film about the particle and hence prohibit the particle from being wetted by water, *i.e.*, from entering the water phase. Then the galena will only enter the water phase when the water is more dense than the oil, and, further, when the galena particles are of such a size that the force of gravity overcomes the adhesion of the oil film to the oil.

Returning to purely theoretical considerations, Hofmann draws certain conclusions at this juncture which deal with the supposition that solid particles will then remain in the surface separating two immiscible liquids, if those particles are wet partially by each liquid. I quote Bancroft at length on this matter:[6] "The solid particles tend to go into the water phase if they adsorb water to the practical exclusion of the other

[4] *Zeit. Phys. Chem.*, vol. 83, p. 385, 1913.

[5] *Arch. Entwicklungsmechanik*, vol. 7, p. 325, 1898.

[6] Bancroft: *Journal of Physical Chemistry*, vol. 19, No. 4, p. 287 (April, 1915).

liquid; they tend to go into the other liquid phase if they tend to adsorb the other liquid to the practical exclusion of the water; while the particles tend to go into the dineric interface in case the adsorption of the two liquids is sufficiently intense to increase the miscibility of the two liquids very considerably at the surface between solid and liquid."

Any simultaneous adsorption of two immiscible liquids by a solid would probably tend to result in the formation of a homogeneous liquid phase at the surface of the solid.

In regard to the effect of contaminants or other impurities in contact with two immiscible liquids, this condition obtains: If the contaminant is soluble in one liquid but not in the other, and also lowers the interfacial tension of the two, the equation set forth by Gibbs exacts the requirement that the contaminant should obtain in the interface. Examples of this prove the validity of the law.

The terms adsorption and absorption have been used interchangeably in some writings, thus contributing to the already existing confusion of ideas.

*Absorption or Occlusion*

There are three ways by which gases can be held with reference to solids: viz., (1) By surface adsorption; (2) in solid solution; and, (3) by occlusion. The term "occlusion" has been applied indiscriminately to any of the above methods by which gases are held by solids. Strictly speaking, by occluded gas is meant gas which is absorbed and held in finely divided pores or openings which may be of microscopic size. A recent theory[7] holds that occlusion plays the operative rôle in the flotation of minerals by all processes. I am unable to reconcile myself to this explanation for a number of reasons. Marked instances of occlusion at normal temperature are known only in certain amorphous substances like charcoal. Many metals, of course, both in the liquid and solid states, have the power of occluding gases, often in marked degree. There may be and undoubtedly are fine pores in the floatable minerals, which may in a sense be considered as an assemblage of capillary tubes; these can and do occlude gas. Yet occlusion is marked only in amorphous substances and in certain metals as just stated. It is definitely known that occluded gases are retained with very great tenacity by the substances occluding them and therefore expelled only with difficulty. It seems anomalous then to hold that the occluded gas can depart from the mineral occluding it with sufficient speed to aid the air bubbles in the liquid in the process of flotation. I believe rather firmly that occlusion is not a cogent factor in flotation by any process, and that a more con-

[7] Durell: *Mining and Scientific Press*, vol. 111, No. 12, p. 428 (Sept. 18, 1915) and Durell: *Metallurgical and Chemical Engineering*, vol. 14., No. 5, p. 251 (March 1, 1916).

sistent theory may be formulated without postulating these conjectures regarding occlusion.

### *Colloids*

Colloids, in the original definition of the term by Thomas Graham, are not a definite class of substances; rather a large number of different substances may be made to assume the colloidal state if proper precautions are taken. All of which reveals the striking fact that this colloidal condition is a *state* and not a *form* of matter. The ultra-microscope of R. Zsigmondy and H. Siedentopf has increased the knowledge of colloids to a great extent. A rather general statement may be made regarding colloids and that is, that they do not show osmotic pressure in any appreciable amount. Colloidal solutions—sols—are regarded as systems of two phases, in which the dissolved substance is the disperse phase and the solvent the continuous phase.

Since in flotation the ore is often as small in size as certain of the colloids, the flotation pulp (ore, water, etc.) can be looked upon as a coarse suspension, and the laws of colloids apply here with equal force as in the realm of colloidal chemistry. So-called suspensions are systems consisting of solid particles of microscopic size distributed through a liquid. As mentioned by Ralston,[8] Reinders has treated at length the particular case of a solid phase maintained in contact with two liquid phases, *i.e.*, two immiscible liquids. His work is based on the different interfacial tensions existing, and his experiments and those of Hofmann, as mentioned in an earlier paragraph, have considerable bearing on the flotation problem.

### *Emulsions*

Emulsions are fairly coarse dispersions of one liquid in another with which it is immiscible. The simplest and commonest emulsions are the oil-water emulsions, *i.e.*, the pure oil-water emulsions, containing no emulsifying agent such as soap, proteids, etc. In such systems the oil globules can be coagulated by electrolytes, show the Brownian movement strikingly, and can even be retained by some filtering media. Any process of emulsification is dependent on a surface tension lowering, or, to be more precise, on a lowering of the interfacial tension between the two phases. According to Briggs and Schmidt,[9] the two essential requirements of an emulsifying agent are these: (1) The property of condensing by adsorption in the dineric interface; and (2), the ability to form under these circumstances a strong coherent film. Temperature is a cogent factor in emulsification, for its effect is to reduce the interfacial

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[8] Ralston: *Mining and Scientific Press*, vol. 111, No. 17, p. 623 (Oct. 23, 1915).

[9] Briggs and Schmidt: *Journal of Physical Chemistry*, vol. 19, No. 6, p. 479 (June, 1915).

tension between phases and also to lower the viscosity of the phases. In the production of emulsions, a considerable amount of surface energy is produced because of the relatively large surface area of the disperse phase; an emulsion is the more speedily effected if such surface energy be reduced by the use of a liquid having a low surface tension as the continuous phase. Some emulsions, under certain conditions, display a great increase in viscosity over that of either of the immiscible phases, *e.g.*, the emulsions of the Pickering order—up to 99 per cent. of oil in 1 per cent. of soap solution—can be cut into cubes. Any emulsion produced with soap solution is at once rendered *nil* by the addition of acid, as the latter will decompose the soap.

If solid particles are suspended in a liquid, they tend to increase the viscosity of that liquid only gradually depending on the amount of solid particles present. Experiments have shown that whenever a substance which is in suspension is wetted by two immiscible liquids simultaneously, it will go into the dineric interface in the manner already referred to, and will therefore tend to produce an emulsion. If, however, the suspended particles can not coalesce due to adsorbed oil film or for other reasons and thus effect the production of a coherent film, the emulsion will not be stable. Very little data are available on the production of emulsions by the oils used in flotation work, or on the matter of interfacial tensions between such oils and water. However, we are no doubt dealing with emulsified or partially emulsified pulp in some of the flotation processes, *e.g.*, the oil-froth process at least.

### *Electrolytic and Electrostatic Phenomena*

Any substance which is placed in contact with water or many other liquids will assume an electric charge, the origin of which is, as yet, not set forth. Most substances when in contact with water become negatively charged, but these charges can be differed at will or reversed by the addition of the proper electrolyte in requisite amount. These electric charges are by no means confined to submicroscopic particles but are found also on the particles of a coarse suspension. Gangue minerals, *e.g.*, quartz, when suspended in water, are negatively charged, and sulphide minerals, *e.g.*, pyrite, are positively charged under like conditions. Oil drops are negatively charged, as are air bubbles under certain conditions which will be given subsequently. These charges are very minute when referred to the mass of the particle. Substantial evidence is at hand which goes to show that floatable minerals have the positive sign of electricity when suspended in water or can be made to assume that sign by the addition of proper electrolytes in sufficient amount. As Callow[10] observes, there is a parallelism between electrostatic character-

[10] J. M. Callow: *Bulletin* No. 108, p. 2334 (December, 1915).

istics and the flotation properties of ores. Many of the electrostatic principles have either been carried too far or misapplied as recent work shows.

Experiments in the realm of colloid chemistry indicate that the contact films are charged and that such charges markedly affect the dispersion or coherence of the particles in suspension. Naturally, oppositely charged contact films will coalesce while similarly charged contact films will repel each other, if the charges are sufficient in amount to overcome the force of cohesiveness; in the latter, dispersion is the result. The oil and air contact films having negative charges would tend to attract the sulphide particles, but further than this possibility electrostatics probably plays little part in flotation for the reasons given in the following paragraph.

As referred to above, it is pretty generally admitted that only minerals which are good conductors are suitable to flotation. Now then, as the electrical theory contends, electrified bubbles must be supplied to float the conducting minerals which are attracted, leaving behind the minerals which are not. The bubbles in flotation are simply air spaces contained by a mantle of oil or of water, and there is, therefore, nothing within to bear the charge. In case it could carry a charge, which would only be possible by the presence of contained ionized gases or water vapor, the charge would be speedily dissipated by contact with the interfacial boundary. Then in order for a bubble to carry a charge it must be protected by a dielectric film. Further, electrostatics plays probably little part in holding the sulphide particles and the gas bubbles together as neither the bubble nor the particle can have a charge of sufficient magnitude when referred to the size. The electrical theory has been strongly championed by at least one writer[11] and has been tolerated by some others. A recent article[12] indicates that the principles of electrostatics have been considerably misapplied. It is my belief that electrostatics may be a small contributing factor in flotation in a manner not as yet understood because of a lack of data concerning charges at the interfacial boundary between immiscible phases, *e.g.*, where the colloidal state is introduced in oil-water emulsions. Apparently, as far as any data at hand are concerned, they all serve to condemn any attachment of great importance to the electrical theory.

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[11] Bains: The Electrical Theory of Flotation, *Mining and Scientific Press*, vol. 111, No. 22, p. 824 (Nov. 27, 1915).

Bains: The Electrical Theory of Flotation, II, *ibid.*, vol. 111, No. 24, p. 883 (Dec. 11, 1915).

[12] Fahrenwald: The Electrostatics of Flotation, *ibid.*, vol. 111, No. 11, p. 375 (March 11, 1916).

## THE CHARACTER AND FORMATION OF FLOTATION FROTH

Bearing in mind the phenomena discussed above, let us turn to the flotation of minerals by means of the froth process and look into the character and formation of froth. A froth has been defined as a multiplicity of bubbles; this seems to be wanting in some respects but will be requisite and sufficient for this purpose. The froth of flotation is formed by the action of air or gas in water containing impurities or contaminants, for pure water will not maintain a froth.

### *Froth and Bubbles*

The idea has been abandoned of late, by most people, that a low surface tension is the essential requirement for froth formation. As mentioned by Coghill in a recent writing, the contamination of the liquid with an impurity which will cause a variable surface tension is the real requirement. A bubble of air is spherical in shape and this shape can only be maintained if the external pressure exceeds the internal pressure. Since a bubble does not expand *per se*, large bubbles can only be accounted for by heat, coalescence or electrification. Viscosity is an important factor in froth persistence as it increases the tenacity of the liquid film and thus prevents ready rupture. The rupture or bursting of bubbles is accounted for by these reasons:

1. Concussion upon a surface film which is deficient in the requisite viscosity and variable surface tension.
2. Relief of pressure—here the gas of the bubble in expanding exerts a pressure which exceeds that of the liquid film.
3. Adhesive force of the entrained gas for the atmospheric air.
4. Evaporation of the liquid film.

Flotation bubbles will burst then for any one or a combination of all these reasons.

Solutions in which the continuous phase is a solution of soap, various products from the saponification of albumens, etc., will froth voluminously even in a very diluted condition; frothing never occurs in pure liquids and is a definite proof that the solute or disperse phase lowers the surface tension of the solvent. A froth, which shows adsorption at the interfacial boundary of solution and gas, depends for its persistence on the production of a viscous film at that boundary; these viscous films are the direct result of surface adsorption of the disperse phase, *i.e.*, dissolved contaminants, the amount of which is small—disappearingly so. The work of Hall and of Miss Benson shows that in a foaming liquid the foam is richer in dissolved contaminant than is the bulk of the liquid. Froth formation in the Callow cell is the result of the injection of air into the pulp (already emulsified); the froth continues to obtain as long as

there is sufficient air injected into pulp of the proper consistency. The nature of the froth in the Callow cell is governed by the kind of oil used and by the amount of air. A pneumatic froth is unstable or ephemeral, *i.e.*, it dies rapidly when removed from the influence of the injected air. The mechanical froth, on the other hand, is thick and persistent, and must be broken up in dewatering the concentrates.

### *Oils*

Oils have a well-marked selective action for metallic sulphides, tellurides and some other minerals. The fact that both the oil and the air or other gas have a selective adhesion for sulphides prevents them from being wetted by water. Conversely, the quartz and other minerals exhibit just the opposite characteristics. The gangue minerals, generally, do not exhibit adhesion for either gas or oil and hence they are readily wetted by water. Gases have a well-defined adhesiveness for oils; hence the air or gas adheres strongly to the oil film. The stability of a froth depends, in the main, on the kind of oil used, *e.g.*, pine oil makes a weak brittle froth, and creosote makes a stable elastic froth. The work of Devaux[13] on oil films explains how so small an amount of oil as is used in the various flotation processes can be so efficacious. From a consideration of the immiscible oil-water interface, if any oil will film the internal surface of a gas bubble the sulphide mineral particles would be contained in the oil-water interface no matter what the nature of the gas contained by the water film. The sulphide if it enters the oil phase would then present an oiled surface to the water phase. There are three conditions then: (1) The mineral enters the oil phase completely; or (2) the mineral enters the water phase completely; or (3) the mineral enters the oil-water interface.

Some experiments made to determine the nature of the frothing power, selective and collective action of different oils show some interesting results. I made some tests on a zinciferous slime from the Joplin area with different oils and the results obtained indicate that a definite mixture of oils will effect better recoveries than any one oil alone. The best combination consisted of pine oil as a frother, plus wood creosote as a frother and selector, plus refined tar oil as a froth stiffener. The complete results of these experiments will be given in a later paper.

In general, pine oil makes a brittle froth which immediately dies; creosotes make a more elastic froth, the bubbles of which may expand to 3 in. in diameter or more before rupture. Coal-tar products are rather poor frothing agents and if used must be aided by either creosote or pine oil to produce a good froth. Oils of a lubricating nature seem to be of little value as flotation oils, while such light oils as gasoline and naphtha are of value only for thinning the heavy coal and wood tars.

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[13] Devaux: Oil Films on Water and on Mercury, *Smithsonian Report of* 1913, p. 261.

### *Air and Gas*

At this time, there are three ways by which a gas may be forced into a solution mechanically, viz.:

1. By beating it into the solution by means of beaters or paddles—as in the Minerals Separation and similarly mechanically agitated machines.

2. By pneumatic means—as in the Callow cell where the air is divided by the porous blanket bottom into minute sprays.

3. By so-called liquid jets—as in a process recently patented in which the air is introduced as a surface film surrounding a liquid jet by surface tension.

Conversely, there are three methods by which dissolved gas may be expelled from a liquid, viz.:

1. When the liquid is supersaturated, the excess gas is expelled.

2. By heating the liquid, when some of the gas is expelled due to an increase in its volume.

3. By pressure reduction, as in the Elmore vacuum process, where, according to the law of Henry,[14] "the amount of gas dissolved by a liquid is proportional to the pressure to which the gas is subjected."

An air or gas bubble on being introduced into a liquid is at once surrounded by a film of the liquid. Such a bubble will rise to the surface (carrying the metallic sulphides by reason of the forces already mentioned) on account of gravitation, by which is understood that the adherence of the air to the liquid is less than the force of gravity.

## Résumé

From a consideration of the foregoing, it is believed that the theory based on the different interfacial tensions involved is the dominating one as well as the more consistent at this time. Probably flotation is due to a combination of phenomena which are rather high in the scale of complexity. The theory based solely on occlusion goes "by the board" as it has been shown that the contributing effect of this phenomenon has been interpreted rather laxly.[15] The phenomenon of electrostatics may be a small contributing factor but recent work indicates that the principles have been misapplied. An explanation more in consonance with fact can be given in terms of the interfacial tensions involved, without postulating either occlusion or electrostatics.

The main and essential requirements for froth flotation are: (1) The production of a persistent froth by any means; (2) the attachment of the bubbles of air to the sulphides or other material to be floated; and (3)

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[14] Jones: *Elements of Physical Chemistry*, p. 177, 1907.

[15] Ralston: Why Do Minerals Float? *Mining and Scientific Press*, vol. cxi, No. 17, p. 623 (Oct. 23, 1915).

the maintaining of a selective action of the froth bubbles for the sulphides or other material to be floated.

Why this is accomplished is outlined in the above, particularly under the subhead, "Adsorption." Probably, before any generally satisfactory estimate of the many complex factors, which are supposed to be involved, can be secured, a more thorough investigation of all of them will have to be made than has thus far been attempted. However, in this line of inquiry there has been a steady progression of thought and a remarkable increase in knowledge in the past few years; and the art of flotation will continue to improve and develop as knowledge of the different factors involved increases and allows the exercise of a better control of them.

## Bibliography

There is appended here a portion of the bibliography Concentrating Ores by Flotation, *Bulletin of the School of Mines and Metallurgy, University of Missouri*, vol. 8, No. 1, 1916, compiled by Jesse Cunningham.

## Colloids and Surface Tension

### Bibliography

Müller, A. Bibliography of Colloids. *Zeitschrift für Anorganische Chemie*, vol. 39, p. 121, 1904. "A bibliography of colloids containing three hundred and fifty-six references. There is an appendix in which the articles are grouped according to their subject" (*Journal of Physical Chemistry*).

### *General Works*

Börnstein, Richard, and Roth, Walther A. *Landolt-Börnstein physikalisch-chemische tabellen.* 4 ed., Berlin, 1915. Tables and data.

Boys, C. V. *Soap-bubbles: Their Colors and the Forces Which Mould Them.* London, 1912, Illus.

Findlay, Alex. *Practical Physical Chemistry.* 3 ed., London, 1915.

Freundlich, Herbert. *Kapillarchemie. Eine darstellung der kolloide und vernandter gebiete*, Leipzig, 1909. The author has divided the work into two parts, the first is concerned with the properties of surfaces in general and part two is devoted to dispersed systems. Foams are treated in part two.

Getman, Frederick H. *Laboratory Exercises in Physical Chemistry*, N. Y., 1904.

Gibbs, J. Willard. The scientific papers, 2 vol., N. Y., 1906. The equilibrium of heterogeneous substances, pp. 54–353, especially p. 219. Reprinted from *Transactions of the Connecticut Academy*, vol. 3, pp. 108–248; 343–524. 1875–1877. Abstract of the above paper, pp. 354–371. Reprinted from *American Journal of Science*, Series 3, vol. 16, pp. 441–458, 1878.

Thermodynamische studien unter mitwirkung des verfassers aus dem Englischen ubersetzt von W. Ostwald, Leipzig, 1892.

Gray, Francis W. *A Manual of Practical Physical Chemistry*, London, 1914, pp. 65–69.

Janek, A. *A Textbook of Dispersoidology* (Modern colloidal chemistry), Petrograd, 1915. Carefully systematized material on the subject of colloidal chemistry

with references and methods. Valuable for experimental and research work. In Russian.

KOHLRAUSCH, FRIEDRICH. *Lehrbuch der practischen physik.* 11th ed., Leipzig, 1910.

LUTHER, R. UND DRUCKER, K. *Ostwald-Luther hand- und hülfsbuch zur aufführung physiko-chemischer messungen.* 3 auflage, Leipzig, 1910, pp. 233–241.

MARIE, CH. ED. *Tables annuelles de constantes et donnees numeriques de chimie, de physique et de technologie.* vol. 1–date; 1910–date, Paris; 1912–date. Tables and data.

MÜLLER, ARTHUR. *Allgemeine chemie der Kolloide. Handbuch der angewandten physikalischen chemie,* Band 8, 1907. A valuable summary of much of the work on colloids giving a mass of well-arranged data under the following heads: Methods of preparing inorganic colloids; organic colloids; properties of colloidal solutions; colloidal gels; theories of colloids; systematic classification of colloids.

OSTWALD, WILH. *Lehrbuch der allgemeinen chemie.* 2 auflage, 1 Band, 1903, pp. 514–543. Chapter on surface tension.

PARTINGTON, JAMES RIDDICK. *A Textbook of Thermodynamics* (with special reference to chemistry), pp. 429–449, Illus., N. Y., 1913.

POCKELS, F. KAPILLARITÄT. (In Winklemann, A. *Handbuch der physik.* 2 auflage, 1 Band, 2 Hälfte, pp. 1119–1234, 1908.)

ROHLAND, PAUL. *The Colloidal and Crystalloidal State of Matter,* London, 1911.

SPENCER, JAMES FREDERICK. *An Experimental Course of Physical Chemistry,* pp. 139–148, Illus., London, 1911.

STIEGLITZ, JULIUS. *The Elements of Qualitative Chemical Analysis,* pp. 125–138, N. Y., 1911. A very clear explanation of the colloidal condition, easily understood by students.

SVEDBERG, DR. THEODOR. *Die Methoden zur Herstellung Kolloider Lösungen Anorganischer Stoffe,* Dresden, 1909.

TAYLOR, W. W. *Chemistry of Colloids and Some Technical Applications,* N. Y., 1915.

VAN BEMMELEN, J. M. *Die Absorption Gesammelte Abhandlungen über Kolloide und Absorption, mit unterstutzung des verfassers neu herausgegeben von Wo. Ostwald,* Dresden, 1910. The collected works of Van Bemmelen on colloids and absorption.

WATSON, W. *Textbook of Physics.* 8th imp., 4 ed., London, 1907, pp. 180–184.

WILLOWS, R. S. AND HATSCHEK, E. *Surface Tension and Surface Energy and Their Influence on Chemical Phenomena,* Philadelphia, 1915. Treats existence and theory of surface tension; relations between surface tension and other physical constants; relations between surface tension and chemical constants; factors affecting distribution of a solute in solution; effect of electric charge on surface tension.

WINKELMAN, A. Absorption und adsorption. (In his *Handbuch der physik.* 2 auflage, 1 Band, 2 Hälfte, pp. 1507–1542, 1908.)

ZSIGMONDY, RICHARD. *Colloids and the Ultramicroscope.* A manual of colloid chemistry and ultramicroscopy. Authorized translation by Jerome Alexander, N. Y., 1909. A general outline of colloid chemistry with a detailed account of the characteristics of special colloids including colloidal sulphides, oxides, metals, and non-metals.

*Articles in Periodicals*

ALLEN, A. W. Colloids in Ore Dressing. *Mining and Scientific Press,* vol. 107, pp. 109–110, July 19, 1913.

Ashley, Harrison Everett. The Colloid Matter of Clay and Its Measurement, 1909. *U. S. Geological Survey, Bulletin* 388.

Technical Control of the Colloidal Matter of Clays, November, 1911. *U. S. Bureau of Standards, Technical Papers No.* 23.

Bancroft, Wilder D. The Coagulation of Albumin by Electrolytes. *Journal of Physical Chemistry*, vol. 19, pp. 349–359, May, 1915. *Transactions of the American Electrochemical Society*, vol. 27, pp. 195–207, 1915.

Naturalization of Adsorbed Irons. *Journal of Physical Chemistry*, vol. 19, pp. 363–376, May, 1915. *Transactions of the American Electrochemical Society*, vol. 27, pp. 175–194, 1915.

The Theory of Colloidal Chemistry. *Journal of Physical Chemistry*, vol. 18, pp. 549–558, October, 1914.

The Theory of Emulsification. *Journal of Physical Chemistry*, vol. 16, pp. 177–233, March, 1912, Illus., pp. 345–372. May, 1912; pp. 475–512. June, 1912, Illus.; pp. 739–758. December, 1912; vol. 17, pp. 501–519. June, 1913; vol. 19, pp. 275–309. April, 1915; pp. 513–529. June, 1915.

These papers develop and expound the best general working theory of colloids, froths, emulsions and adsorption that can be found. With rare critical acumen, the complete literature on these subjects is passed in review, sorted, sifted, amended, and recombined into an eclectic system which, while years in advance of contemporaneous treatises, is reduced to such simple, common-sense terms as to be within range of any serious-minded student.

The style is unhappily difficult, so that the following suggestions may not be amiss. After several readings of the short paper on The Theory of Colloid Chemistry, read first the excellent summaries at the ends of the Theory of Emulsification papers before undertaking a rapid survey of the whole set; follow this by a more careful consideration of the summaries with re-reading of portions of the text when necessary; continue this process until the drift of the argument begins to reveal itself.

Other papers from the same author's laboratory on jellies, emulsions, and the theory of dyeing will be found to be not unrelated, although seemingly more remote to a possible future theory of flotation.

An eighth article on Emulsification appears in the *Journal of Physical Chemistry*, vol. 20, p. 1, January, 1916, and of this series there will be at least one more preliminary paper on froths followed by an article on the flotation of ores.—V. H. Gottschalk.

Bartlett, J. C. The Action of Small Spheres of Solids in Ascending Currents of Fluids, and in Fluids at Rest. *Transactions American Institute of Mining Engineers*, vol. 6, pp. 415–427, May, 1877.

Benson, Clara C. The Composition of the Surface Layers of Aqueous Amyl Alcohol. *Journal of Physical Chemistry*, vol. 7, pp. 532–536, October, 1903, Illus.

Billitzer, Jean. Eine Theorie der Kolloide und Suspensionen. *Zeitschrift für Physikalische Chemie*, 45 Band, pp. 307–330, Oct. 13, 1903.

Theorie der Kolloide, II. *Zeitschrift für Physikalische Chemie*, 51 Band, pp. 129–166, Feb. 28, 1905.

Boys, C. V. Experiments with Soap Bubbles. *Smithsonian Institution Annual Report*, 1912, pp. 211–218.

Bradbury, Robert H. Colloids and Crystals, the Two Worlds of Matter. *Journal of the Franklin Institute*, vol. 176, pp. 319–328, September, 1913.

Briggs, T. R. Experiments on Emulsions: Adsorption of Soap in the Benzine-Water Interface. *Journal of Physical Chemistry*, vol. 19, pp. 210–231, March, 1915, Illus.

Briggs, T. Roland and Schmidt, Hugo F. Experiments on Emulsions, II. Emul-

sions of Water and Benzene. *Journal of Physical Chemistry*, vol. 19, pp. 478–499, June, 1915, Illus.

CAETANI, GELASIO. Sand, Slime and Colloids in Ore Dressing. *Mining and Scientific Press*, vol. 106, pp. 438–442, Mar. 22, 1913.

CAMERON, FRANK K. Soil Colloids and the Soil Solution. *Journal of Physical Chemistry*, vol. 19, pp. 1–13, January, 1915.

Colloids and Their Importance. *Mining and Scientific Press*, vol. 107, p. 87, July 19, 1913.

CORLISS, HARRY P. The Distribution of Colloidal Arsenious Sulphide between the Two Liquid Phases in the System Water, Ether, Alcohol. *Journal of Physical Chemistry*, vol. 18, pp. 681–694, November, 1914, Illus.

DEVAUX, H. E. Oil Films on Water and on Mercury. *Smithsonian Institution, Annual Report*, 1913, pp. 261–273, Illus.

DONNAN, F. G. AND BARKER, J. T. Experimental Investigation of Gibb's Thermodynamical Theory of Interfacial Concentration in the Case of an Air-Water Interface. *Royal Society Proceedings*, Series A, vol. 85, pp. 557–573, Nov. 30, 1911.

GARRISON, FIELDING, H. Josiah Willard Gibbs and His Relation to Modern Science. *Popular Science Monthly*, vol. 74, pp. 470–484. May, 1909; pp. 551–561. June, 1909; vol. 75, pp. 41–48. July, 1909; pp. 191–203. August, 1909. One of the best statements of Gibb's theory to be found.

GARVER, M. M. A New Method of Determining the Range of Molecular Action and the Thickness of Liquid Films. *Journal of Physical Chemistry*, vol. 16, pp. 234–246, March, 1912.

On the Molecular Constitution of the Free Surfaces of Liquids. *Journal of Physical Chemistry*, vol. 17, pp. 386–389, May, 1913.

GIBSON, A. H. The Behavior of Bodies Floating in a Free or a Forced Vortex. *Manchester Literary and Philosophical Society Memoir No.* 7, 1911, 19 p.

The Manner of Motion of Water Flowing in a Curved Path. *Manchester Literary and Philosophical Society Memoir No.* 13, 1911.

HARRIS, J. E. Some Adsorption Phenomena in Soils and Kaolin. *Journal of Physical Chemistry*, vol. 18, pp. 355–372, April, 1914.

HAULTAIN, H. E. T. Some Early Notes on Surface Tension Action in Wet Concentration. *Transactions Canadian Mining Institute*, vol. 16, pp. 115–136, 1913.

JAMISON, R. On the Film or Skin of Warmed Milk and Other Proteid Solutions. *Journal of Physiology*, vol. 27, p. 26, 1901–1902.

KENRICK, FRANK B. Some Lecture Experiments on Surface Tension. *Journal of Physical Chemistry*, vol. 16, pp. 513–518, June 6, 1912, Illus.

LEWIS, W. C. M. An Experimental Examination of Gibb's Theory of Surface-concentration, Regarded as the Basis of Adsorption, with an Application to the Theory of Dyeing. *Philosophical Magazine*, vol. 15, 6 Series, pp. 499–526, April, 1908; vol. 17, 6 Series, pp. 466–494, April, 1909; Illus. From the Muspratt Laboratory of Physical and Electro-chemistry, University of Liverpool.

MILLER, W. LASH AND MCPHERSON, R. H. The Behavior of Colloidal Suspensions with Immiscible Solvents. *Journal of Physical Chemistry*, vol. 12, pp. 706–716, December, 1908, Illus.

MINKOWSKI, H. KAPILLARITÄT. *Encyklopädie der Mathematischen Wissenschaften*, Band 5, Heft 4, pp. 558–613, 1907, Illus.

MUKHOPADHYAYA, JNANENDRANATH. Coagulation of Arsenious Sulfide Sol. by Electrolytes. *Journal American Chemical Society*, vol. 37, pp. 2024–2031, September, 1915.

NICOLAI, G. Zur Klärung von Erzaufbereitungsabwässern. Beiträge zur Entwick-

lung der mechanischen und neuere Erfahrungen auf dem Gebiete der mechanisch-chemischen Aberwasserklärung unter besonderer Berücksichtigung von Bleierzaufbreitung. *Metall und Erz*, vol. 12 (N. F. III), pp. 135–140; 155–162, 1915, Illus.

NOYES, ARTHUR A. The Preparation and Properties of Colloidal Mixtures. *Journal American Chemical Society*, vol. 27, pp. 85–104, February, 1905. A general discussion for students.

The Preparation and Properties of Colloidal Mixtures. *Popular Science Monthly*, vol. 67, pp. 268–279, July, 1905.

Oil Films on Water. *Mining and Scientific Press*, vol. 111, p. 156, July 31, 1915.

PAPE, W. A. C. Why Do Many Solids Float on Their Liquids? *Metallurgical and Chemical Engineering*, vol. 10, p. 392, July, 1912.

POCKELS, AGNES. Ueber das spontane Sinken der Oberflächenspannung von Wasser, wässeringen Lösungen und Emulsionen, *Annalen der Physik*, IV, 8 Band, pp. 854–871, July, 1902.

QUINCKE, G. Die Bedeutung der Oberflächenspannung für die Photographie mit Bromsilbergelatine und eine neue Wirkung des Lichtes. *Annalen der Physik*, IV Folge, 11 Band, pp. 1100–1120, July, 1903.

NEWMAN, F. R. Experiments on Emulsions. *Journal of Physical Chemistry*, vol. 18, pp. 34–54, January, 1914.

QUINCKE, G. Die Messungen des Hrn. Gallenkamp mit Adhasionsplatten. *Annalen der Physik*, IV Folge, 10 Band, pp. 453–456, January, 1903.

Niederschlagmembranen und Zellen in Gallerten oder Lösungen von Leim, Eiweiss und Starke. *Annalen der Physik*, IV Folge, 11 Band, pp. 449–488, June, 1903.

Die Oberflächenspannung an der Grenze von Alkohol mit wässeringen Salzlösungen. Bildung von Zellen, Sphärokrystallen und Krystallen. *Annalen der Physik*, IV Folge, 9 Band, pp. 1–43, August, 1902.

Die Oberflächenspannung an der Grenze wässeriger Kolloidlösungen von verschiedener Koncentration. *Annalen der Physik*, IV Folge, 9 Band, pp. 793–836, November, 1902; pp. 969–1045. December, 1902, Illus.; 10 Band, pp. 478–521. February, 1903; pp. 673–703. March, 1903.

Oberflächenspannung und Zellenbildung bei Leimtannatlösungen. *Annalen der Physik*, IV Folge, 11 Band, pp. 54–95, April, 1903.

Die schaumstrukture des schwefels und deren einfluss auf doppelbrechung, dichroismus, elektrische eigenschaften und kristallbildung. *Annalen der Physik*, IV Folge, 26 Band, pp. 625–711, July 28, 1908.

Ueber kolloidale Lösungen. *Annalen der Physik*, IV Folge, 12 Band, pp. 1165–1168, November, 1903.

Ueber die Klärung truber Lösungen. *Annalen der Physik*, IV Folge, 7 Band, pp. 57–96, December, 1901.

Ueber unsichtbare Flüssigkeitsschichten und die Oberflächenspannung flüssiger Niederschlage bei Niederschlagmembranen, Zellen, Colloiden und Gallerten. *Annalen der Physik*, IV Folge, 7 Band, pp. 631–682, March, 1902, pp. 701–744, April, 1902.

RAMSDEN, W. Die Coagulirung von eiweisskörpern auf mechanischem wege. *Archiv für Anatomie und Physiologie. Physiologische abtheilung*, 1894, pp. 517–534.

Separation of Solids in the Surface-layers of Solutions and "Suspensions." Observations on surface-membranes, bubbles, emulsions, and mechanical coagulation. (Preliminary account.) *Chemical News*, vol. 88, pp. 49–51, July 31, 1903.

RAYLEIGH, LORD. Investigations in Capillarity. The size of drops. The liberation of gas from supersaturated solutions. Colloiding jets. The tension of

contaminated water-surfaces. *Philosophical Magazine*, vol. 48, pp. 321–337, October, 1899.

On the Superficial Viscosity of Water. *Royal Society Proceedings*, vol. 48, pp. 127–140, June, 1890, Illus.

RICHARDSON, CLIFFORD. A Unique Geophysical Phenomenon, Trinidad Asphalt, Interesting from the Point of View of Dispersoid Chemistry. *Journal of Physical Chemistry*, vol. 19, pp. 241–249, March, 1915.

SULMAN, H. L. New Discoveries in the Physics of Ore Flotation. (In Presidential address.) *Institute of Mining and Metallurgy Bulletin No.* 79, pp. 9–17, April 19, 1911.

Suspension Colloids. *Mining and Scientific Press*, vol. 106, p. 209, Feb. 1, 1913.

SWINBURNE, J., AND RUDORF, G. The Physics of Ore Flotation. *Engineering and Mining Journal*, vol. 81, pp. 276–277, Feb. 10, 1906. *Mining and Scientific Press*, vol. 92, pp. 126–127, Feb. 24, 1906; abstract of a paper read before the Faraday Society, Dec. 12, 1905.

WASHBURNE, C. W. The Capillary Concentration of Gas and Oil. *Transactions American Institute of Mining Engineers*, vol. 50, pp. 829–858, 1915.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Diastrophic Theory*

## A Contribution to the Study of the Mechanics of Oil and Gas Accumulation in Commercial Deposits

BY MARCEL R. DALY,† SEATTLE, WASHINGTON

(Arizona Meeting, September, 1916.)

THE writer has devoted a number of years to practical operations and to the study of geology in the oil fields. In consequence, he has been brought to investigate the theories advanced to account for the accumulation of oil and gas in commercial deposits. It is the result of these investigations and his personal conclusions that he wishes to sum up in this paper.

The writer is an advocate of the organic origin of petroleum found in pools. He has been led to believe that the present theories of oil and gas accumulations are incomplete and, in certain respects, incorrect, although they may embody certain elements of truth; that the forces that are called upon to explain the movement are only secondary forces in the process, and insufficient, by themselves, to cause this movement; and that the phenomenon of accumulation is of much larger order than heretofore admitted and bears an intimate relation with the general as well as with the local deformations of the crust and is a direct consequence and a mechanical effect of these deformations to which the term "diastrophic" has been applied. As a complement to this theory, the writer offers a new interpretation of the "rock pressure" and of the mechanism of the "sealing up" of this pressure in oil and gas pools.

### *The Anticlinal Theory*

The theory most generally accepted to account for the accumulation of hydrocarbons in commercial deposits is the anticlinal or structural theory. This theory, the principles of which have been laid down by I. C. White, Orton, Winchell and others, and which has since been modified and perfected, is too well known to need a detailed statement. A clear interpretation of it may be found, in the paper of W. T. Griswold, in *Bulletin No.* 318 *of the United States Geological Survey*, and the reader is

* Originally presented at a meeting of the Puget Sound Section on Jan. 29, 1916.

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referred to it. In this theory, the *force* which is supposed to have caused the motion is the gravity of the hydrocarbons; and the principal *factors* which have intervened in the process are the structure and the general stratigraphic conditions of the rocks, their texture and porosity, and their water content. The oil and the gas are supposed to enter the porous rock that is to act as a reservoir, by some indefinite process. Any further movement toward accumulation is only considered as possible if a dip exists in this rock sufficient to overcome the friction, and practically impervious areas of the rock will be more or less perfect barriers against the movement. Then, if this rock is completely saturated with water, or if the hydrocarbons have entered it below the line of complete saturation, the oil and the gas will gradually move up the slope by the effect of buoyancy; the gas, with its lower specific gravity, occupying the higher places. Should the rock be dry, or if the hydrocarbons have entered the rock above the line of saturation, the oil will flow down, as long as gravity is sufficient to overcome the resistance to motion, and the gas will diffuse with the air or water vapor contained in the pores of the rock.

From this statement it will appear that the following assumptions are an absolute requisite for the anticlinal theory:

First, the structural deformations (dips, anticlines and synclines, domes, terraces, etc.) must be supposed to have existed previous to the introduction of the hydrocarbons in the porous strata; in other words, it must be assumed that structural deformation has preceded the movement toward accumulation. Indeed, in this theory, the preëxistence of structural deformation is the very basis of the action of gravity and, at the same time, the theory implies that the forces which have caused the deformations have had no bearing on the movement of the hydrocarbons.

Second, gravity or buoyancy is to be considered the sole agency through which accumulation has been brought about and, as such, is supposed to be adequate to explain accumulation under any condition of dip.

To which the following considerations may be added: The movement toward accumulation would have to take place in the porous rocks when they are already solidified and partially cemented. There would be no connection between the causes which have led: To the introduction of the oil in the porous strata; to the movement of the fluids in these same strata; and to the "rock pressure" itself. Further, this theory does not provide for any satisfactory explanation as to the mechanism of the "sealing up" of the pressure in the pools.

The first assumption is begging the question. To the best knowledge of the writer, no attempt has been made by the authors of the theory even to discuss the point or to prove the accuracy of the inference. It seems as if, from the start, it was admitted as self-evident that the gravity of the fluids was the only possible force entering into action. The writer will try to show later on why, in his opinion, this assumption

falls short of truth and how its admittance may be explained by the fact that the problem has not been attacked from its true angle. On this first point, to say the least, the anticlinal theory is not established on proven ground.

The second assumption may be seriously contested. The force of motion due to buoyancy is a function of the sine of the dip. It would be maximum for a vertical stratum and null for a horizontal one. Many anticlines in the Eastern fields of the United States have a dip so low that a motion caused by buoyancy alone could hardly be understood. Dr. Ashburner has found a maximum dip in the Bradford region of 69 ft. per mile, or less than 1 ft. for 76 ft., while Carll has shown that the dip of the oil sands in the Venango belt, and the southern end of the Butler belt, rarely exceeds 34 ft. per mile (Sir Bowerton Redwood), or less than 1 ft. for 155 ft. In West Virginia, the maximum dip of many anticlines is less than 1 ft. for 50 ft., which corresponds to an angle of about 1° 8′ 45″. For this last dip, the force due to buoyancy, which is supposed to move the oil up the slope, would be reduced to 0.0066 of the weight of the oil, as pointed out by Malcolm J. Munn, of the U. S. Geological Survey. A volume of 1 c.c. of oil would thus be submitted to a moving force of a few milligrams. The oil could be set in motion only if this force were greater than the resistance due to static friction, viscosity and surface tension. It is not possible, with the data now at hand, to submit the question to a complete mathematical treatment. As Van Hise points out,[1] "the pore openings in sandstones are for the most part capillary", and the flow of oil through the rock would have to obey the laws of flow through capillary tubes, which are not yet satisfactorily expressed. Further, the pore openings would be extremely irregular in shape and dimensions. But it is possible to reduce the problem to a simpler form. If, barring viscosity and surface tension, we would only consider the resistance due to static friction, we would obtain an inferior limit to the resistance. And if it were possible to show that the force of motion due to buoyancy is insufficient to overcome such a frictional resistance, in the case under consideration, it would be proof that the anticlinal theory can not be accepted, at least in its present form, and that other forces besides buoyancy are to be considered in order to explain the motion.

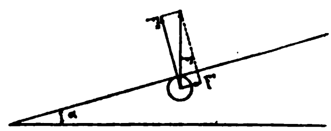

FIG. 1.

Let us imagine a globule of oil appressed against the roof of the sandstone stratum by the effect of buoyancy (Fig. 1). Let $F$ be the force of buoyancy. This force may be decomposed into two components: $F'$ parallel to the dip, which would cause the globule to slide along the roof, and $F''$ normal to the roof, which would appress the drop of oil against it.

[1] A Treatise on Metamorphism, *Monograph* 47, *U. S. Geological Survey*, p. 138 (1904).

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The effect of this pressure is to create a resistance to the sliding, which is proportional to the pressure and directed down dip. This proportion, which remains constant when the nature of the surface remains the same, is the friction coefficient. Let it be $c$. The movement will become possible only if $c < \frac{F'}{F''}$ or $c <$ tang $\alpha$, $\alpha$ being the angle of the dip. If the dip is 1 ft. to 50 ft., the tangent is about 0.02 and the condition reads $c < 0.02$.

There are no data available for the friction coefficient of oil and sandstone under the conditions stated. But one may get an idea of the order of magnitude of this coefficient by a simple experiment. Let us take a slab of some oil-bearing sandstone, some Berea grit, for instance. The lower surface of this slab must be planed, but not ground, so that the grains of sand remain entire. Let us immerse the slab in some salt water of proper density, in a horizontal position, and loosen a drop of oil from a syringe in the water, under the slab. The drop will rise to the bottom of it, and here it will remain if the slab is perfectly horizontal. Let us then incline the slab progressively and note the smallest angle of dip at which the drop will be set in motion. The tangent of this angle will be equal to the coefficient of static friction.

The writer has made this experiment repeatedly and has never succeeded in getting the drop in motion for any such angle as 2°, which would tend to prove that the coefficient is superior to the maximum limit required.

It is further to be noted that the spherical form is the general form of oil drops in an emulsion, where the particles of oil are, in great majority, very small; it is also the form which, every other thing being equal, offers the least resistance to motion. Hence, if the motion is shown to be impossible under this form, it will be equally impossible under any other form, and the whole theory falls through.

The change from a spherical form to an irregular one, by the flattening of the drops when compressed against the walls or when passing through irregular and narrow channels, the indentations, the penetration of the oil between the grains, the effect of viscosity, the capillary action—especially if gases be held in the emulsion—would be as many causes furthering an increased resistance to motion.

Now, the fact that the movement toward accumulation is not possible under the sole influence of gravity, with the low dips so characteristic of many oil regions, does not mean that gravity has had no influence in the process. It simply means that this force alone would be inadequate to explain the motion and that some other force or forces must have entered into play. But though gravity of hydrocarbons may not be a primary agent of their accumulation, it is a force which must be taken into account in any general theory of the process.

*The Hydraulic Theory*

In opposition to the anticlinal theory of oil and gas accumulation through gravity, Malcolm J. Munn has propounded[2] another theory in which the moving force would be the hydraulic (not hydrostatic) pressure and capillarity of underground waters. This is termed *the hydraulic theory.*

"The fundamental idea of the hydraulic theory," writes Mr. Munn, "is that moving water under either hydraulic or capillary pressure has been the direct agent of accumulation of oil and gas pools. To this idea may be added another of equal value—the pools of oil and gas are held in place by water under hydraulic and capillary pressure which effectively seals up all the pores of the surrounding rock and prevents the dissipation of pressure by diffusion." This theory is very interesting and, as will be seen later on, the writer agrees with Mr. Munn on two points, viz., that water under hydraulic pressure has really been the primary agent of motion, and that the "sealing up" of the pools is a phenomenon of surface tension. But, as far as the writer can see, Mr. Munn has reduced the hydraulic pressure to that of underground waters circulating in the same way as they do now, and his statement relating to "hydraulic and capillary pressure" is rather undetermined.

"Capillary pressure" is a somewhat misleading term. If by this is meant the general action of forces due to surface tension, it is well to note that surface tension may create a resistance against the flow as well as a tendency toward it, as Mr. Munn himself seems to acknowledge when he comes to the "sealing up" of the pools. If it simply means the process of infiltration, the creeping of a liquid through capillary channels, this process has its limitations and would seem to be a process of dissemination rather than of concentration. On the other hand, hydraulic pressure of underground waters may be inadequate. Hydraulic pressure is a function of the square of velocity, and the velocity of underground waters is known to be extremely small. "The motion of the ground water as a whole," writes C. S. Slichter,[3] "is somewhat like the slow motion of very viscous sirup or the slowly creeping ice of a glacier." It has been a shown, for instance, that the ground waters of the Arkansas River flow in gravels at a rate not greater than 3 to 5 m. a day. This would mean a velocity between 0.00361 and 0.0058 cm. per second, or less than from 13 to 21 cm. per hour. It is possible to show that, should the static coefficient of friction between oil and sandstone be greater than 0.02, as heretofore admitted, a velocity of 0.006 cm.-sec. would not overcome the frictional resistance of oil in an inclosed pipe, on a horizontal

[2] *Economic Geology*, vol. 4, No. 6, p. 523 (September–October, 1909).

[3] The Motions of Underground Waters. *Water Supply. Paper, No.* 67, *U. S. Geological Survey,* p. 35.

plane, barring the additional resistances due to viscosity, surface tension, deformation, etc., which would interfere in this case. Though the analogy may not be entirely correct, it has a tendency to show that the influence of hydraulic pressure of underground waters is here doubtful and at least remote in importance. There will be always a great difference between the transportation of a material in the state of solution, which may be followed by subsequent re-deposition, and the transportation of insoluble solid material, as oil in water. Further the theory is open to the same general line of objections as the anticlinal theory, and more so, as structural deformation, which is so intimately connected with the phenomenon of accumulation, becomes here entirely subordinate and incidental.

### *The Diastrophic Theory*

The fundamental weakness of the theories which we have reviewed may perhaps be traced to the hesitation of the authors of these theories to choose deliberately between the different conceptions of the origin of oil itself. This origin once ascertained, it is evident that the surest way to determine how oil and gas may have accumulated in certain places, would be to try to follow the possible movements of the oil from the time of its first appearance in the strata, down to the pools where we find oil today. In order to be rational, a history ought to be complete; and we have no more right to limit the history of petroleum to the space of time during which the strata, in which we find it now, may have existed in the same present shape and condition, than we would have the right to reduce the story of a man to the last years of his life. It is further obvious that the history of petroleum will have to be entirely different, dependent upon its origin, either from emanations coming from the depths, or from organic decomposition in the strata themselves. In the writer's opinion, many reasons favor the theory of organic origin for the petroleum found in commercial deposits, though this does not mean that petroleum of a different origin does not exist elsewhere. Hence, this is a point on which we have first to agree, or else agree to disagree. Should the organic origin of the petroleum that is found in pools be granted, the following interpretation is offered for the mechanics of its accumulation.

Oil developed by the process of decomposition of organic matter—whether vegetal, animal or both—would have first to exist in the sediments which contain the parent matter, in the state of dissemination. As Orton remarks, "disseminated petroleum is well nigh universal, but accumulations are rare."

In order that such an accumulation of disseminated particles may take place, three elements are necessary: An adequate source of supply; a reservoir to hold the oil and accompanying gas under pressure, and a process of concentrating the disseminated material and conveying it to

the reservoir. The source of supply will have to be ascertained in every individual case. It is supposed here to exist, for the sake of argument. The reservoir is easy to imagine, inasmuch as we are able to corroborate our theoretical views by actual knowledge. A porous stratum overlaid by an impervious one, may act as a reservoir. This is the general case. Or the reservoir may be constituted by a porous stratum highly inclined and cut off abruptly below ground by a fault, which thus seals the oil and gas and prevents their escape. This kind is found in some Californian fields and elsewhere. Or the reservoir may be constituted by joint cracks in a shale, as is the case in the Florence oil field, Colorado,[4] etc.

The interpretation of the process of concentration and eventual migration, *i.e.*, of accumulation, is more complicated. As a natural process, it must obey physical laws, and especially the laws which govern the motion of fluids and gases; but the *forces* which may cause the motion (gravity in its different forms, heat, surface tension), the *agents* through which these forces accomplish certain results (water, oil and gas in varying physical or chemical states), and the *factors* that intervene in the process (structure and texture of the rocks, porosity, stratigraphic conditions, water and gaseous contents, viscosity, capillarity, depth and time) may have or have had a widely varying range of influence, according to past and present conditions. These elements would have to be seriated, according to their probable rank of importance, and their relative degree of influence determined, before a tentative conclusion could be reached. It thus becomes evident that if some broad general principle may be laid down, a great number of variations are to be expected for each individual case.

A few general observations may help us from the start. According to the organic theory, which is accepted here, the ultimate source of the hydrocarbons is to be found in sedimentary organic deposits, and oil accumulations in commercial quantities (oil pools) are always associated with sedimentary strata. The immediate inference is that hydrocarbons must have been submitted, from the date of their origin, to the action of the forces that may have affected these same strata.

If we refer to a map of the known oil fields of the globe, we will notice, with Sir Bowerton Redwood, that "whilst petroleum exists very generally distributed throughout the world, the principal deposits occur along well-defined lines, often associated with the principal mountain chains." This remark is important and deserves to be emphasized. It means that petroleum deposits are mainly associated with the lines of lesser resistance of the globe, with the general direction of geosynclines and consequently with the areas of general orographic movement and deformation. The distribution of the Eastern oil fields of the United States and Canada along the Paleozoic trough, or geosyncline, of the

[4] C. W. Washburn. *Bulletin No.* 381, *U. S. Geological Survey.*

Appalachian region, the location of the Western oil fields of the Pacific coast, from Alaska to California, followed by the Mexican oil fields, the deposits of the West Indies, of the northern coast of Venezuela, Columbia and a part of the Andes, along the general trend of a Mesozoic syncline; the oil fields of Galicia, Roumania, of the Caucasus, of Burma, of the Islands of Java, Sumatra, Borneo and New Zealand, following the path of another syncline, are extremely suggestive.

This first observation may be completed by a second. An oil field is always accompanied by a certain amount of local structural deformation, which sometimes is reduced to simple undulations, as is the case of the Appalachian region, or may reach a stage of high disturbance, with contorted strata, as is the case of the Galician oil fields. But local deformation has been observed everywhere. It becomes thus difficult to escape the impression that structural deformation, general as well as local, is more or less connected with the phenomenon of accumulation, and our inquiry is thus directed toward the possible action of the forces which have produced these deformations in the sediments from which hydrocarbons have proceeded and their possible action on the hydrocarbons themselves.

As stated before, oil would have first to exist in the stratum where organic decomposition takes place, in the shape of a finely disseminated matter. The sediments themselves would have to be deposited under water, in lagoons, marshes, deltas or at the bottom of the seas, in the form of mud or ooze. Such a mud would be composed of extremely fine mineral particles, intermingled with water in large amount. The particles of oil would take the form of spherical globules under the influence of surface tension, and, as these globules would be larger than the mineral particles of mud, they would be mechanically held in the mixture. Experiments conducted by Murray Stuart, Assistant Superintendent of the Geological Survey of India and others, have clearly established this capacity of muds to hold oil in sediments by purely mechanical action.[5]

The preceding condition may be termed the *first phase* of the process. It is characterized by a layer of mud or ooze, holding mechanically, between its very fine mineral particles, disseminated drops of oil and gas bubbles, in a "matrix of water," as Murray Stuart expresses it.

When the conditions of sedimentation which allowed the deposition of the mud have changed, some other material may be deposited over this first layer; say some sand. Then, another change in sedimentary conditions will occur, and a new layer of material will cover the sand; suppose another layer of mud; and so on, successive layers of various sediments will be piled up in the sedimentary syncline.

Now, the progressive increase of weight due to this accumulation of

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[5] *Records of the Geological Survey of India*, vol. 40, Pt. 4, pp. 320–333 (November, 1910), and *Economic Geology*, vol. 7, No. 1, p. 91 (January, 1912).

successive layers will cause a progressive compression to take place. The effect of compression on a mass of practically incompressible mineral particles imbedded in water is to draw these particles closer together, to reduce the space available between them, and to squeeze out more or less of the liquid contents. The limit of compression is reached when the particles come in contact. For this reason, the result would be somewhat different, according to the size of the component particles of the layers. Mud or ooze particles are exceedingly small. When, in the syncline, the lower layer of mud would be compressed, the resulting texture would be close, and the tendency would be for the fluids to be squeezed out to a great extent; whereas the grains of a layer of sand are much larger than the particles of clay, and such a layer may become incompressible when it still contains a great percentage of holes. It may thus act as a reservoir for liquids even under considerable pressure. The overlying sand stratum would then remain porous to a large extent, and the liquids, escaping from the underlying mud, would rise and become confined in it. This may be termed the *second phase* of the process, and is characterized by the progressive transfer of a large part of the oil and gas from the clayey layer in which they have originated to the overlying porous stratum in which they still remain distributed in a finely disseminated state. At this stage, the porous rock may be supposed to be saturated with an emulsion of oil and gas in water.

Finally, in the course of time, sedimentation is stopped in the syncline of deposition, and orogenic movement begins. Whatever may be the cause of this movement, it is a tendency toward a new adjustment which evinces itself by a lateral thrust. In the simpler form, one rim of the syncline remains immovable, and acts as a resistance, and the other rim is brought nearer to the first one by a movement tangential to the crust, which squeezes the syncline as between the jaws of a vise. The strata are bent and lifted above the level of the waters.

It is possible to get an idea of the order of magnitude of the thrust and to compare it with the forces due to buoyancy and hydraulic pressure invoked in the preceding theories, and which we have found to be insufficient to explain the migration of the oil. For this, we have to consider that this new force exceeds the resistance to crushing offered by the most resistant sedimentary rocks. This crushing strength becomes thus an inferior limit to the force. Limestone, for instance, may be crushed under a pressure of between 400 to 1,200 kg. per square centimeter; so the thrust is to be measured at least in hundred kilograms to the square centimeter. We have already seen that the force of buoyancy, by which a cubic centimeter of oil is supposed to be lifted along a dip in the anticlinal theory, is to be measured in fractions of a gram (in milligrams for a dip of 1 ft. for 50 ft.), and hydraulic pressure due to underground waters, with the velocity admitted, would be still less important. The magnitude

of the force we are considering now—the tangential thrust—is then at least one million times greater per unit surface than the two other forces previously considered. Further, this force has left its imprint in all sedimentary strata, not only by the flexure of the strata or their relative displacement, but frequently by dynamo-metamorphic actions which it has brought about. These actions have taken place between the time the strata were lying at the bottom of the waters and the time they have acquired their present situation. If then, any oil was ever existing at the

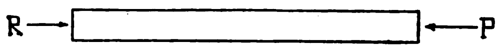


FIG. 2.

time of the deposition of the strata and has remained in some stratum since, it must equally have been subjected to the action of the thrust. The *third phase* of the process begins with this action of the thrust. But this needs a more complete analysis of facts.

When a square prism of a solid material, such as a piece of wood or an iron bar (Fig. 2), disposed horizontally on a support in order to avoid the action of gravity, and propped against a resistance $R$ at one end, is com-

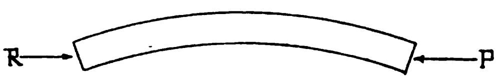


FIG. 3.

pressed at the other end by a force $P$, in the direction of its axis, the compression is instantaneously transmitted to the resistance, from end to end. Then, if the pressure is sufficiently increased, the prism will bend (Fig. 3). This flexion will extend the fibers on the convex side and contract them on the concave one, and there will be an intermediate layer which will be neither contracted nor extended by the flexion. This

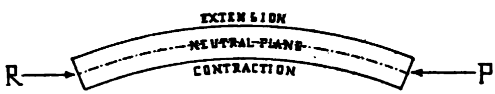


FIG. 4.

is the neutral plane (Fig. 4). Superimposed to this first effect, there will be a general contraction along the axis, which will affect the whole prism.

If, instead of a prism of solid material, we operate on a prism made of some homogeneous plastic material, like moist clay, the result would not be exactly the same. The compression at one end would not be immediately transmitted at the other end, but only progressively, in proportion as the inertia of the successive sections of the prism and the friction against the supporting stand would be overcome, the one after the other. And defor-

mation would begin at the end where the thrust $P$ is applied, even before any compression would become noticeable at the other end, if the prism were long enough.

Let us now imagine a prism composed of a series of parallel, superposed, horizontal and homogeneous layers, individually uniform in thickness, but variable in composition and resistance, such as clayey, sandy and calcareous layers would be. Suppose such a prism submitted to a uniformly distributed load, representing the load of superincumbent strata; the friction against the stand reduced by the interposition of a soft base, and proper precautions taken to avoid lateral deformation. And let us subject this prism to the same process of compression. Here again the compression would be transmitted progressively from the point of applied pressure to the point of applied resistance, and deformation would begin at the extremity nearer to the thrust. This deformation may vary, according to the relative thickness and disposition of the "competent" and "incompetent" strata, as they have been termed, but would generally take the form of a local bend, some sort of anticline, which may be overthrown in the direction of the resistance. If now

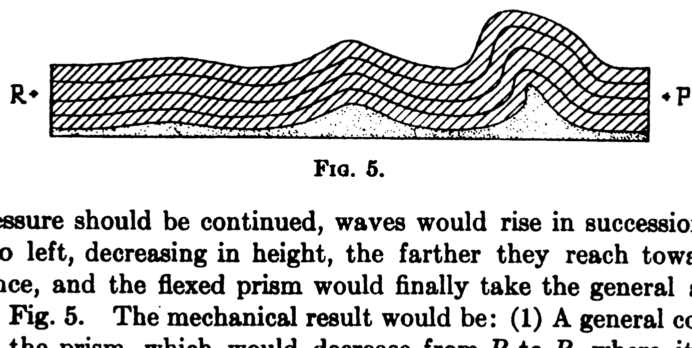


Fig. 5.

the pressure should be continued, waves would rise in succession, from right to left, decreasing in height, the farther they reach toward the resistance, and the flexed prism would finally take the general appearance of Fig. 5. The mechanical result would be: (1) A general compression of the prism, which would decrease from $P$ to $R$, where it might disappear, and (2) deformations by bending, with appressed folds toward $P$ and lower anticlines toward $R$; the folds being generally overthrown in the direction of the resistance $R$.

Now, we have already seen that when a material is compressible, compression will tend to draw nearer to each other the solid particles of which the material is composed. Compression would thus tend to a reduction of volume and to a reduction of porosity, or capacity for liquids. The first mechanical effect of our experiment, then, tends to reduce the capacity for liquids of the material at a decreasing rate from $P$ to $R$, from the point of applied pressure to the point of applied resistance.

To this would be superimposed the mechanical effect due to flexure. Anticlines would be extended on their convex side and synclines compressed on their concave side. If, then, the thickness of the prism is sufficient, and if we consider only its upper portion, above the neutral

plane, the general effect would be as follows: A general decrease of compression and a general increase of capacity for liquids from $P$ to $R$, superimposed on a wavy succession of local increases and decreases of capacity. This would create a succession of zones of increasing capacity from right to left (from $P$ to $R$), individually elongated in a transverse direction to the prism, or in a parallel direction to the folds. If then the porous strata were saturated with a liquid, the tendency would be for the liquid to move from the more compressed parts to the less compressed or decompressed ones. This movement would take place along the lines of lesser resistance; viz., (*a*) from the clayey layers to the sand strata, which would complete the concentration of the oil emulsion, already begun by the effect of vertical compression, into the sandy layers, and (*b*) along the sand strata themselves. Pools would be constituted parallel to the flexures, which would become more important the nearer they approached the resistance, or increasing from right to left.

Further, compression and consequent migration of the emulsion through the sand would have a tendency to allow oil drops and gas bubbles to coalesce. Temperature may play a part in the phenomenon, as pressure would generate heat. This heat may be negligible in the zone of reduced compression, but may be important in the highly compressed one. Rocks may afford sometimes a certain amount of heat without noticeable changes; but organic liquids would be more or less modified. A slight increase of temperature would lower the viscosity of the oil and facilitate its migration. A sufficient rise may partially decompose it, and complex reactions may occur if this decomposition should take place under pressure, in the presence of water vapor. This side of the question will have to be investigated more completely, should the general theory be admitted. For there could be found possibly a partial origin of the different grades of oil, and the source of a large part of the gas which accompanies oil in the pools today. The complexity of the products would still be increased by the filtration of the oil through certain rocks, which an excessive compression may allow, or which capillarity may induce in other circumstances.

At any rate, gas and oil concentration in the zones of decompression or of reduced compression would result; and water, oil, and gas may finally settle in the reservoirs according to density.

On the other hand, we have seen that the anticlines would be extended along their crests and the synclines contracted along their bottoms, if we consider only the part of the prism above its neutral plane. The liquid would thus be squeezed out from the synclines and would have a natural tendency to collect, by the effect of compression, in the higher and more open parts of the anticlines. Later, if, for some reason, water were reduced in amount, the oil would follow the level of the receding waters and sink toward the bottom of the synclines, as long as porosity would permit.

It might occur, with a sufficient increase of the lateral thrust, that the material subjected to compression would break instead of bend, and that overthrusts and faults would result. These deformations would naturally take place in the more compressed region, on the side of the thrust. This would open a way of escape to the gas and to the liquids, and the affected parts of the oil-bearing strata might be deprived of their contents, by outflow or evaporation. The oil and gas would be lost and these parts of the layers would become dry.

As far as we have gone, we have only considered theoretical views and results of laboratory experiments. The material which we have been handling was simple, the individual layers homogeneous, of constant thickness and horizontally disposed; the forces were acting in the vertical plane of symmetry of the prism, etc. And from a starting point fraught with geometric simplicity we have reached simple and geometric results. The merit of this way of proceeding is that it has allowed us to understand more clearly than could be done otherwise, the trend of events when the elements of the problem are reduced to their simpler lines.

But in Nature, things are much more complicated. Let us consider, instead of a prism, a syncline of deposition and note the new set of conditions which may affect the result. Between the geometric dispositions of our previous scheme and the dispositions encountered in the field, we will remark at once the following differences:

The material will not be homogeneous, but varied to the extreme. Not only will the strata be different from one another in composition, texture, resistance, flexibility, etc., creating a lack of homogeneity in the vertical direction, but similar differences will be found in individual strata, so that the lack of homogeneity will become almost general.

The materials will not be geometrically disposed. The outlines of the compressed area, in horizontal projection, will be irregular, and its thickness at great variance from point to point. The layers will be incurved, presenting original bends of deposition. Their own thickness will be irregular, and the so-called "planes of stratification" will not be planes at all, but complicated surfaces.

If we consider one particular stratum, the pressure due to overlying formations will not be the same everywhere.

The thrust will be irregular, and to simple compression and flexion will be superimposed a torsional movement. Consecutive thrusts, varied in direction and importance may affect the same region. Further, an orogenic movement does not result simply in a series of more or less parallel folds, elongated in one direction; it is complicated by transverse or orthogonal undulations, which create successive elevated and depressed areas along the general strike of the folds; the anticlines have a tendency to be transformed into elongated domes, the synclines into elongated basins.

Finally, breaks of greater or lesser magnitude frequently occur in the folded masses, and folds and overthrusts take place.

For these reasons, the resulting structure of a syncline subjected to orogenic movement becomes sometimes very intricate. Further, the superposition to the simpler effect of lateral compression of a series of incidental effects, derived from irregular forces which enter into play, would put the problem out of reach of mathematical treatment.

But there remains a way open to us. We may consider such a deformed syncline in nature, and try to ascertain how much the facts collected in the field may agree with the general lines of our theoretical views.

A good example of a folded chain may be found in the mountain system of the Appalachian province, which extends from New York to Alabama with a general northeast-southwest trend, and in the immediate neighborhood of which some of the most productive oil fields of the United States are situated. The thrust is supposed here to have taken place from the southeast. The whole Paleozoic series of strata to the floor of the crystalline Archean rocks—in some parts 40,000 ft. thick—have been involved in the system of flexure. The flexures are generally parallel to the main direction of the chain, and tend to be arranged *en échelon* in overlapping series. They are mostly unsymmetrical, *i.e.*, they bear a "rear and front structure," the steeper side generally facing northwest, away from the Atlantic Ocean. Bailey Willis, in his *Mechanics of the Appalachian Structure* recognizes four districts in the Appalachian belt. The eastern portion of the system is a district of "close folding," extending nearly its entire length and terminated at the south by a district of "folding with cleavage." The western part of the belt, on the north, through central Pennsylvania and the Virginias, is a district of "open folding." South of it, and always on the western side of the belt, comes a district of "folding and faulting," which passes through eastern Kentucky and Tennessee to Alabama.

The district of "open folding" terminates, on the west, in large undulations of low dip and merges finally into the monoclinal or gently undulating structure of northwestern Pennsylvania and eastern Ohio.

Beyond the southern region of "folding and faulting," proceeding westward, comes the high plateau or tableland of Kentucky and Tennessee.

As a whole, and barring irregularities of detail, we find here the characteristics which have been outlined in our theoretical statement: Successive parallel folds, more closely appressed on the east (the side of the thrust) and decreasing in importance the farther we come west (toward the resistance), with a general tendency for an overthrow of the folds in the direction of the resistance.

All the oil and gas fields are located along the western side of this

belt, on its outer margin; *i.e.*, on the side that is farther from the point of application of the thrust. From Pennsylvania down to Alabama, their main direction closely follows the direction of the chain, and individual pools are frequently disposed in rudely parallel rows, with their major axis parallel to the folds. This parallelism between the positions of the oil and gas areas and the trend of the mountains has long ago been pointed out, especially for western Pennsylvania.

Another remark is that the oil belt is especially developed in front of the region of "open folding," to the north, but rapidly decreases in importance as soon as it reaches the front of the region of "folding and faulting," to the south. The effect of faulting would be to relieve the strains and, in so doing, to reduce the amount of deformation farther west, as well as to open a way of escape to the fluids, if any. This double effect would reduce the tendency toward accumulation.

The Pennsylvanian oil belt offers a further interesting feature. When entering the State from the south, the mountain belt is at first bent eastward in its trend, and it takes again a northern course after crossing the Susquehanna. Fig. 6 represents this curve, which has the form of the letter *S*. Now, incurvations of this kind are frequently encountered in mountain chains, and they have been interpreted sometimes as the result of the resistance against orogenic movement due to an ancient mass of consolidated rocks (*Massif d'ancienne consolidation*) in the region *M*, acting as a pier which would deflect the direction of the thrust. The mechanical effect of such a deflection would be an increased pressure in the concave part of the curve, toward *A*, and a decompression in the region *B*, where the curve becomes convex. The fact that the principal oil areas of Pennsylvania and southwestern New York—which are among the most productive in existence—are located precisely in this zone of decompression and do not extend farther east in the compressed zone, is interesting.

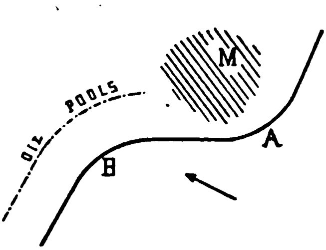


FIG. 6.

The same relations between orogenic deformations and oil-producing areas may be observed in other fields, as, for instance, in the oil fields of central Europe. The chain of the Alps, followed by the Carpathian Mountains, the Transylvanian Alps and the Balkans, are interpreted today as a system of folded chains. The belt develops from southern France to Galicia, Bukowina, and northern Roumania in the form of an arc, with its convexity directed northward. Its eastern extremity presents the well-known sigmoidal inflexion formed by the Transylvanian Mountains and the Balkans. The thrust has been directed from the inside of the curve toward the outside. Everywhere, the oil fields are

located on the exterior slopes of the belt; *i.e.*, on the side that is farther from the point of application of the thrust, and the pools are distributed in rows parallel to the trend of the folds, frequently with their major axis in the same direction.

In these districts the deformations are no more the gentle anticlines and synclines or the terraces of the Appalachian region; the strata are highly disturbed, contorted and even faulted. The thrust in the Alpine belt has been much more powerful than in the Appalachian region, and in certain places overthrusts of considerable magnitude have taken place. In fact, the Alps proper and the Carpathians seem to have been submitted to folding and deformation at least at two different periods: First immediately after Oligocene times, and second during the middle Miocene; and the present state of affairs is the result of the cumulative effect of both thrusts. It seems reasonable to admit that the region has passed through a first stage similar in some respects to the one presented by the Appalachian belt, and during which the greater part of the accumulation may have been effected, before reaching the further stage of greater disturbance, in which a part only of the accumulated hydrocarbons seems to have been preserved, in consequence of special stratigraphic conditions.

Similar relations may be observed in many other fields.

The preceding remarks have led the writer to believe that the facts observed in the field agree with the theoretical views previously expressed, and the hypothesis which he proposes for oil and gas accumulation may be summarized as follows:

*First Stage.*—The oil proceeding from organic remains, perhaps still in process of decomposition at the very origin of the movement, is at first distributed in the water-laden sediments of the geosyncline of deposition in the state of disseminated particles.

*Second Stage.*—The increasing compression, due to the continuous accumulation of superimposed strata, expels an increasing amount of the water of deposition with its contents of hydrocarbons, from the original layers, which, at the same time, are the most easily affected by the compression (argillaceous or limy sediments), to some other layers less affected by it (especially sands and gravels). This displacement may take an upward or a downward trend, the only condition being that the fluids must move along the line of lesser resistance, from a more compressed to a less compressed zone. This movement may take place between two entirely different strata, or between two layers of the same group of deposits, provided the layer acting as a temporary reservoir is less compressible or more porous than the former one. (This latter condition obtains in the dolomitic layers of the Trenton rock.)

*Third Stage.*—As soon as orogenic movement begins, a more or less horizontal compression, due to the thrust, takes place and becomes added to the vertical pressure due to superincumbent weight. The result of the

intervention of this new force is to create: first, a general increase of compression from the point of applied resistance to the point of applied lateral pressure; second, successive and parallel zones alternately compressed and decompressed, whose strike is normal to the direction of the thrust. The waters which saturate the strata are submitted to the effect of this unequal pressure and move from the highly compressed regions to the lesser compressed ones, carrying the hydrocarbons with them in their course, finally collecting in pools parallel to the folds. The movement would have to take place along the lines of lesser resistance, *i.e.*, toward and along the more porous layers of the formation (sandy layers, etc.). Pressure would reduce the viscosity of the oil, favor the coalescence of the globules and perhaps induce some chemical changes of the hydrocarbons. The more probable places of accumulation would be the crest of anticlines, the summit of domes, the rims of terraces, or, in the main, the places where a change occurs in the dip or along the strike of the strata, in the form of convex edges or arches; for the reason that at these places the local reduction of compression, buoyancy, and resistance to motion due to a change in the direction of flow, would act together and accumulate their effect.

When the fluids reach the zones of lesser compression, if the physical condition of the reservoir in which they collect is such as to hold the hydrocarbons and prevent their escape, an equilibrium is established, and the final pressure in the pools must be equal to the original pressure less the losses of head encountered on the way.

Further, a progressive settlement would take place in the reservoir, according to gravity; water would congregate at the lower places, oil would have a tendency to collect at its surface, and gas—either brought with the water or dissolved in the oil and further released by decompression, or simply produced from the oil itself—would reach the higher places.

Considered as a whole, the process would be a consequence of the mechanical principle of least action. In this process, the agent of transportation of the hydrocarbons would be water; the moving force would be hydraulic pressure created by vertical and lateral compression; and the extent of the movement would be variable in the extreme according to local conditions of the strata. But compression, especially compression due to the lateral thrust, whose action would be irreglar and continuous and would have to be extended over a long period of time, would not act as the permanent head of water which nowadays is the ultimate source of the velocity and of the hydraulic power of circulating underground waters. Frictional resistance may reduce and even stop the motion of underground water once for all in a given direction; a thrust would act by a succession of jerks and repeat the effort again and again. There would be periods of activity, during which the compressive force would exceed the resistance of the rocks and deformations would

ensue, followed by periods of rest, brought by the momentary relaxation of strains due to deformation itself and during which the compressive forces would accumulate; and the thrust would become the source of a periodically renewed energy. The consequent hydraulic pressure would follow a similar wavy movement, with periods of maxima, to which a maximum of velocity would correspond, and periods of minima or of rest, where frictional resistance might bring the movement to a stop. The action of the liquids would thus become similar to that of a water ram, with a maximum of efficiency periodically renewed.

*Fourth Stage.*—A new stage will be reached by the gradual reduction of the water contents of the strata, producing consequent changes in the level of complete saturation and in the local disposition of the pools, by gravity.

*Fifth Stage.*—Sometimes, a new period of folding may take place, in which the thrust may have or may not have the same direction as the previous one. New zones of compression and decompression may be created, and the liquids may be put again in motion. The results may become thus very intricate, especially if the strata are deformed to a large extent.

All gradations must be expected to be found in the oil fields between these two extreme sets of conditions. The Appalachian belt may represent the first set, which stops at the fourth stage of our description. The fields of central Europe would represent the further and more complex stage.

The research for oil becomes thus a problem of tectonic effects as well as of stratigraphy.

### *Rock Pressure*

The origin of "rock pressure" has been traced to one of the following causes: Hydrostatic pressure, weight of superincumbent strata, gradual accumulation of the inclosed gas, capillary diffusion.

The hypothesis of hydrostatic pressure is untenable as a general one. Hydrostatic pressure would agree, to some extent, with the pressure found in Ohio and Indiana, but could not account for the heavy pressures encountered in western Pennsylvania, as shown by Prof. J. P. Lesley and J. F. Carll, nor for the pressure of most of the deep West Virginia wells. Further, this theory is contradicted by the well-known fact that flow and pressure are found to decrease in any given well with the age of the well. A certain amount of constancy in the flow and in the pressure ought to be expected under artesian action, which is not the case.

The weight of overlying strata, under *present* conditions of the rocks would be mechanically inadequate.

The progressive accumulation of the gas may be a cause of pressure, and, according to David T. Day's experiments, capillary diffusion through

Fuller's earth bears a curious analogy with osmotic phenomena and the pressure due to this cause may be compared with osmotic pressure. But, if both theories may explain the origin of a certain pressure, they are not entirely satisfactory, in the writer's mind, for the following reasons: The progressive accumulation of the gas would rapidly be checked by the increasing pressure itself, and it has not been shown that such a limitation would not occur before reaching the high pressures encountered in some wells. On the other hand, the action of capillary diffusion seems to require certain physical conditions which are not met everywhere, and therefore it can not be admitted as a general cause. Further, none of the preceding theories explain the following facts: At least in the eastern fields of the United States, the rock pressure, in the main, increases with the depth of the "sand," and, at the same time, there seems to be a decrease of pressure with an increase of distance from the principal axis of the folding. The closed pressure in the Trenton limestone of Ohio and Indiana averages 200 to 300 lb. per square inch and only exceptionally reaches over 600 lb.; whereas the pressures in western Pennsylvania and in West Virginia, farther east, easily reach the double figure. In other words, it seems as if the pressure would increase, as a whole, in the same direction as the compression to which the rocks have been submitted at the time of their folding, both vertically and horizontally. A partial origin of the rock pressure, at least, would thus have to be traced to orogenic deformation. The two other forces—due to progressive gas increase and capillary diffusion—may have, and possibly have played a more or less important part in the final result, but this effect would have been produced later, and, in this respect, is to be considered as subordinate.

### *How Capillary Pressure Seals an Oil Pool*

One of the most interesting problems involved in the study of oil and gas accumulation is the process by which gas or oil may accumulate in the pay streaks under heavy pressure, without this pressure being dissipated through neighboring rocks or through the sandstones. Imperviousness of the superincumbent strata and of the oil-bearing bed itself has been frequently advocated. But unaltered rocks of the type encountered in oil fields are never impervious. From 10 to 40 per cent. of their bulk is made of pore space, and the pore space of the inclosing beds of shales averages no less than 6 per cent., which means that every square foot of so-called impervious rocks contains an average of 8½ in. of holes (M. J. Munn).

In his "hydraulic theory," M. J. Munn has suggested[6] that "the pools of oil and gas are held in place by water under hydraulic and capil-

[6] The Anticlinal and Hydraulic Theories of Oil and Gas Accumulation, *Economic Geology*, vol. 4, No. 6, pp. 523 and 527.

lary pressure which effectively seals up the pores of the surrounding rock, and prevents the dissipation of pressure by diffusion." "Pressure in pools," he writes, "is maintained by the expansive force exerted by gas. . . . Such gas could not diffuse because of the saturated conditions of the surrounding rocks." But Mr. Munn does not go further than a general statement, and no detailed explanation of the process by which such a "sealing up" is rendered possible has yet been given, as far as the writer knows. The following is proposed as a tentative explanation.

Hydraulic pressure is here discarded, this problem being a problem of statics rather than of dynamics, and in which the velocity is naught.

There is a peculiar and interesting phenomenon which appears to have escaped the notice of those who have tried to explain the "sealing up" of a pool. It has been pointed out by Jamin, a French physicist, and is illustrated by the experiment of "Jamin's tubes." If a capillary tube is incompletely filled by water and the water distributed through the tube in such a way as to constitute a string of droplets, a pressure may be applied to one of the extremities of the tube which will not be transmitted to the other end. In other words, the string of droplets will act as a resistance. If there is a large number of drops in the tube, the difference of pressure at the ends arising in this way may amount to several atmospheres. The explanation follows: A drop of liquid, like water or oil, which does not wet the tube, will be limited on both ends by a meniscus (Fig. 7). The superficial tension which results in this form is caused by the tension of glass and air, glass and liquid, and air and liquid. The two first sets of forces are parallel to the axis of the tube, and, being equal and directly opposed two by two at both ends of the drop, neutralize each other when conditions of equilibrium are formulated. The third set of forces, caused by the tension of air and liquid, is tangent to the meniscus all around the tube, at both ends, as shown in the figure, and makes with the axis of the tube an angle $\theta$ which is the "angle of contact." The weight of the drop may be neglected here, as its action is insignificant with regard to that of surface tension. When the pressure is equal at both ends of the drop, the meniscuses are identical, and so are the angles of contact $\theta$ and $\theta'$. But if the pressure $P$ at one end is increased, both meniscuses will alter their curvature. The meniscus in front of $P$ will decrease its angle $\theta$, which will tend toward zero, whereas the meniscus opposed to $P'$ will increase its angle $\theta'$, which will tend toward 90°. The result of this deformation is to increase the force $T \cos \theta$ directed against $P$, and to decrease the force $T \cos \theta'$ directed with it. The difference will be a resistance against motion expressed by $T(\cos \theta - \cos \theta')$, which will draw closer to the limit $T$ as the pressure is increased.

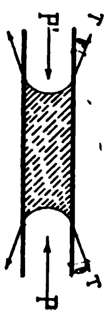

FIG. 7.

The same conditions would obtain in shales or clays capping a gas pool. There is nothing like a plane of separation between the gas of the pool and the water that fills the pores of the superincumbent rock; but there is a more or less irregular intermediate zone in which the gas and the water are commingled. The pores of the shale, by their juxtaposition, would constitute the Jamin tubes of the experiment, and these would be filled by the mixture, which would take the form of bubbles of gas intermingled with droplets of water.

Van Hise remarks[7] that "the majority of the particles of most clays, shales and slates are much smaller than 0.0012 mm., and therefore the openings of the rocks are subcapillary." He defines subcapillary openings as those which are 0.0002 mm. and less in diameter. Starting from these data, it is possible to calculate, with a sufficient degree of approximation, the thickness of the shales or of the clay which would seal up a given pressure. The writer has found that a few feet would be amply sufficient to seal up a pressure of 1,200 lb. per square inch.

It may be conceived that the same process would apply for the sealing of gas and oil pools laterally, along the dip and strike of the porous layers themselves, wherever the sealing is not already produced by changes in the nature of the "sand" from a pervious to an impervious one, or by the presence of water. Pore openings of sandstones are, for the most part, capillary; and should water not be present in the sand, oil may replace it for the Jamin tube effect. Calculation shows that the marginal zone thus constituted may reach a width of a few hundred feet. Pressure would decrease in this zone from the inside to the outside of the pool, progressively, a feature which is readily observed in the field.

[7] A Treatise on Metamorphism, *Monograph*, 47, *U. S. Geological Survey*, p. 138 (1904).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Modern Methods of Mining and Ventilating Thick Pitching Beds

BY H. M. CRANKSHAW,* B. S., HAZLETON, PA.

(Arizona Meeting, September, 1916)

THE early methods of mining anthracite in the steep pitching Mammoth bed consisted in driving breasts up the pitch from the gangways and airways driven in the bed along the strike (Plate 2, Fig. 1). Breasts are simply rooms driven up the pitch the full thickness of the bed (Plate 2, Fig. 2), having a width of from 18 to 30 ft., and a manway or traveling way on one or both sides. (For detail of breast method see Whildin, *Trans.*, vol. 50, p. 704).

Many difficulties were encountered, and the maintenance cost of the timbered gangways was so great as to be practically prohibitive. To reduce this expense, the gangways were driven in the thin Skidmore bed underlying the Mammoth, and the coal mined through rock holes. (Plate 2, Fig. 2, *A*.)

It was found that after the breasts were driven up 80 to 100 ft., the coal almost invariably rushed or "ran away," making good ventilation almost impossible and the extraction of coal from the upper portion of the lift very difficult. The economic length of a lift is from 200 to 250 ft., and the percentage of extraction by the old method above mentioned was often under 50 per cent., while the coal recovered was so badly broken up as to reduce materially the percentage of large sizes.

## *Principles of Present Mining Methods*

The present methods designed for the recovery of a maximum percentage of coal and a maximum yield of the prepared sizes depends largely on the following principles:

*First.*—Formerly the practice in mining bituminous coal was to space rooms so that as much coal as possible could be extracted in first mining (Plate 3). The small pillars were often lost, and an improvement in total extraction was found in the system of driving narrow rooms with wide pillars, with a much larger final recovery (Plate 4).

*Second.*—The early method of ventilation was to maintain a single

* Manager, Harwood Coal Co.

current of air throughout the mine. This has been modified by systematically splitting the air and so establishing more efficient ventilation.

*Third.*—The coal should be so mined as to allow the overlying measures to break and relieve the pressure, the break to be controlled and regulated by the method of mining so that it occurs where it is desired. This principle, which governs longwall mining in England, is fully recognized by the Consolidation Coal Co., and was well brought out in Mr. Grady's paper,[1] but in the type of mining under discussion it has rarely been applied. It has been claimed also by some mining engineers that the principle is inapplicable to the anthracite region, as the tough overly-

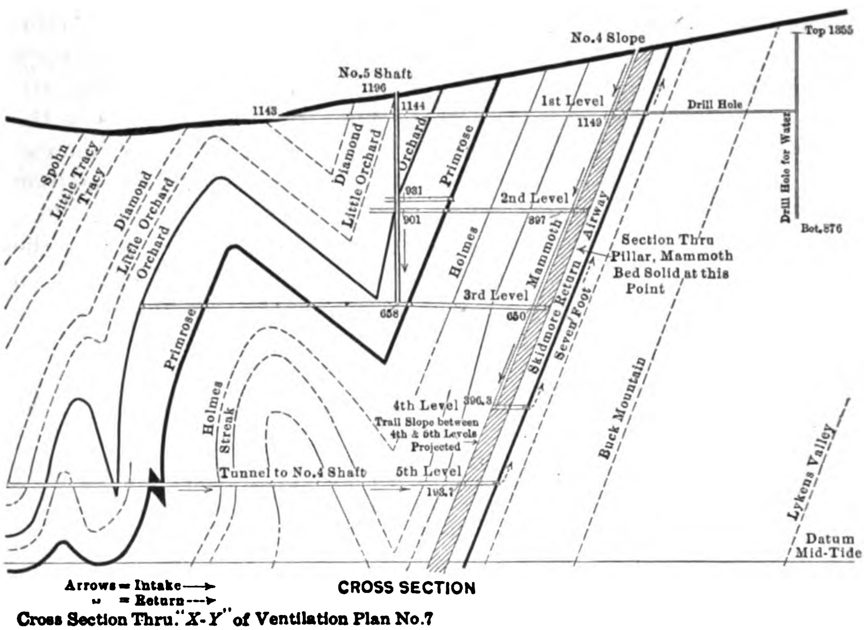


PLATE. 1.—METHOD OF MINING STEEP PITCHING MAMMOTH BED.
(Typical Section)

ing strata would not break; that this conclusion is unwarranted is indicated by the fact that in longwall bituminous coal mining it is not unusual for several acres of roof to stand before the first break occurs, after which the roof breaks without further trouble.

Therefore, we believe it has been proven that we must in first mining leave strong pillars; we must scientifically split the air to secure efficient ventilation; and we must so regulate the extraction of pillars that the roof breaks may be controlled.

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[1] *Trans.*, vol. 51, p. 138.

## *Disadvantages of Old Method*

In the old method of mining the Mammoth bed, the coal was first partly worked over, after which the ground squeezed and settled through a period of years until the coal left behind had been so consolidated by pressure as to again allow of chutes being driven in the squeezed bed, and

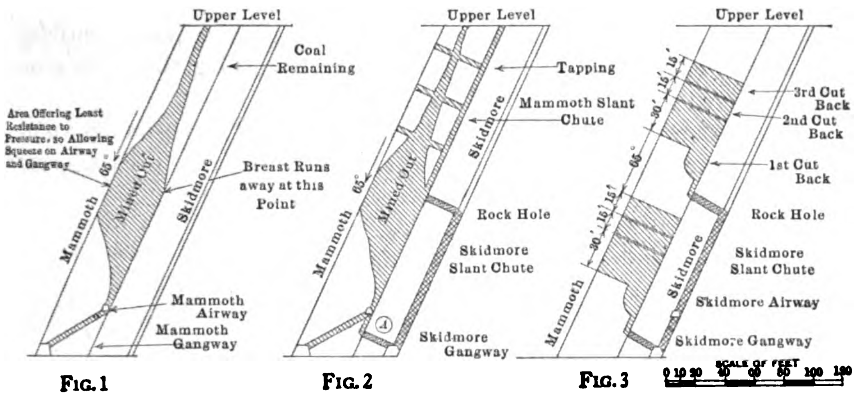


PLATE 2.—OLD AND NEW METHODS OF MINING MAMMOTH BED.

the original method of mining repeated, this process sometimes being gone over several times before the final extraction of the coal from the bed. The squeezing considerably shattered the coal and resulted in a much decreased percentage of the prepared sizes in the product of the second and subsequent minings.

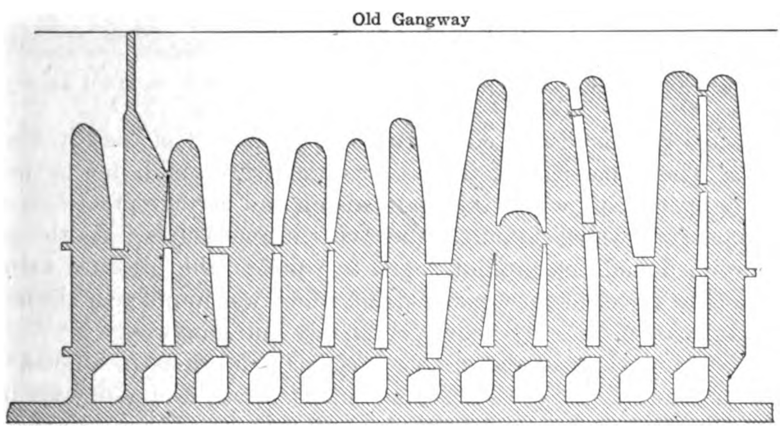


PLATE. 3.—SKETCH SHOWING PRIMITIVE METHOD OF MINING.

The modern methods aim to take out as much coal as possible in the first mining, and to avoid going over the same ground several times as was formerly necessary.

The method described is the result of 3 years of careful work and experiment, using at the same time all knowledge obtainable from the failures of the past (Plates 5 and 6). The paper by Mr. Whildin[2] described the methods developed for reworking mined-over ground. While doing this work constant studies were directed toward the development of a method which would be applicable to virgin ground, and which would overcome the defects of the old methods, the chief of which were: Low percentage of extraction; excessive cost of timbering; poor ventilation, and low percentage of prepared sizes, due to reworking the same ground after the coal had been crushed and squeezed.

It was found that after a breast had "run away" and been drawn until the rock from old workings appeared at the battery, the greater portion

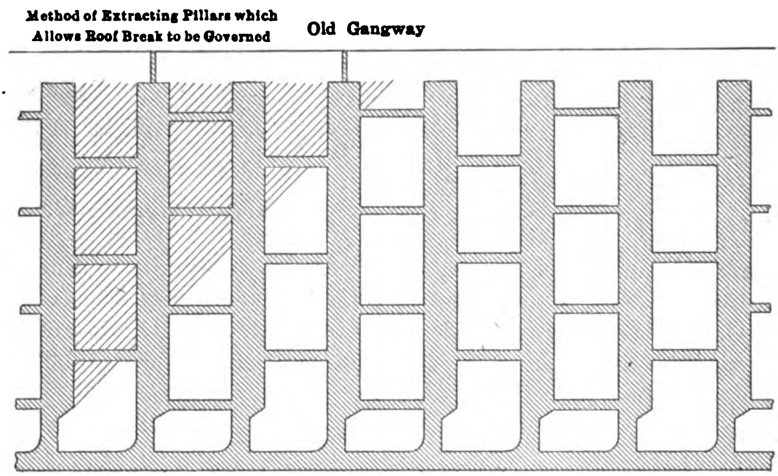


PLATE. 4.—SKETCH SHOWING MODERN METHOD OF MINING. SCALE 1 IN. = 15 FT.

of the coal recovered had come from the lower part of the lift, Plate 2, Fig. 1. The upper part of the lift was practically solid, due to the fact that the space from which the coal had run had become practically filled with loose rock which supported the overlying measures. As the pillars were robbed and the ground began to squeeze, the pressure naturally did the most havoc at the weakest point, which was found near the bottom of the breasts, or directly above the airway and gangway.

Since in most cases the mining was not done so as to allow a clean break of the roof, the pressure came on the airway and gangways in the form of a slow squeeze which broke the timber. This resulted in the conclusion that heavier timber was needed, and in many cases, sticks 24 to 30 in., and even 36 in. diameter were used for timbering purposes, resulting in an excessive timber cost.

[2] *Trans.*, vol. 50, p. 698.

A careful study of Plate 2, Fig. 1, will show why the cost of timbering in Mammoth airways and gangways is excessive, and how the squeeze was brought on by the old method.

To reduce this timber cost the gangways were driven in the Skidmore bed underlying, instead of in the Mammoth (Plate 2, Fig. 2), involving the use of rock chutes from the Skidmore and the driving of slant chutes in the latter. The result was the development of a complete method of working the Mammoth bed from the underlying beds (Plate 2, Fig. 3).

### *Description of Present Method*

In the working of a lift in the Mammoth bed 250 ft. long on a pitch of 65°, the bed being 40 ft. thick, the gangways and airways are first

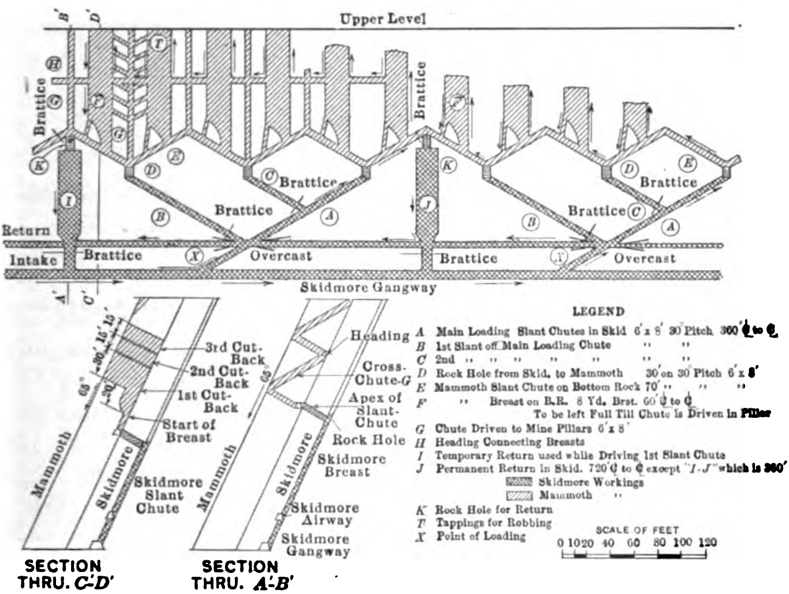


PLATE. 5.—PRESENT METHOD OF MINING, UPPER SECTION.

driven in the usual way in the underlying bed. Rock holes or chutes to the Mammoth from this gangway level are not driven until the workings in the underlying bed, following the plan to be described just below, have reached a point about halfway up the lift, Plate 5, with the result that the breasts in the Mammoth are restricted to a length of about 100 ft. The length of breasts was limited to 100 ft. because, as previously observed, past experience showed that the breasts almost invariably ran away when this length had been driven, making it practically impossible to carry to completion any system of working involving any greater length.

In the development of this plan, chutes *A*, *B*, *C*, Plate 5, are driven in the Skidmore bed, underlying the Mammoth, and rock holes *D* put through to the Mammoth 120 ft. apart, and from a point halfway up the lift. These rock holes are then connected by slant chutes *E* along the bottom rock of the Mammoth, from which the breasts are started on 60-ft. centers in the usual way. Each rock hole handles the coal from two breasts and the coal from three rock holes comes together at one loading chute at *X*.

It has been found desirable not to drive the rock chutes much in advance of the mining of the coal in the Mammoth, and to mine this coal as rapidly as possible. By this method after the work is once fully

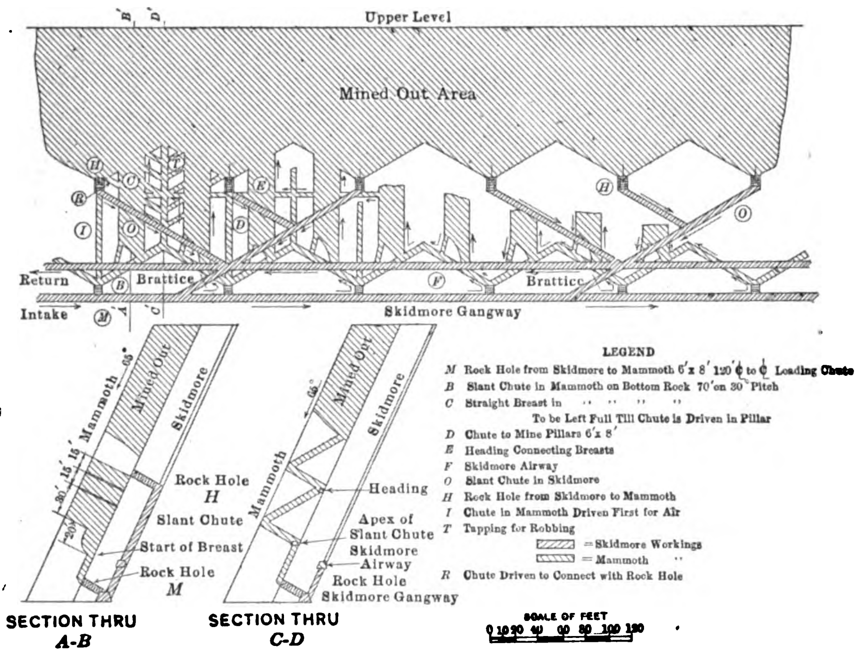


PLATE 6.—PRESENT METHOD OF MINING, LOWER SECTION.

opened out the cost of the rock work is only about 20 per cent. of the total cost of timbering under the former method of mining.

Each set of six breasts has a separate split of fresh air and a ventilating system of its own. The ventilation is arranged very simply, on the split system, by driving a Skidmore breast *I* and *J* from the Skidmore airway far enough up the pitch so that a rock hole *K* put through from the top of the breast will connect with the apex of the slant chutes *E*. The main loading chutes *A*, 360 ft. apart, then become the main intakes, and as shown by the arrows, the air returns through rock holes *K* and Skidmore breasts *J* to the Skidmore return airway.

After the breasts are driven through to the upper level, cross chutes *G* are driven across the bed from the apex of the slant chutes *E*, for the purpose of robbing the pillars. The breasts are allowed to stand full until the cross chutes *G* are driven through to the upper level, and are drawn as the cross chutes *G* are worked down, the pillars being completely robbed by means of the tapping chutes *T*.

In this way all the chutes are driven in solid coal, the men are in strong ground all the time, thus being well protected, and a good supply of fresh air is always available to dilute any outburst of gas.

When the upper half of the lift has been robbed back, the lower half is worked in a similar way from the rock holes *M*, Plate 6, driven direct from the Skidmore gangway. These rock holes are connected by slant chutes *B* driven along the bottom rock of the Mammoth, and the whole operation repeated as for the upper half of the lift. Working the upper half first, however, has given an opportunity to ventilate the lower half easily, as follows:

As soon as the first breast *C* is driven up, a short chute *R* is driven from near the top of the breast to connect with the rock holes *H*; the slant chutes *O*, in the Skidmore bed, which previously were used as coal chutes, now serve as return airways, with no expense for upkeep.

The main chutes, wherever possible, should be driven 6 by 8 ft. so as to provide a good-sized manway for traveling and for carrying up timber. In order to secure a reserve pillar for fire protection and for limiting possible squeezing a large pillar is left every 720 ft., the length of each double section.

This method was first tried on the west side of the fifth level of No. 4 Shaft, Lansford Colliery, where the upper half of the lift in the first part of the work is now being robbed down. The results as to cost and extraction have been excellent.

### *Ventilation System*

One result of the method of mining before described is to get all the main airways in strong permanent ground, so that they are not subjected to squeezing, and require a minimum expense for upkeep. Two mines producing 700,000 tons per year were working the section where the ventilation was remodeled, this work taking 2 years to complete. Two electrically driven Sirocco fans were installed, of 100,000 and 150,000 cu. ft. capacity, respectively, the larger one operated blowing and the smaller exhausting.

The mines were originally ventilated by fans located in the center of the property in the usual way (Plate 7). The air traveled along the main haulageways as intake airways and returned along return airways parallel to the main haulageways. As the mining concentrated at the

extremities of the property, the following defects became apparent, resulting in unsatisfactory quality of air in the working places.

To get to the fans the air had to travel 2 miles; the main haulageways were so filled with traffic as the mines increased in production that their function as an intake airway was materially impaired; the return airways, where they were in the Mammoth bed, were exceedingly expensive to maintain, and where they were in the Skidmore, in many cases were too small to carry, for the required long distances, the volume of air needed.

By replacing the central system of ventilation with fans located near the extremities, the length of travel of the air was halved, and it was found practicable to abandon sections of the Mammoth airway which were expensive to maintain. In this way, an efficient method of ventilation was worked out, which made it possible to work coal which had been inaccessible for many years.

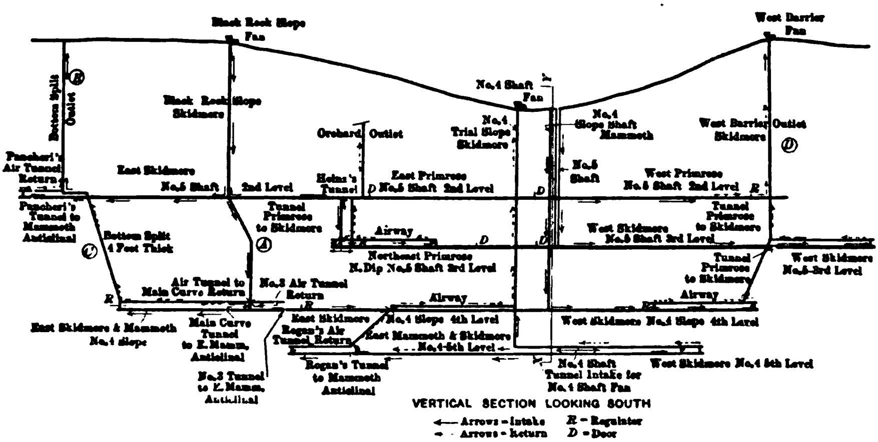


PLATE. 7.—VENTILATING SYSTEM.

The diagram, Plate 7, shows the ventilating system employed in these mines, the central fan, by exhaust, ventilates the fifth or lower level, which is known as No. 4 Shaft level, and the "Black Rock Slope" force fan ventilates the second, third and fourth level on the east side, and the "West Barrier" exhaust fan ventilates the second, third and fourth levels on the west side.

By making the "Black Rock Slope" a blowing fan, it was possible to ventilate the section known as Pancheries' Tunnel, and the section known as the N. E. Primrose, without the necessity of having long return airways, the haulageways acting as intake airways for these sections.

When the three fans are in operation, the territory is so divided that there are a proportionate number of men to each fan, and the main air current is in each case divided into at least three main splits, each main split being again split up in the working places as required.

To date the adoption of the mining and ventilation methods described have given the following results:

In 1915, up to Oct. 30, 50,000 tons were mined by the new method, the labor cost being 19 c. per car lower than by the old method under similar conditions, and the saving in cost of timber at least 30 per cent.

While reliable figures as to extraction will not be available for about 2 years it is reasonable to believe that as the coal is "mined" instead of being allowed to run, less rock will be extracted, and the total car yield will be materially increased.

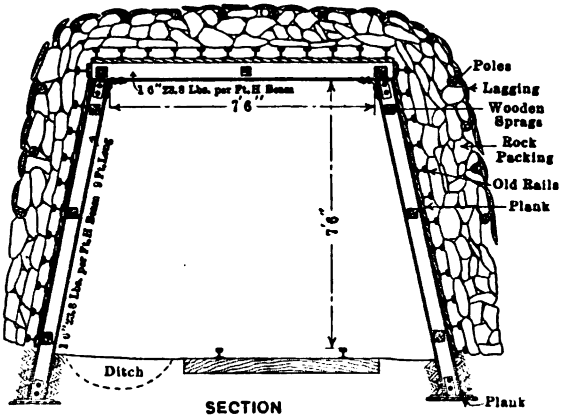


PLATE. 8.—METHOD OF ERECTING STEEL TIMBER IN MAMMOTH BED.

### *Use of Steel Timber*

The use of steel timber is only commercially justified in gangways in the Mammoth bed when such gangways are not subject to squeeze due to mining operations in their immediate vicinity; but all gangways, turnouts and other places in the Mammoth bed, which are required for use over a period of many years, should be lined with steel timber properly erected.

For general use, it has been found advisable to use H-section steel for both legs and collars; 5- and 6-in. H-beams have been found most suitable for sets in gangways, with sizes for turnouts and special work as required by the conditions.

The method of erection is shown in Plate 8.

Figs. 9 and 10 illustrate typical examples of this method of timbering. In the gangways shown, the Mammoth bed is 60 ft. thick, very dry and very liable to run, the gangway being 1,000 ft. below the surface. The best of wooden timber rots out in 2 years.

Fig. 9 shows the gangway timbered up to the rock lying between the

PLATE. 9.—GANGWAY IN WHICH STEEL "TIMBER" IS USED.

PLATE 10.—WOOD BEING REPLACED BY STEEL FOR GANGWAY SUPPORTS.

Mammoth and Skidmore beds. The tunnel was afterward driven without any of the timber being disturbed by the blasting of the rock.

Fig. 10 is an example of the replacement of wood by steel. Note the comparative size of wood and steel.

The author desires to express his appreciation of the assistance given by H. D. Kynor, John L. Richards and A. L. Walbridge in carrying out the work described and in preparing this paper.

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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Diesel Engines versus Steam Turbines for Mine Power Plants

BY HERBERT HAAS,* SAN FRANCISCO, CAL.

(Arizona Meeting, September, 1916)

CHEAP power is essential to large-scale mining and metallurgical operations, particularly where fine grinding of large tonnages has to be resorted to, as is the case with an increasing number of mines treating disseminated copper ores, or low-grade gold ores, in the grinding of which considerable power is consumed. A comparison of the cost of generating power with Diesel engines and steam turbines and a brief discussion of the different factors governing the profitable use of either type of prime mover may be timely in view of the importance of a cheap supply of power to the mining industry, and the prominence that has lately been given to Diesel engines in America.

As many of our most important mines and largest producers of metals are located in sections where neither cheap coal nor water power is available, and the price of oil fuel is increased by long hauls, the Diesel engine will become a prime mover of increasing importance. This applies particularly to the Southwestern States, where an ample supply of cold water for condensing purposes is unusual and local climatological conditions interfere in securing the high vacua essential to high turbine efficiency.

About 12 times as much water will be required for condensing purposes with steam turbines under general conditions as is needed for jacket-cooling of two-stroke cycle Diesel engine; four-stroke cycle Diesel engines require only one-twentieth of the cooling water.

Another condition favoring the use of Diesel engines is the high load factor at which most mine power plants operate. The influence of load factor on the cost of generating power is shown in Tables 1 and 2 and in Fig. 1.

* Consulting Engineer.

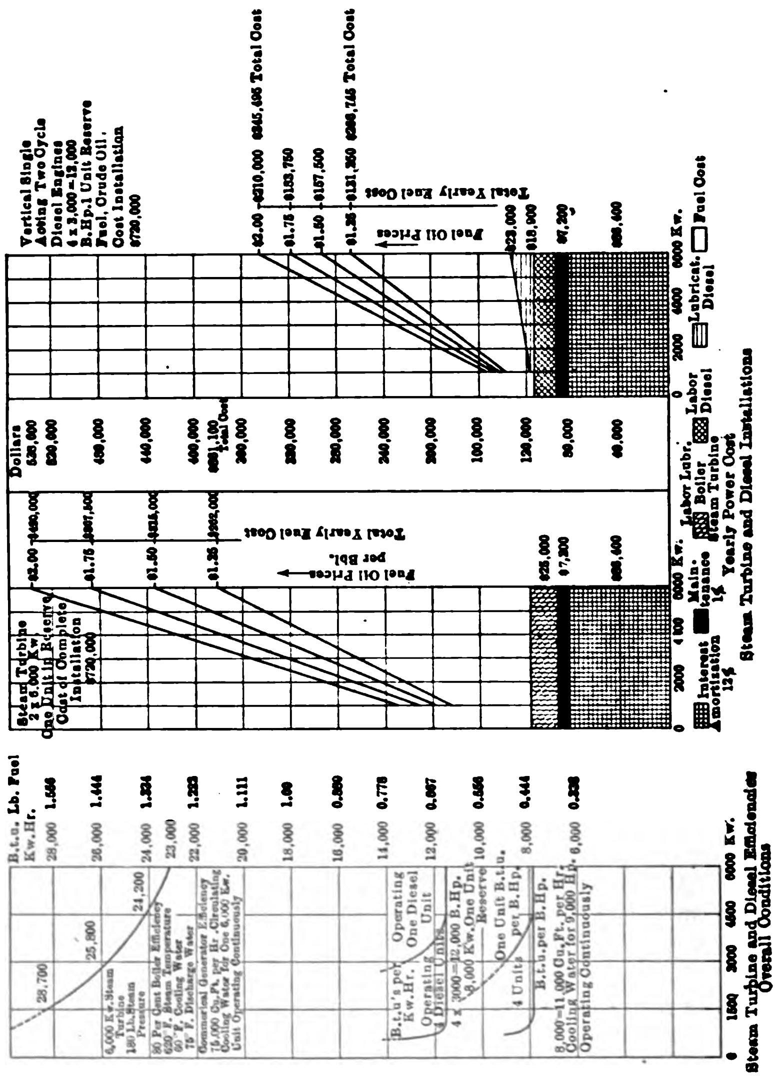


FIG. 1.—COMPARISON OF STEAM TURBINE AND DIESEL ENGINE EFFICIENCIES AND COSTS.

TABLE 1.—*Cost of Generating Power with Four* 3,000 *b.hp.* (2,000 *Kw.*) *Sulzer Diesel Engines Direct-Connected to Alternators and Exciters*

Costs are based on the following conditions:
Cost of complete plant .......... $720,000
Cost of fuel, $1.25 per bbl. of 320 lb.
Cost of lubricating oil, 35 c. per gallon.

| | | | Per Month | |
|---|---|---|---|---|
| Labor— | 1 chief engineer | at $200 | $200 | |
| | 3 watch engineers | at $150 | 450 | |
| | 3 switch-board men | at $125 | 375 | |
| | 3 helpers and oilers | at $ 90 | 270 | |
| | 2 machinists | at $140 | 280 | |
| | 12 men, total per month and year | | $1,575 | $18,900 |
| Maintenance—1 per cent. of cost of installation, per year | | | | 7,200 |
| Interest, at 6 per cent. per annum | | | | 43,200 |
| Amortization at 6 per cent. and re-invested at 4½ per cent. to redeem capital of $720,000 in 15 years | | | | 43,200 |

The kilowatt-year, figured at 8,760 kw.-hr. Four, three, two, or one unit operating, respectively, furnish full, ¾, ½ or ¼ load, generating 70 million, 52½ million, 35 million and 17½ million kw.-hr. per year respectively. Three units to operate continuously, one unit to serve as stand-by. Cooling water in circulation for 9,000 hp. operating continuously, 8,000 to 11,000 cu. ft. per hour without re-cooling arrangements.

| | 12,000 hp. or 8,000 kw. | | 9,000 hp. or 6,000 kw. | | 6,000 hp. or 4,000 kw. | | 3,000 hp. or 2,000 kw. | |
|---|---|---|---|---|---|---|---|---|
| Kw.-hr. per year | 70,000,000 | | 52,500,000 | | 35,000,000 | | 17,500,000 | |
| Barrels | 140,000 | | 105,000 | | 70,000 | | 35,000 | |
| | Cost per Year | | Cost per Year | | Cost per Year | | Cost per Year | |
| | Per Kw.-hr., | Per Kw.-hr., Cents | Per Kw.-hr., | Per Kw.-hr., Cents | Per Kw.-hr., | Per Kw.-hr., Cents | Per Kw.-hr., | Per Kw.-hr. Cents |
| Fuel cost | $175,000 | 0.2500 | $131,250 | 0.2500 | $ 87,500 | 0.2500 | $ 43,750 | 0.2500 |
| Labor | 18,900 | 0.0270 | 18,900 | 0.0360 | 18,900 | 0.0540 | 18,900 | 0.1080 |
| Lubrication | 30,660 | 0.0438 | 22,995 | 0.0438 | 15,330 | 0.0438 | 7,665 | 0.0438 |
| Maintenance | 7,200 | 0.0103 | 7,200 | 0.0137 | 7,200 | 0.0206 | 7,200 | 0.0412 |
| Total direct | $231,760 | 0.3311 | $180,345 | 0.3435 | $128,930 | 0.3684 | $ 77,515 | 0.4420 |
| Interest and amortization | 86,400 | 0.1234 | 86,400 | 0.1646 | 86,400 | 0.2468 | 86,400 | 0.4936 |
| Total cost | $318,160 | 0.4545 | $266,745 | 0.5081 | $215,330 | 0.6152 | $163,915 | 0.9356 |
| Cost per kw.-yr | 39.81 | ...... | 44.51 | ...... | 53.89 | ...... | 83.96 | ...... |
| $1.50 oil | 210,000 | 0.3000 | 157,500 | 0.3000 | 105,000 | 0.3000 | 52,500 | 0.3000 |
| Total cost | 343,160 | 0.5045 | 292,995 | 0.5581 | 232,830 | 0.6652 | 172,665 | 0.9856 |
| $1.75 oil | 245,000 | 0.3500 | 183,750 | 0.3500 | 122,500 | 0.3500 | 61,250 | 0.3500 |
| Total cost | 388,160 | 0.5545 | 319,245 | 0.6081 | 250,330 | 0.7152 | 181,415 | 1.0356 |
| $2 oil | 280,000 | 0.4000 | 210,000 | 0.4000 | 140,000 | 0.4000 | 70,000 | 0.4000 |
| Total cost | $423,160 | 0.6045 | $345,495 | 0.6581 | $267,830 | 0.7652 | $190,165 | 1.0856 |

TABLE 2.—*Cost of Generating Power with Two* 6,000-*Kw. Steam Turbines*

Costs are based on the following conditions:

| | |
|---|---|
| Cost of complete plant | $720,000 |
| One unit operates continuously at full load, one unit is stand-by. | |
| Cost of fuel oil, $1.25 per bbl. of 320 lb. | |
| Labor, lubrication, boiler upkeep, per year | 25,000 |
| Maintenance, 1 per cent. of cost of installation, per year | 7,200 |
| Interest and amortization, 12 per cent. of cost of installation, per year | 86,400 |

Kilowatt-year figured at 8,760 kw.-hr. Turbines to operate at 180 lb. steam pressure, 620° F. steam temperature, 80 per cent. boiler efficiency, 60° F. inlet temperature, 75° F. discharge temperature of condenser circulating water; quantity of cooling water per 6,000-kw. unit : 74,000 cu. ft. per hr. Commercial generator efficiencies.

| | 8,000 kw. | | 6,000 kw. | | 4,000 kw. | | 2,000 kw. | |
|---|---|---|---|---|---|---|---|---|
| Kw.-hr. per year | 70,000,000 | | 52,500,000 | | 35,000,000 | | 17,500,000 | |
| Barrels | 300,000 | | 210,000 | | 150,000 | | 83,700 | |
| | Cost per Year | | Cost per Year | | Cost per Year | | Cost per Year | |
| | Per Kw.-hr., | Per Kw.-hr., Cents | Per Kw.-hr., | Per Kw.-hr., Cents | Per Kw.-hr., | Per Kw.-hr., Cents | Per Kw.-hr., | Per Kw.-hr., Cents |
| Fuel cost | $375,000 | 0.5357 | $262,000 | 0.5000 | $187,000 | 0.5357 | $104,625 | 0.6000 |
| Labor, etc | 25,000 | 0.0357 | 25,000 | 0.0476 | 25,000 | 0.0715 | 25,000 | 0.1430 |
| Maintenance | 7,200 | 0.0103 | 7,200 | 0.0137 | 7,200 | 0.0206 | 7,200 | 0.0412 |
| Total direct | $407,200 | 0.5817 | $294,700 | 0.5613 | $219,200 | 0.6278 | $136,825 | 0.7842 |
| Interest and amortization | 86,400 | 0.1234 | 86,400 | 0.1646 | 86,400 | 0.2468 | 86,400 | 0.4936 |
| Total cost | $493,600 | 0.7051 | $381,100 | 0.7259 | $305,600 | 0.8746 | $223,225 | 1.2778 |
| Cost per kw.-yr | 61.75 | ...... | 63.50 | ...... | 76.60 | ...... | 111.70 | ...... |
| $1.50 oil | 450,000 | 0.6430 | 315,000 | 0.6000 | 225,000 | 0.6430 | 125,500 | 0.7171 |
| Total cost | 568,600 | 0.8123 | 433,600 | 0.8259 | 343,100 | 0.9818 | 244,200 | 1.3949 |
| $1.75 oil | 525,000 | 0.7500 | 367,500 | 0.7000 | 262,500 | 0.7500 | 146,375 | 0.8342 |
| Total cost | 643,600 | 0.9194 | 486,100 | 0.9259 | 380,600 | 1.0890 | 265,124 | 1.5120 |
| $2 oil | 600,000 | 0.8571 | 420,000 | 0.8000 | 300,000 | 0.8571 | 167,250 | 0.9513 |
| Total cost | 718,600 | 1.0257 | 538,600 | 1.0259 | 418,100 | 1.1962 | 286,050 | 1.6291 |

The last three double columns show the effect of increased cost of fuel oil on power cost, the total fuel cost being shown, as well as the total cost of power, which is merely increased in proportion to the increased fuel expenditure.

The higher the cost of fuel and the load factor, the greater will be the proportion of the fuel to the other costs making up the total cost of generating power. Capital charges (interest and redemption), maintenance, operating labor, and lubrication are practically constant amounts regardless of the load-output of the station; the fuel consumption and its cost are practically the only variables with the load.

The costs here given are based on conditions as generally found in the Southwestern States, and on Diesel engine operating experience in

European countries, where this type of prime mover has reached a high state of perfection, units upward of 4,000 hp. having proved entirely successful in operation.

This paper will not deal with questions of design, nor types of engines, as much space would be needed to do justice to the large number of excellent makes of engines. It is assumed that the reader has a general knowledge of Diesel engines.

The reasons favoring the selection of two 6,000-kw. steam turbines and four 2,000-kw. Diesel engine generating sets, can be briefly stated as follows:

1. Uninterrupted power service is essential, as failure of power would entail far greater losses in operating revenue than the interest on the cost of the reserve turbine unit.

2. The selection of two 6,000-kw. rather than three 3,000-kw. steam turbines is made on account of the greater efficiency of a 6,000-kw. over two 3,000-kw. units, the load to be supplied being nearly 6,000-kw. continuously. (In plants with fluctuating loads which can be supplied by one or more smaller units operating in parallel during different portions of the day, each unit operating at or near its full-load capacity, a number of small units is justified; such diversified load conditions, however, apply mainly to central stations in cities, rather than mine, mill, and smelter power plants, which have invariably a high load factor during the entire day.)

3. The reserve boiler capacity would be an additional third of the boiler capacity in continuous operation, regardless of size of turbines selected. The extra cost of the surplus turbine capacity by reason of installing two 6,000-kw. instead of three 3,000-kw. turbines is therefore moderate when taking future power costs and other factors into consideration.

4. The selection of four 2,000-kw. Diesel engine units is justified on the ground that larger engines are not any more efficient in fuel economy, and the relatively much higher cost of this prime mover[1] makes it policy to reduce reserve capacity to a safe minimum. An excessive number of small Diesel units should also be avoided, as the cost per kilowatt of installed power-plant increases, and the multiplication of cylinders, valves and engine parts to be taken care of increases maintenance and labor charges.

The heat utilization and cooling-water requirements of Diesel engines

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[1] A Diesel engine with its direct-connected compressor, alternator and exciter is a complete prime mover, and should be compared with a complete steam-power plant comprising boiler and condenser equipment, turbine and generator. Diesel engines are simple in that the installation is compact, comprising few parts, and occupying only about 50 to 60 per cent. of the space of a steam-turbine installation of like power.

and steam turbines at full load are graphically shown in Fig. 2 for units of 2,000 hp.

In general, the selection of either type of prime mover will be governed by the following economic considerations:

1. Fuel is of chief influence on the total power cost where both the B.t.u. price[2] and load factor are high. These conditions favor the use of the Diesel engine.

2. Interest and redemption (amortization) are of chief influence on the total power cost with low B.t.u. price and low load factor; this applies

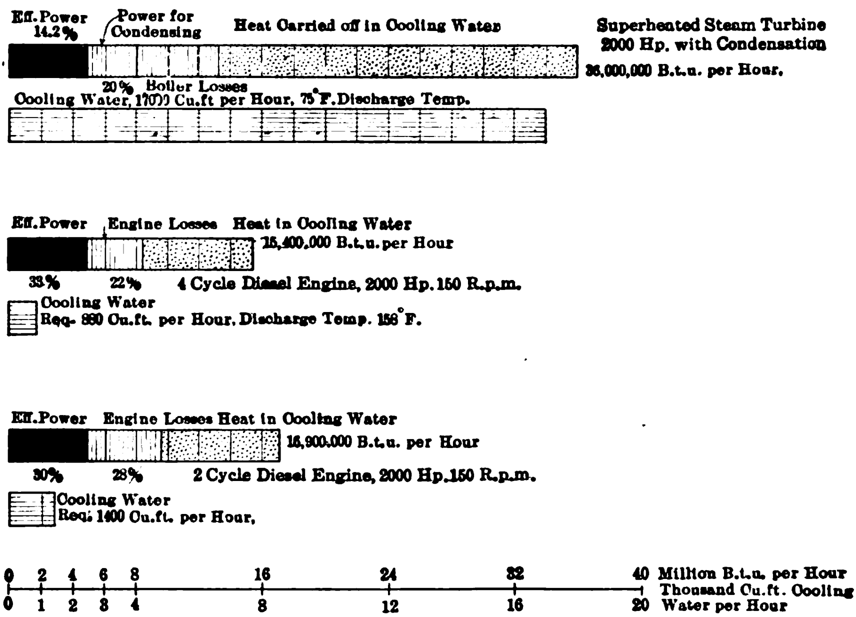


FIG. 2.—COMPARATIVE EFFICIENCIES OF STEAM TURBINES AND DIESEL ENGINES OF 2,000 EFFECTIVE HORSEPOWER.

particularly to stand-by plants, which are operated only occasionally or have to supply recurring peak loads. The installation cost of such plants must be kept as low as possible so as to avoid heavy capital charges distributable over a relatively small kilowatt-hours output of the station. Such conditions favor the steam turbine.

3. Exceptions to (2) are cases where the constant and instant readiness of the Diesel engine give it preference, and installation cost is of secondary importance. Here we would have to balance the cost of keeping boilers under steam continuously against the difference of interest charges of steam-turbine and Diesel engine plants.

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[2] By B.t.u. price is meant the price or cost for a given number of B.t.u., say 1,000,000 B.t.u., all fuel prices being reduced to this standard of comparison.

4. Power plants located at the source of the fuel, either in the oil fields, or at the coal mine, and therefore enjoying the advantages of a very cheap fuel supply, will select prime movers costing least to install, *i.e.*, steam turbines, since fuel expenditures will weigh less than interest and redemption charges.

5. In many cases combination plants using Diesel engines for supplying the continuous and nearly constant main load, and steam turbines for furnishing periodically occurring peaks by the use of high-duty boilers with large water and steam spaces, capable of being forced when necessary, will prove most profitable. Thus, as an example, the periodical peaks produced in hoisting may be taken care of by a turbine floating on the line and operating in parallel with Diesel engines that supply the main load and operate constantly at or near full load.

6. Steam power will remain the cheapest power wherever waste-heat gases are available, as for instance gases from reverberatory smelting furnaces, where nearly one-half of the fuel used in smelting can be utilized for steam generation. Nearly 3,000,000 B.t.u. for every ton of charge smelted are thus available for steam generation, or about 150 hp.-hr. per ton of charge.

7. Up to capacities of 1,000-hp. steam turbines can compete with Diesel engines only in special cases, such as supplying exhaust steam for heating purposes. For such small units, particularly for greatly varying loads, high-grade reciprocating steam engines are preferable. For larger plants, from 1,000 to 10,000 kw. capacity, careful analysis must be made of the relative advantages of Diesel engines and turbines, a knowledge of the load factor, fuel prices and water conditions being necessary. Power plants larger than 10,000 kw. using units from 6,000 kw. upward, preferably use steam turbines, unless a combination of high load factor, high fuel cost, and poor water conditions favor a Diesel plant. This is a special condition frequently met.

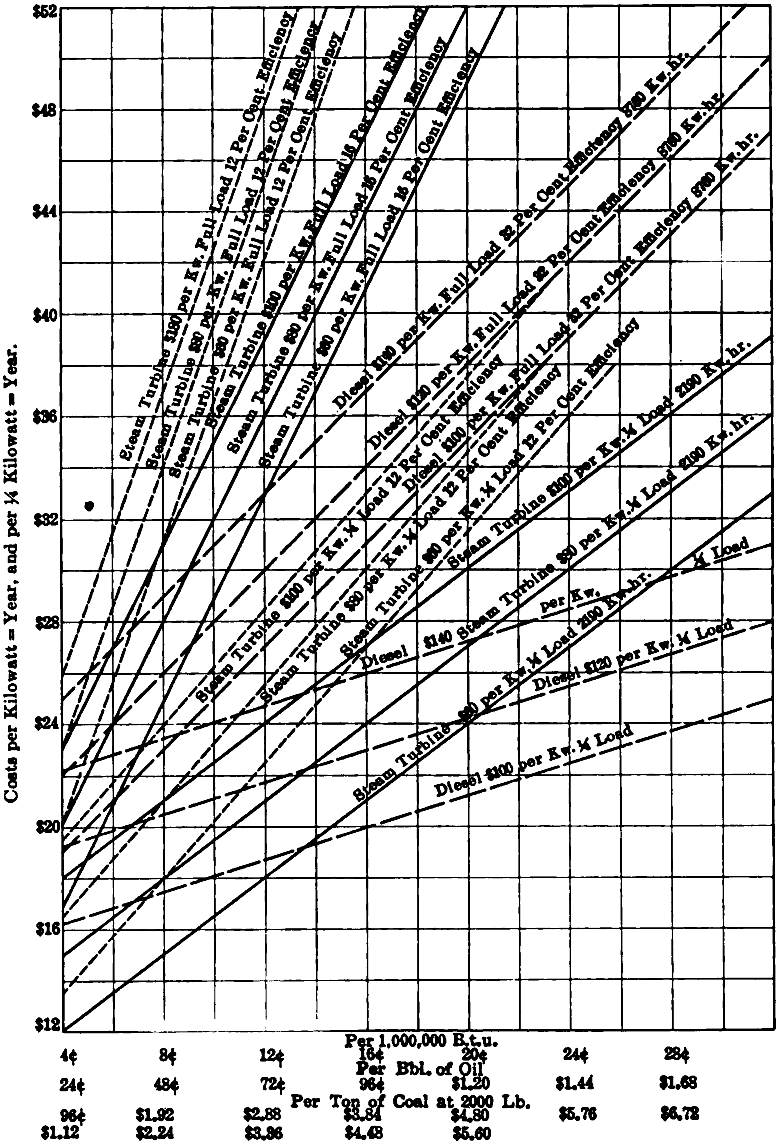
### *Fuel Costs*

In comparing prices of different kinds of fuel, such as gas, coal, or liquid fuels, it is well to reduce all fuel prices to a common basis of absolute cost for 1,000,000 B.t.u. We have, then, for the cost of 1,000,000 B.t.u. transformed into mechanical work:

$$\frac{\text{Cost of 1,000,000 B.t.u.}}{\text{Over-all thermal efficiency of plant}}$$

and the fuel cost per brake horsepower-hour =

$$\frac{\text{Cost of 1,000,000 B.t.u.} \times 2{,}545}{\text{Over-all thermal efficiency} \times 1{,}000{,}000}.$$




Oil Costs Based on 1 Bbl. Containing 6,000,000 B. t. u.
Coal Costs Based on 1 lb. Containing 12,000 and 14,000 B. t. u.

FIG. 3.—DIAGRAM SHOWING COMPARATIVE COSTS OF STEAM TURBINE AND DIESEL ENGINE PLANTS AT GIVEN FUEL PRICE AND LOAD FACTOR.

Knowing the thermal efficiencies of different prime movers, we can then arrive at the absolute fuel cost of each per brake horsepower-hour with different kinds of fuel. Thermal efficiencies of steam engines and turbines vary from 6 per cent. operating simple and non-condensing to 16 per cent. with large units, depending on size, and operating conditions; and of Diesel engines from 30 to 35 per cent., at full load. Thus, for a steam plant with a 12 per cent. efficiency, an absolute heat price of 12c. for 1,000,000 B.t.u. comes to 100c. for every 1,000,000 B.t.u. turned into effective mechanical work, whereas for a Diesel engine plant with, say 30 per cent. efficiency, 1,000,000 B.t.u. transformed into effective mechanical work would cost only 40c., and the respective costs per brake horsepower-hour would be 0.2545c. and 0.1018c.

Where a supply of liquid fuel is available, the sound economic application of steam turbine and Diesel engine is determined by fuel prices, and load factor. The graph in Fig. 3 shows at a glance whether a steam turbine or a Diesel engine power plant is in place with a given fuel price and load factor. The abscissæ denote absolute fuel prices for each 1,000,000 B.t.u. These prices are translated into corresponding fuel prices per short ton (2,000 lb.) of bituminous coal having a fuel value of 12,000 and 14,000 B.t.u. per pound and per barrel of oil containing 6,000,000 B.t.u. The ordinates denote costs. The power costs represented by the curves include capital costs and fuel costs, but not operating labor, this being about the same for either type of plant, except for very large steam-turbine plants with units above 6,000 kw., which require less operating labor than Diesel engines and with which the latter can compete only under special conditions. The capital charges cover interest, amortization and maintenance, and are taken as 15 per cent. of the cost per kilowatt of installed plant capacity, which is for a complete, modern power plant with prime movers, switchboard equipment, and building, including necessary accessories, but without transformers or transmission line. The installation costs varying with size of plant and locality, three different costs per kilowatt of installed capacity are considered for each type of prime mover. For the turbine plant, $60, $80, $100, and for the Diesel plant $100, $120, and $140; 15 per cent. of these respective amounts are the capital charges added to the fuel costs, the curves marked "full load" showing the combined cost per kilowatt-year (8,760 kw.-hr.) of fuel and capital charges, the cost per kilowatt-year being shown in dollars in the ordinates.

The curves marked "quarter load" cover fuel cost for one-fourth kw.-year (or 2,190 kw.-hr) and the same capital charges as for full load; to arrive at the cost per kilowatt-year (8,760 kw.-hr.) *at quarter load,* the amount shown in dollars must be multiplied by four. This product divided by 8,760 will give the true cost per kilowatt-hour generated at quarter load.

Any other fractional load cost can be obtained by interpolation. An example will best illustrate the use of the graph: With a turbine plant costing $60 per kw. of capacity and a Diesel plant $100, the curve marked "steam turbine $60 kw. full load 16 per cent. efficiency," will be seen to intersect the curve "Diesel $100 per kw. full load 32 per cent. efficiency" at a fuel cost of 6c. per 1,000,000 B.t.u. and the cost per kilowatt-year (8,760 kw.-hr.) is $21. Under these conditions, the costs being equal for both prime movers, it would not pay to use a Diesel plant, a turbine plant having certain mechanical advantages. To the right of the intersection of these two curves, *i.e.*, with an increase in the price of fuel, the Diesel engine is the more economical prime mover. Assuming a 20c. fuel cost per 1,000,000 B.t.u., we find a cost of $35 per kw.-year at full load for the Diesel plant, and $49 for the turbine plant (16 per cent. efficiency), the difference of $14 representing the fuel saving per kilowatt-year of the Diesel plant under full-load conditions. A plant of 2,000 kw. would save $28,000 per year. The corresponding curves marked one-fourth load will be seen to intersect at a fuel cost of 13½c. per 1,000,000 B.t.u. and a cost of $19.20 for 2,190 kw.-hr. (¼ load). The kilowatt-year at quarter-load conditions would cost 4 × $19.20 = $76.80. In this case, Diesel engines would only pay if the fuel costs were higher than 13½c. per 1,000,000 B.t.u. The divergence to the right between these curves indicates the fuel saving above capital charges for the Diesel engine (or, to the left, saving in capital charges above fuel saving, with turbines) per kilowatt at quarter load. At 20c. for 1,000,000 B.t.u., this divergence shows ($24–$21.20) a net saving of $2.80 for the Diesel plant per kilowatt-year, for each 2,190 kw.-hr. per year (¼ load). For 75 and 50 per cent. load factor, the critical fuel prices can be ascertained by taking the difference between those at quarter and full loads, and adding two-thirds of this difference to the amount corresponding to quarter load to arrive at 75 per cent. load, and adding one-third of the difference to the amount for quarter load to arrive at 50 per cent. load. Any other critical fuel prices for other load conditions can be easily ascertained by interpolation.

It will be seen that curves based on 12 and 16 per cent. efficiency of steam-turbine plants are given. The higher figure is secured only with large units in sizes from 10,000 kw. upward. (These refer to the over-all efficiency of the entire power plant.) For smaller units, up to 2,000 kw., 12 per cent. represents good average practice. How a decrease in thermal efficiency favors the Diesel engine is strikingly shown by the curves for the steam turbine, with 12 per cent. efficiency at full and quarter load.

To secure complete power costs, the cost of operating labor and lubrication must be added to the above costs. The labor cost per kilowatt-year varies greatly with the size of the plant. Fig. 4 shows average

labor operating charges per kilowatt-year at full load for different-sized Diesel engine plants. This labor charge remains practically constant, regardless of load variations, so that with fractional loads the labor

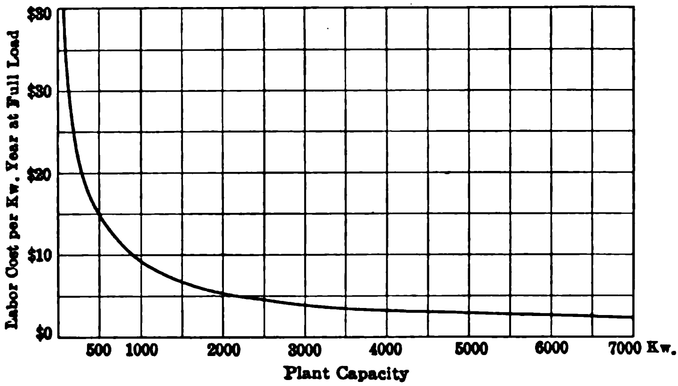



FIG. 4.—OPERATING LABOR COST PER KILOWATT-YEAR (8,760 KW-HR) AT FULL LOAD IN DIESEL ENGINE PLANTS.

increases in inverse proportion. In turbine and Diesel plants here considered, the labor cost is about the same for either type of plant, as any increased attendance required by Diesel engines is offset by firemen, water

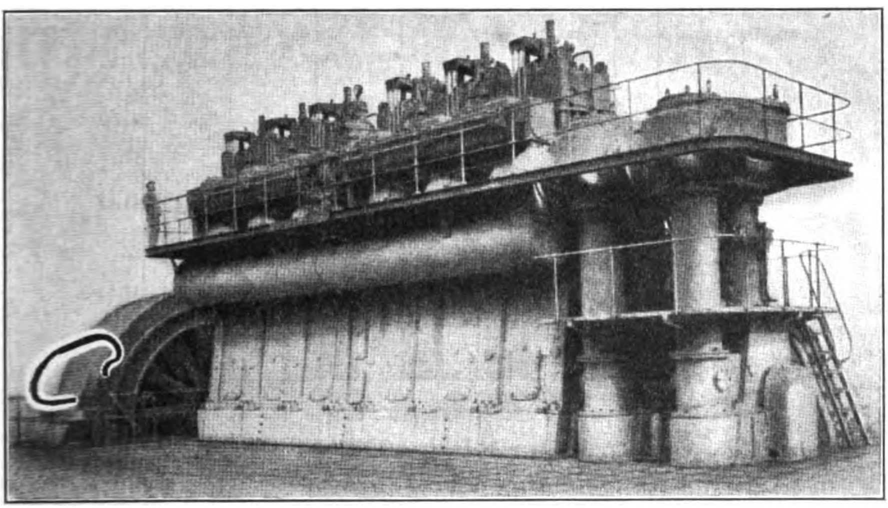


FIG. 5.—HIGH-DUTY, 4,000-HP. DIESEL ENGINE.

tenders, and usual operating force of turbine plants. The cost for lubrication is considerably higher for Diesel plants than for turbine plants, and amounts to from $2 to $4 per kilowatt-year, depending on the type of

lubricating system used and the care of oilers. As a rule, a number of units are used in large Diesel installations, so that the lubrication expense is nearly proportional to the load, as with a decrease in load, units can be shut down. In turbine plants this greater expense for Diesel engines is partly balanced by the large amount of water required for condensing,

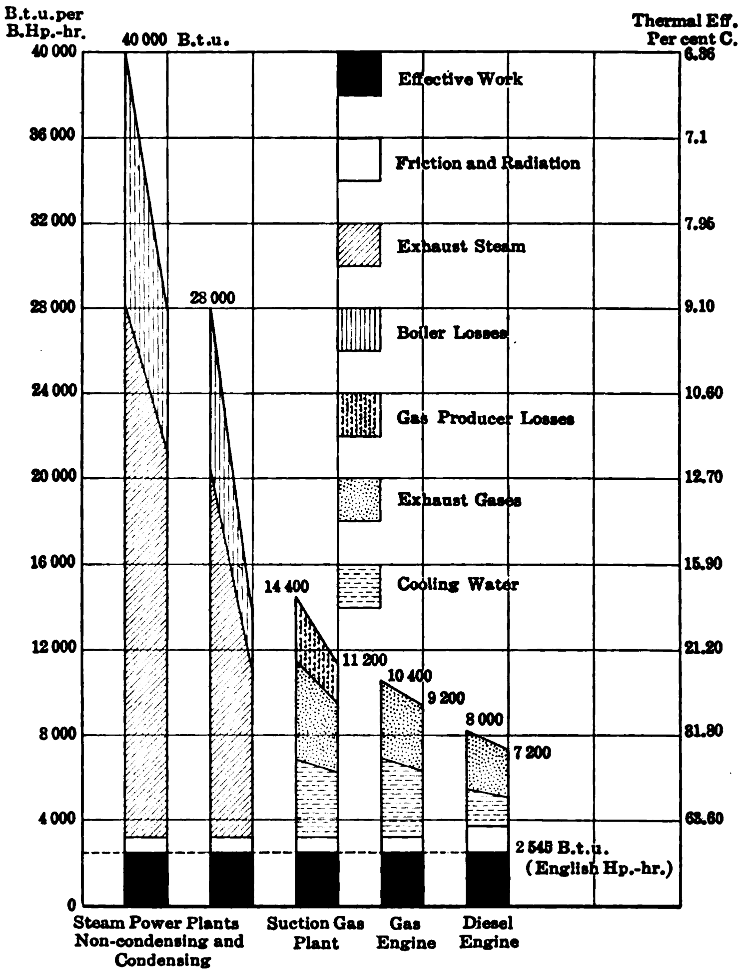



FIG. 6.—THERMAL BALANCE OF DIFFERENT HEAT ENGINES.

which is 12 to 20 times greater than the supply needed for cooling Diesel engines. Only in plants with units in excess of 6,000 kw. have turbines decided advantages over Diesel engine plants.

Fig. 5 is a good example of a high-duty 4,000-hp. Diesel engine. In conclusion, Fig. 6 shows the present status of different types of prime movers, expressed in thermal balances.

It is hoped that the above notes will be found helpful to those having to select the most economical prime mover to meet certain conditions, either turbines or Diesel engines having a wide field of usefulness; a combination of both types often best meets particular conditions. The selection should be governed by the above discussed economic considerations, the most important of which will always be the fuel price and load factor.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Comparisons between Electrolytic and Two Varieties of Arsenical Lake Copper with Respect to Strength and Ductility in Cold-Worked and Annealed Test Strips*

BY C. H. MATHEWSON,† PH. D., NEW HAVEN, CONN., AND E. M. THALHEIMER,§ NEW HAVEN, CONN.

(Arizona Meeting, September, 1916)

## Character of the Work in Hand

In planning the present experiments, we have made a particular effort to secure that adjustment of working conditions which would render the forthcoming tests most serviceable by way of indicating the comparative properties of the three different commercial grades of copper used. More specifically, we have endeavored to obtain certain definite information with respect to the relative behavior under test of ordinary electrolytic copper and furnace-refined lake copper containing (*a*) a moderate percentage of arsenic and (*b*) a rather large percentage of arsenic, in addition to the usual content of silver and other impurities.

These comparisons were made systematic and comprehensive by testing throughout a series of (eight) reductions running from 0 to 70 per cent. in the case of cold-rolled metal, and throughout a series of (15) temperatures running from 100° C. to 1,000° C. in the case of annealed metal. In other words, a survey of properties similar to that made by Grard,[1] in the case of electrolytic copper, was here repeated (with modifications) and extended to two other commercial varieties of copper.

The great importance of copper both as a material of construction and as the base of our most important alloys (aside from steel) has led to innumerable investigations of its properties, particularly in commercial forms. Naturally, the merits of furnace-refined lake coppers *vs.* those of the very pure electrolytic product have furnished the subject of much experimental work and animated discussion, wherein the effect of arsenic

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[1] C. Grard: Cuivre et Laitons à Cartouches, *Revue d'Artillerie*, February–April, 1909.

has by no means been overlooked. In the literature of the past 10 years, we have found a number of papers in which the ordinary mechanical properties of various brands of copper have been compared after more or less limited or arbitrary forms of treatment, but no paper describing comparative tests after systematic variation of thermal and mechanical treatment. While much of this material is valuable, nevertheless, owing to its diverse and fragmentary character, we have not thought it desirable to use it as a general basis for discussion in connection with our own tests, or, indeed, to attempt any critical review of the literature bearing upon the present subject.

It may be remarked that our experiments do not deal with the effect of arsenic in the absence of other common impurities—little or no progress appears to have been made in this direction since the publication of Bengough and Hill's experiments in 1910[2]—but refer exclusively to comparisons between electrolytic copper and two arsenical brands produced by the Michigan Smelting Co. throughout a wide range of treatment. It is our belief that the interests of consumers and other parties to whom comparative tests are of value may best be served by simple presentation of the results of these tests, leaving special questions of interpretation in the hands of any interested reader. Summaries of recent literature dealing with the properties and structure of copper in various forms may be found in Hofman's *Metallurgy of Copper*[3] and in reports of the Symposium on the Metallurgy of Copper, presented at the meeting of the International Engineering Congress, 1915, in San Francisco.[4]

It may also be appropriate to remark that we have restricted our work to the determination of tensile strength (maximum-load strength), elongation, and reduction of area—properties which are used as a basis for judgment as to quality affecting most engineering requirements. Since there are other criteria of judgment—some specific and capable of routine development, such as the determination of elastic limit, modulus of elasticity and of strength properties under different forms of loading, or at elevated temperatures; others not so well standardized, but, nevertheless, susceptible to quantitative treatment, such as electrochemical potential (rate of solution), varied chemical behavior, etc.; and still others which can only be gaged by manufacturing or service conditions, such as surface qualities, or peculiarities, during drawing operations—we have by no means exhausted the possibilities of comparison in the present series of tests.

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[2] G. D. Bengough and B. P. Hill: The Properties and Constitution of Copper-Arsenic Alloys, *Journal of the Institute of Metals*, vol. 3, pp. 34–97 (1910).

[3] McGraw-Hill Book Co., New York, 1914.

[4] Papers by William Campbell on the Metallography of Copper and by C. R. Haywood on the Physical Properties of Copper.

Questions of homogeneity with respect to the distribution of oxide and other micrographic features have been dealt with in a limited sense in the present paper, as have questions of resistance to the destructive agency of reducing gases during heat treatment ("gassing" of copper).

It has not been possible to investigate the effect of degree of deformation (reduction by rolling) and of time in coördination with temperature and strength properties in connection with the annealing experiments, since such an elaboration of the work would necessitate the performance of several thousand instead of several hundred individual tests. A general treatment of this subject has been attempted by one of us[5] and three-dimensional annealing diagrams (degree of deformation, percentage decrease in tensile strength, temperature for definite period of anneal; degree of deformation, percentage decrease in tensile strength, period of anneal at constant temperature), based upon a limited amount of experimental work in the case of "magnesium bronze" (copper, deoxidized with magnesium) have been given by von Müller.[6]

After due consideration of the questions raised above, we decided to use an annealing period of 40 min. and a degree of deformation amounting to a 50 per cent. reduction in area of section by cold-rolling as a basis for all annealing tests. Such procedure is not inconsistent with ordinary annealing practice and cannot fail to develop the principal annealing characteristics of the material in hand.

## Composition of the Metal Tested

Metal was sent to the rolling mill in the form of commercial cakes, 14 by 17 by 3 in., weighing in the neighborhood of 275 lb.

Two of these cakes were cast at the works of the Michigan Smelting Co., Houghton, Mich., and these represent the average quality of brands known as Mohawk and Copper Range copper, respectively. The other cake was obtained from The United States Metals Refining Co. ("D R W" copper) and is taken to represent electrolytic copper of average quality.

Determinations of oxygen and arsenic were made in the laboratory of the Michigan Smelting Co. Determinations of silver were made in the Hammond Laboratory.

Details of analytical procedure need not be introduced. All determinations were made by experienced chemists and we have every confidence in their accuracy. No determination was made of the very small quantities of iron, sulphur, nickel, cobalt, tellurium, etc., which may be present in the metal.

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[5] C. H. Mathewson: A Metallographic Examination of Some Ancient Bronzes. *American Journal of Science*, Series 4, vol. 40, pp. 550–555 (1915).

[6] von Müller: Beitrag zur Erkenntniss des Einflusses der Gluhdauer auf die Erweichung verschieded stark gereckter Leitungsbronze, *Metall und Erz*, Neue Folge, vol. 3, pp. 213–220 (1915).

The analytical results follow:

| | Oxygen, Per Cent. | Arsenic, Per Cent. | Silver | |
|---|---|---|---|---|
| | | | Per Cent. | Ounces |
| Electrolytic copper............ | 0.071 | 0.000 | 0.0005 | 0.15 |
| Mohawk copper............... | 0.052 | 0.096 | 0.069 | 20.0 |
| Copper Range copper.......... | 0.055 | 0.296 | 0.052 | 15.0 |

## Previous History of Test Pieces and Experimental Procedure

The experimental work was divided into two parts: (1) determination of the alteration in strength and ductility (strain hardening) produced by rolling the three kinds of material from an initially soft condition through a series of reductions as specified below, and (2) determination of the alteration in strength and ductility (removal of strain hardening) brought about by annealing the three kinds of material after a previous reduction of 50 per cent. in area of section by rolling.

Accordingly, two separate sets of test strips were prepared. In preparing both sets, the original cake was first planed to a thickness of 2 in., then hot-rolled and finally cold-rolled and annealed before variation in treatment designed to produce the two different sets of test strips was introduced. This procedure insured the production of test strips which had been thoroughly worked, both hot and cold, under conditions which are representative of the treatment commonly received by most mill products of this sort.

### *Preparation of the First Set of Test Strips*

One of two methods may be selected in preparing a set of test strips of different tempers (set 1 according to the above designation), *i.e.*, the strips may be finished at a succession of gages, or all of them may be finished at the same gage.

Steps (*a*), (*b*), and (*c*), of the accompanying diagram, Fig. 1, represent the preliminary stages of rolling which may be considered common to both methods. The relative changes in thickness from one stage of reduction to another are correctly shown in the sketches and the actual gages and percentage reductions between stages are written into the sketches. Sketch (*a*) represents the original cake planed to a thickness of 2 in.; sketch (*b*) represents the material after hot-rolling to gage No. 0, B. & S. (0.325 in.), a reduction of 83.75 per cent.; and sketch (*c*) represents this material after cold-rolling to gage No. 8 (0.128 in.), a reduction of 60.6 per cent. At this point the metal is annealed.

If it is desired that the final product shall be obtained according to the first method, the annealed metal (*c*) is cut into strips which are rolled to a series of measured reductions, in which case the thickness will decrease progressively from that of the first strip (minimum reduction) to that of the last (maximum reduction).

This method was adopted in the present investigation and the numeri-

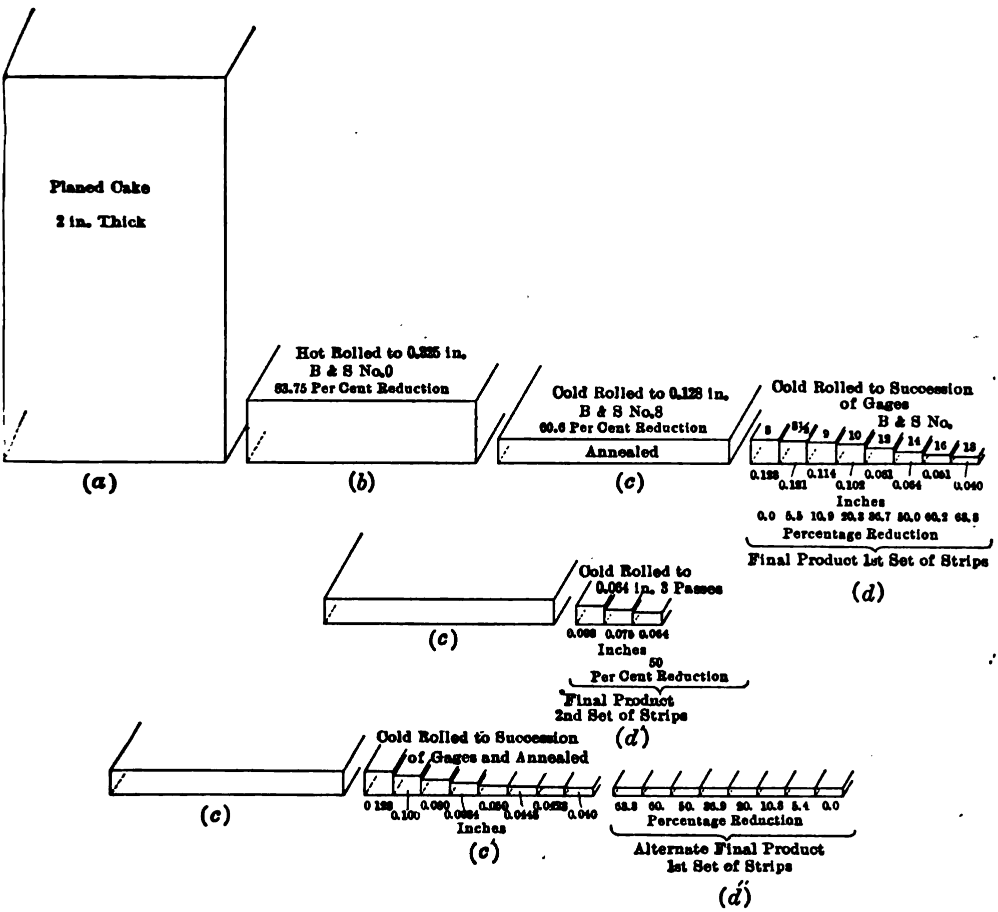


Fig. 1.—Diagram to Represent Preparation of Test Strips.

cal specifications sent to the mill[7] to cover this part of the work are written into sketch (*d*) of Fig. 1, which represents the series of eight strips composing this set. For convenience, even B. & S. gage numbers running from 8 to 18 (exception, 8½) were used to limit the separate stages. The percentage of the initial thickness (0.128 in. in all cases) taken off in

[7] The material used in all of the tests described in this paper was rolled by the Bridgeport Brass Co.

the reduction is entered below the value for the final thickness under the sketch of each respective strip. Thus, in the case of the last strip:

$$\frac{0.128 \text{ in. (initial thickness)} - 0.040 \text{ (final thickness)}}{0.128 \text{ in. (initial thickness)}} \times 100 = 68.8.$$

In view of the convenient survey of the rolling stages afforded by the sketches of Fig. 1, it appears superfluous to draw up a table of the data involved.

If it is desired that the final product shall be obtained according to the second method (which was not used in these experiments), stages ($c'$) and ($d''$) may be substituted for stage ($d$). According to ($c'$), the metal from ($c$) is cut into strips which are given a series of reductions as shown, so that, after annealing the lot, another set of reductions will produce test pieces of the same thickness, but different tempers, as indicated in ($d''$). In order to make the comparison perfectly clear, the reductions marked in sketch ($c'$) have been calculated so as to yield the same series of reductions after the strips have been rolled to a final product of uniform thickness (0.040 in.) as was obtained in the final product according to the first method. These reductions are marked in sketch ($d''$).

Since we had to consider carefully the respective advantages of these two methods, it seems appropriate to devote some space to a consideration of this subject. It is unquestionably desirable, other things being equal, to perform all tests upon strips of the same cross-sectional area. The relation between surface and volume of the test piece influences to some extent the testing value calculated per unit of sectional area. There appears to be a "skin effect" which tends to produce higher values of tensile strength in thin specimens than in thick specimens. Notably, in very thin specimens, the testing results are considerably dependent upon the surface conditions of the test piece and comparatively insignificant scratches, or indentations, may lead to premature rupture.[8]

There are, however, certain difficulties in the way of producing a satisfactory set of test strips of uniform thickness and varying percentages of reduction. In the first place, the different samples must receive different amounts of work before the necessary anneal which precedes the final impression of a graded series of reductions bringing all strips to the same finishing thickness. This means that the micrographic characteristics, notably grain size, of the different strips cannot be uniform; those which received a heavy reduction before the anneal will possess a more highly refined grain than those which received a light reduction. It is

[8] Metallographists are generally familiar with the phenomenon of surface flow, first demonstrated by Beilby, which produces a surface hardening, quite marked in the case of soft metals such as copper, during any ordinary process of surface dressing or finishing.

true that we may depart somewhat from the order of changes sketched in (*c*) and (*c'*) of Fig. 1 by cold-rolling in (*c*) to a lesser degree, say, to gage 0.220 in., instead of gage 0.128 in., omitting the anneal at this point, and continuing the rolling so as to produce strips of the gages shown in (*c'*). All of these strips would then have received comparatively heavy reductions and grain refining would be more uniform in the ensuing anneal. This procedure would modify but not entirely remove the difficulty in question.

From a practical standpoint, there would be no check on the rolling-mill work, since the different gages in the intermediate product are all reduced to the same gage in the final product and any departure from specifications in the intermediate work could not be recognized in the final product. This is serious, because such inaccuracies might greatly influence the temper of the product, particularly in the case of light reductions. It may be noted, in this connection, that to produce a final product of gage 0.040 in., having a reduction of 5.4 per cent., the strip is required to be only 0.0423 in. thick before rolling. This requirement is too rigid for ordinary mill work. When the same reduction is made according to the first method, it is performed upon a thicker strip and 0.007 in., instead of 0.0023 in., is taken off. To eliminate this difficulty, the whole set of strips would have to be finished thicker.

In the case of the first method, we are in possession of a reliable check upon the rolling-mill work, since the amount of reduction shows directly in the individual strips and may be measured up with the micrometer.

On the basis of these remarks, it is perfectly clear that the first method is preferable when the metal must be sent to a rolling mill for preparation. With respect to the shortcomings of metal prepared in this way, it may be stated that from the standpoint of comparison between different brands of metal, as in the present case, such very largely disappear, since they only affect the comparison between different tempers and the validity of the succession of testing values thereby secured. It is here assumed that the different kinds of metal follow the same schedule in the mill.

### *Preparation of the Second Set of Test Strips*

The test strips of the second set were finished at gage No. 14 (0.064 in.) with a reduction of 50 per cent. taken off in three passes beginning with annealed metal at gage No. 8 (0.128 in.). These strips were taken from the same cakes as the first set of strips. The rolling operations are, accordingly, those shown in Fig. 1, (*a*), (*b*), and (*c*) followed by (*d'*), instead of (*d*).

### *Specifications*

It has already been stated that the numerical specifications for rolling are indicated in the sketches of Fig. 1. In sending these specifications

to the mill, it was requested that the rolling conform as closely as possible to the data submitted, but a margin of 0.003 in. above or below any given gage was allowed. The mill work was conducted under the supervision of the works laboratory and we were generally satisfied with the product.

Slight departure from the specifications scarcely affects the percentage reductions figured from the specifications except in the case of very light reductions, as in the first two of the first set. Here, the actual reductions varied considerably in the three kinds of material, which would not have been the case if each had been passed through the same setting of the rolls. This was recommended but did not appear feasible in the mill. However, such variations are not serious, since we have the data for calculating the exact reductions obtained. All of the data in question will be found in Tables 2 and 3.

Since we are chiefly interested in comparisons between the three kinds of material, it is important that all receive the same treatment in prescribed annealing operations. On this account, it was insistently urged that the different lots be annealed simultaneously and in the same part of the load. From our micrographic examination, we believe that these specifications, which called for uniform anneal for 1 hr., at 650° to 700° C., were properly carried out.

*Procedure with the First Set*

In the case of the first set of test strips, the laboratory work consisted solely in testing the prepared strips.

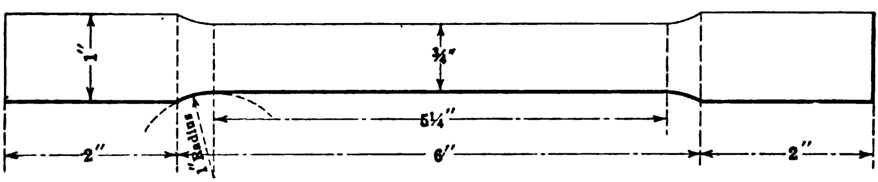


FIG. 2.—DIMENSIONS OF TEST STRIP.

For this purpose, a 50,000-lb. Riehle Universal testing machine was used. Each final result represents the average of tests on four separate strips. A uniform rate of loading was used in all cases, light scratch marks instead of punch marks were made on the strips, and it is believed that all essential precautions were observed in the interest of obtaining the most trustworthy results. Test pieces were accurately milled according to the accompanying sketch (Fig. 2). It may be noted from this sketch that the original strips were 10 in. long by 1 in. wide, while, in milling, three-quarters of the original sectional area was retained in the test section. Our intention in this respect was to cut away just enough material from the sides to insure a properly located break without de-

creasing the breaking load to a point which would render the errors at the machine serious in comparison. The thinnest (most severely reduced) strips (0.040 in.), although the strongest per unit of sectional area, broke under the lightest load, viz., some 1,700 lb. The minimum breaking load for the regular annealed strips of the second set (0.064 in.) was some 1,250 lb., while the weakest of the small number of strips tested after anneal in a reducing atmosphere broke at 930 lb. Thus, if the machine can be read accurately to 10 lb., the maximum testing error is only about 1 per cent.

Breaks were considered faulty when they could not be placed near the middle of a 2-in. length selected with the aid of the scratch marks along the milled section of the strip, viz., elongation was measured in 2 in. and the measurements were made with proper regard for the distribution of elongation along the broken strip.

Measurements of reduction of area were made with the micrometer and, owing to the comparatively small sectional area of the strips, the results cannot be considered very accurate, particularly in the case of all values below about 25 per cent.

*Procedure with the Second Set*

The strips of the second set were all tested after a prescribed annealing treatment. The following is a summary of the methods adopted in this part of the work: The annealing was done in a ribbon-wound alundum tube 18 in. long and 1½ in. in inside diameter. This resistor was covered with alundum cement and mounted in a rectangular box made of transite asbestos wood, as shown in Fig. 3.

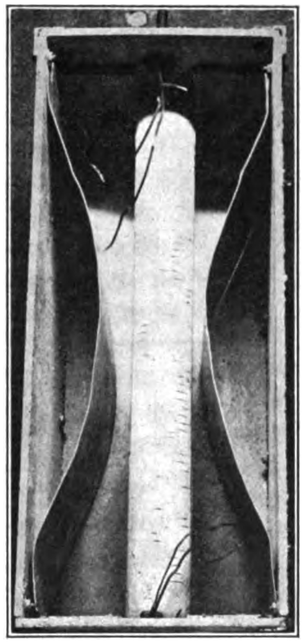

FIG. 3.—INTERIOR OF ANNEALING FURNACE WITH FILLING REMOVED.

In order to flatten the thermal gradient in the tube, the heat capacity of the furnace was increased by means of a loose filling of granular fire clay, and false inner sides bent as indicated in the figure were used to cut down the quantity of filling around the central portion of the tube (which would ordinarily run hottest). In this way, the loss of heat along the sides of the furnace was made to decrease toward the end-regions, thereby compensating for the extra loss at the ends of the furnace.

The gradients along the central 10 in. of the tube, in which the test strips were located, are shown in Fig. 4 at three different temperatures in different parts of the annealing range. It may be noted that the center

of the furnace was kept at a constant temperature (the furnace was allowed to come to equilibrium with a fixed energy input), while the gradients were taken on one side, and again, at a constant temperature in another series of tests (not exactly the same temperature as before), while the gradients were taken on the other side. Interpreted in this sense, these curves show a maximum variation from center to ends (central region of 10 in.) of 15° at approximately 300°; 11° at approximately 600°, and 22° at approximately 900° C. These gradients are made still flatter by the balancing effect of the load.

Four milled strips were annealed at once. It was therefore necessary to make three runs at a given temperature in order to handle the 12 strips

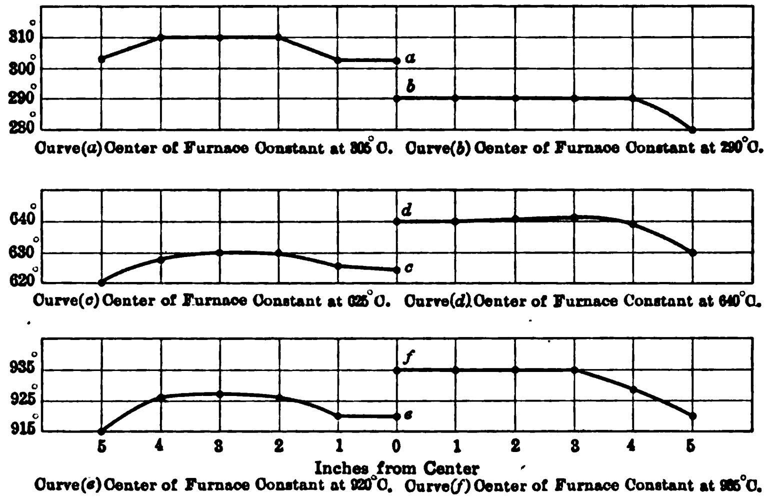


FIG. 4.—TEMPERATURE GRADIENTS OF ANNEALING FURNACE AT APPROXIMATELY 300°, 600° AND 900° C.

required (four each from three kinds of metal). We were able to make these different runs under practically identical conditions, but, in anticipation of possible variations of treatment, each load was composed of strips from the different kinds of metal, so that any differences shown by the average of tests on the several kinds of metal could not be ascribed to differences in annealing treatment. Marking the electrolytic samples 1, the Mohawk samples 2, and the Copper Range samples 3, the different loads were made up as follows:

| First Load | Second Load | Third Load |
|---|---|---|
| 1—2—3—1 | 2—3—1—2 | 3—1—2—3 |

In starting an anneal, the furnace was brought to a temperature some 10° above the actual annealing temperature sought and then the four strips were pushed into the center of the furnace tube side by side with

the end of the thermocouple tube as nearly as possible in the center of the load. When the temperature reached the desired value, attention was paid to the resistances in series with the furnace tube and such alterations as were necessary to keep the temperature constant were made from time to time on the basis of 5- or 10-min. observations of temperature.

The galvanometric method of measuring temperature was employed in connection with an ordinary platinum, platinum-rhodium thermocouple (Siemens and Halske galvanometer of 410 ohms internal resistance). The hot junction of the thermocouple remained in perfect condition throughout all of the experiments. Standardization was made at the freezing points of pure tin, 232°; zinc, 419°; antimony, 631°; and silver (in hydrogen), 961°, keeping the cold junction at 20°. The same conditions were observed in making all anneals. An empirical correction curve of the temperature scale of the galvanometer was made on the basis of the standardization results and all temperatures were corrected by interpolation. Re-standardization of the apparatus after completion of the experiments showed that constancy had been maintained within 10°.

A typical annealing curve shows a preheating period of about 20 min., during which the temperature rises gradually to the value chosen for annealing, and an annealing period of 40 min., during which the temperature varies only a few degrees. Owing to the uniformity which was attained in making these anneals, it does not appear desirable to devote space to an elaborate tabulation of the annealing data. We believe that the annealing temperatures reported are consistently accurate (making allowance for all sources of error) within 15°.

TABLE 1

| Temperature of Anneal, Degrees Centigrade | Preheating Period in Minutes | | | |
|---|---|---|---|---|
| | Load No. 1 | Load No. 2 | Load No. 3 | Average. |
| 100 water | 4 | 2 | 2 | 2.7 |
| 200 oil | 4 | 4 | 3 | 3.7 |
| 250 Electric furnace | 21 | 22 | 20 | 21.0 |
| 275 Electric furnace | 20 | 20 | 21 | 20.3 |
| 300 Electric furnace | 19 | 16 | 17 | 17.3 |
| 325 Electric furnace | 22 | 22 | 23 | 22.3 |
| 350 Electric furnace | 19 | 19 | 19 | 19.0 |
| 400 Electric furnace | 21 | 18 | 20 | 19.7 |
| 500 Electric furnace | 19 | 15 | 14 | 16.0 |
| 600 Electric furnace | 18 | 18 | 19 | 18.3 |
| 700 Electric furnace | 17 | 18 | 18 | 17.7 |
| 800 Electric furnace | 15 | 15 | 15 | 15.0 |
| 900 Electric furnace | 12 | 12 | 13 | 12.3 |
| 1,000 Electric furnace | 13 | 12 | 18 | 14.3 |

With respect to the preheating periods, which were naturally subject to more or less variation in the different experiments, a summary of the principal data appears to be desirable and this is given in Table 1.

Some of the strips of the second set were annealed in an atmosphere of illuminating (coal) gas. The approximate composition of this gas by volume is as follows:

| | Per Cent. |
|---|---|
| Hydrocarbons | 48.5 |
| Hydrogen | 25.0 |
| Carbon monoxide | 18.0 |
| Carbon dioxide | 2.0 |
| Oxygen | 0.5 |
| Nitrogen | 6.0 |

These anneals were made in a furnace similar to the one previously described, in which an iron tube was substituted for the alundum tube and both ends were closed except for a small gas inlet at one end and exit at the other end. The thermocouple was introduced at intervals through a suitable hole at one end and left in the furnace long enough to reach the temperature of its surroundings, after which it was withdrawn to avoid contamination by long exposure to the reducing gas, and the hole plugged up. Less accurate control was secured in these experiments and the preheating periods were not determined.

Strips from the different kinds of metal were mixed in each load and, aside from all questions as to the ultimate accuracy of the temperature measurements, valid comparisons between the different kinds of metal (which was the main object of all tests) were obtained.

Samples for micrographic examination (1 by 1 in.) were cut from the ends of the test pieces after annealing.

## Tabulation of Results

### (1) *Tests in the Cold-Rolled Condition. First Set of Strips*

Table 2 shows the results of tests made on strips rolled to a succession of gages beginning with No. 8 (B. & S.), viz., 0.128 in., and ending with No. 18, viz., 0.040 in. Test strips were milled according to Fig. 2. Micrometer measurements are not given in the table except in case of the thickness. The latter is of importance in indicating character of the the product with respect to temper and an average must be used in calculating the percentage reduction of area effected by rolling, while other measurements are of a routine character, as prerequisites to the calculations of strength and ductility.

The previous history of the strips is given in the sketches of Fig. 1.

TABLE 2.—*Strength and Ductility of Cold-Rolled Strips*

Average thickness before rolling: Electrolytic Copper, 0.1310 in.; Mohawk Copper, 0.1284 in.; Copper Range Copper, 0.1316 in.

| No. | Brand of Copper and Reduction by Rolling* | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 1 | Electrolytic copper, reduced 0.0 per cent. by rolling. | 0.1315 | 33,100 | 40.5 | 52.0 |
| 2 | | 0.1310 | 32,900 | 41.0 | 52.0 |
| 3 | | 0.1310 | 32,500 | 41.9 | 50.0 |
| 4 | | 0.1310 | 32,900 | 40.6 | 51.5 |
| | Average | 0.1310 | 32,850 | 41.0 | 51.4 |
| 5 | Mohawk copper reduced 0.0 per cent. by rolling. | 0.1280 | 32,700 | 43.7 | 51.5 |
| 6 | | 0.1285 | 33,700 | 44.0 | 54.5 |
| 7 | | 0.1280 | 33,500 | 45.5 | 54.5 |
| 8 | | 0.1290 | 33,100 | 45.0 | 54.0 |
| | Average | 0.1284 | 33,250 | 44.6 | 53.6 |
| 9 | Copper Range copper reduced 0.0 per cent. by rolling. | 0.1320 | 33,100 | 43.5 | 53.5 |
| 10 | | 0.1310 | 32,400 | 43.1 | 53.0 |
| 11 | | 0.1315 | 33,100 | 45.0 | 53.0 |
| 12 | | 0.1320 | 33,100 | 45.2 | 53.0 |
| | Average | 0.1316 | 32,925 | 44.2 | 53.1 |
| 13 | Electrolytic copper reduced 6.9 per cent. by rolling. | 0.1220 | 34,800 | 39.3 | 46.0 |
| 14 | | 0.1220 | 34,300 | 38.0 | 45.0 |
| 15 | | 0.1220 | 34,800 | 38.7 | 42.0 |
| 16 | | 0.1220 | 34,700 | 40.0 | 44.5 |
| | Average | 0.1220 | 34,650 | 39.0 | 44.4 |
| 17 | Mohawk copper reduced 6.2 per cent. by rolling. | 0.1200 | 34,200 | 44.3 | 45.0 |
| 18 | | 0.1200 | 34,600 | 44.0 | 44.0 |
| 19 | | 0.1205 | 35,200 | 44.1 | 42.5 |
| 20 | | 0.1210 | 34,100 | 44.5 | 45.0 |
| | Average | 0.1204 | 34,525 | 44.2 | 44.1 |
| 21 | Copper Range copper reduced 8.7 per cent. by rolling. | 0.1200 | 35,500 | 39.5 | 38.0 |
| 22 | | 0.1205 | 35,600 | 36.2 | 34.0 |
| 23 | | 0.1200 | 35,100 | 40.6 | 38.0 |
| 24 | | 0.1200 | 35,600 | 39.3 | 38.0 |
| | Average | 0.1201 | 35,450 | 38.9 | 37.0 |
| 25 | Electrolytic copper reduced 12.9 per cent. by rolling. | 0.1140 | 37,300 | 35.9 | 34.0 |
| 26 | | 0.1140 | 36,900 | 36.2 | 35.5 |
| 27 | | 0.1140 | 36,800 | 35.8 | 34.0 |
| 28 | | 0.1145 | 36,100 | 35.8 | 34.0 |
| | Average | 0.1141 | 36,775 | 35.9 | 34.4 |
| 29 | Mohawk copper reduced 11.1 per cent. by rolling. | 0.1140 | 36,800 | 40.2 | 36.0 |
| 30 | | 0.1145 | 36,600 | 40.7 | 36.5 |
| 31 | | 0.1140 | 37,000 | 40.0 | 38.0 |
| 32 | | 0.1145 | 37,100 | 39.8 | 38.0 |
| | Average | 0.1142 | 36,875 | 40.2 | 37.1 |
| 33 | Copper Range copper reduced 13.1 per cent. by rolling. | 0.1145 | 38,100 | 33.7 | 31.0 |
| 34 | | 0.1145 | 38,300 | 33.7 | 29.0 |
| 35 | | 0.1140 | 36,900 | 32.2 | 27.5 |
| 36 | | 0.1145 | 37,500 | 32.3 | 31.0 |
| | Average | 0.1144 | 37,700 | 33.0 | 29.6 |
| 37 | Electrolytic copper reduced 21.4 per cent. by rolling. | 0.1030 | 40,900 | 23.4 | 13.5 |
| 38 | | 0.1030 | 41,400 | 22.5 | 12.5 |
| 39 | | 0.1030 | 41,000 | 22.5 | 9.5 |
| 40 | | 0.1030 | 41,000 | 24.4 | 16.5 |
| | Average | 0.1030 | 41,075 | 23.2 | 13.0 |
| 41 | Mohawk copper reduced 20.2 per cent. by rolling. | 0.1025 | 41,600 | 27.3 | 16.0 |
| 42 | | 0.1025 | 40,000 | 28.1 | 14.0 |
| 43 | | 0.1020 | 40,600 | 29.8 | 16.0 |
| 44 | | 0.1025 | 40,500 | 29.8 | 16.0 |
| | Average | 0.1024 | 40,675 | 28.7 | 15.5 |
| 45 | Copper Range copper reduced 22.5 per cent. by rolling. | 0.1020 | 43,700 | 22.9 | 10.5 |
| 46 | | 0.1020 | 42,900 | 21.9 | 10.0 |
| 47 | | 0.1020 | 42,800 | 22.5 | 9.5 |
| 48 | | 0.1020 | 43,000 | 23.5 | 11.0 |
| | Average | 0.1020 | 43,100 | 22.7 | 10.3 |

* Calculated from average thickness of strips and average thickness before rolling.

TABLE 2.—*Strength and Ductility of Cold-Rolled Strips (Continued)*

**Average thickness before rolling: Electrolytic Copper, 0.1310 in.; Mohawk Copper, 0.1284 in.; Copper Range Copper, 0.1316 in.**

| No. | Brand of Copper and Reduction by Rolling* | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 49 | Electrolytic copper re- | 0.0810 | 47,900 | 10.0 | 5.0 |
| 50 | duced 38.2 per cent. | 0.0810 | 47,700 | 11.0 | 5.0 |
| 51 | by rolling. | 0.0810 | 47,900 | 7.5 | 5.0 |
| 52 | | 0.0810 | 47,500 | 8.7 | 5.0 |
| | Average........... | 0.0810 | 47,750 | 9.3 | 5.0 |
| 53 | Mohawk copper reduced | 0.0810 | 49,100 | 11.5 | 6.0 |
| 54 | 37.2 per cent. by | 0.0805 | 49,500 | 10.2 | 5.0 |
| 55 | rolling. | 0.0810 | 49,600 | 9.4 | 5.0 |
| 56 | | 0.0805 | 49,600 | 9.4 | 5.0 |
| | Average........... | 0.0807 | 49,450 | 10.1 | 5.2 |
| 57 | Copper Range copper | 0.0815 | 50,500 | 12.1 | 5.0 |
| 58 | reduced 38.1 per cent. | 0.0815 | 49,800 | 7.2 | 5.0 |
| 59 | by rolling. | 0.0815 | 50,400 | 7.3 | 4.0 |
| 60 | | 0.0815 | 50,500 | 6.2 | 5.0 |
| | Average........... | 0.0815 | 50,300 | 8.2 | 4.8 |
| 61 | Electrolytic copper re- | 0.0645 | 49,900 | 4.2 | 3.0 |
| 62 | duced 50.8 per cent. | 0.0645 | 52,400 | 5.0 | 3.0 |
| 63 | by rolling. | 0.0645 | 52,200 | 5.9 | 3.5 |
| 64 | | 0.0645 | 53,100 | 4.9 | 3.5 |
| | Average........... | 0.0645 | 51,900 | 5.0 | 3.2 |
| 65 | Mohawk copper reduced | 0.0645 | 53,800 | 3.0 | 3.5 |
| 66 | 49.8 per cent. by | 0.0645 | 52,400 | 4.4 | 3.0 |
| 67 | rolling. | 0.0645 | 53,500 | 5.0 | 3.0 |
| 68 | | 0.0645 | 52,600 | 5.2 | 3.0 |
| | Average........... | 0.0645 | 53,325 | 4.4 | 3.1 |
| 69 | Copper Range copper | 0.0640 | 54,700 | 2.8 | 4.0 |
| 70 | reduced 51.4 per cent. | 0.0640 | 54,000 | 3.6 | 4.0 |
| 71 | by rolling. | 0.0640 | 54,900 | 4.3 | 4.0 |
| 72 | | 0.0640 | 55,200 | 4.3 | 4.0 |
| | Average........... | 0.0640 | 54,700 | 3.7 | 4.0 |
| 73 | Electrolytic copper re- | 0.0510 | 54,900 | 3.9 | 3.5 |
| 74 | duced 60.8 per cent. | 0.0520 | 55,000 | 2.4 | 2.5 |
| 75 | by rolling. | 0.0510 | 51,000 | 4.8 | 2.5 |
| 76 | | 0.0515 | 56,400 | 3.2 | 3.5 |
| | Average........... | 0.0514 | 54,325 | 3.6 | 3.0 |
| 77 | Mohawk copper reduced | 0.0505 | 57,000 | 2.1 | 3.0 |
| 78 | 60.7 per cent. by | 0.0505 | 57,400 | 2.1 | 2.5 |
| 79 | rolling. | 0.0505 | 57,000 | 2.1 | 3.0 |
| 80 | | 0.0505 | 56,800 | 2.1 | 3.0 |
| | Average........... | 0.0505 | 57,050 | 2.1 | 2.9 |
| 81 | Copper Range copper | 0.0510 | 57,600 | 3.9 | 3.0 |
| 82 | reduced 61.2 per cent. | 0.0510 | 58,100 | 3.9 | 3.0 |
| 83 | by rolling. | 0.0510 | 57,900 | 3.4 | 3.0 |
| 84 | | 0.0510 | 57,400 | 4.0 | 3.0 |
| | Average........... | 0.0510 | 57,750 | 3.8 | 3.0 |
| 85 | Electrolytic copper re- | 0.0410 | 57,000 | 3.3 | 3.0 |
| 86 | duced 69.1 per cent. | 0.0400 | 57,400 | 3.7 | 3.5 |
| 87 | by rolling. | 0.0405 | 57,700 | 2.3 | 3.0 |
| | Average........... | 0.0405 | 57,367 | 3.1 | 3.2 |
| 88 | Mohawk copper reduced | 0.0405 | 57,800 | 2.6 | 2.0 |
| 89 | 68.1 per cent. by | 0.0410 | 58,400 | 2.6 | 2.0 |
| 90 | rolling. | 0.0410 | 59,000 | 2.6 | 2.0 |
| 91 | | 0.0410 | 59,000 | 2.6 | 2.0 |
| | Average........... | 0.0409 | 58,550 | 2.6 | 2.0 |
| 92 | Copper Range copper | 0.0395 | 58,800 | 2.1 | 1.5 |
| 93 | reduced 70.0 per cent. | 0.0395 | 59,300 | 1.4 | 2.0 |
| 94 | by rolling. | 0.0395 | 60,500 | 3.7 | 2.5 |
| 95 | | 0.0395 | 61,200 | 2.4 | 2.5 |
| 96 | | 0.0395 | 60,500 | 2.4 | 2.0 |
| | Average........... | 0.0395 | 60,060 | 2.4 | 2.1 |

*Calculated from average thickness of strips and average thickness before rolling.

TABLE 3.—*Strength and Ductility of Strips Annealed* 40 *Min. at Different Temperatures after a Reduction of* 50 *Per Cent. by Rolling**

Average thickness before annealing, 0.064 to 0.067 in.

| No. | Brand of Copper and Annealing Temperature | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 97 | Electrolytic copper, | 0.0640 | 51,600 | 4.56 | 4.0 |
| 98 | unannealed (as re- | 0.0645 | 51,600 | 3.20 | 4.0 |
| 99 | ceived from the mill). | 0.0670 | 51,600 | 4.57 | 4.0 |
| 100 | | 0.0640 | 51,600 | 4.56 | 4.0 |
| | Average........... | 0.0649 | 51,600 | 4.22 | 4.0 |
| 101 | Mohawk copper, unan- | 0.0670 | 52,200 | 4.94 | 4.0 |
| 102 | nealed (as received | 0.0665 | 52,100 | 5.03 | 4.0 |
| 103 | from the mill). | 0.0665 | 52,100 | 6.02 | 4.0 |
| 104 | | 0.0660 | 52,100 | 4.66 | 4.0 |
| | Average........... | 0.0665 | 52,125 | 5.16 | 4.0 |
| 105 | Copper Range copper, | 0.0665 | 53,000 | 8.83 | 4.0 |
| 106 | unannealed (as re- | 0.0665 | 53,500 | 9.00 | 4.5 |
| 107 | ceived from the mill). | 0.0665 | 53,000 | 8.83 | 4.5 |
| 108 | | 0.0665 | 52,900 | 8.60 | 4.5 |
| | Average........... | 0.0665 | 53,100 | 8.81 | 4.4 |
| 109 | Electrolytic copper an- | 0.0640 | 51,700 | 3.74 | 4.0 |
| 110 | nealed at 100°C. | 0.0640 | 51,600 | 4.56 | 4.0 |
| 111 | | 0.0640 | 51,400 | 4.56 | 4.0 |
| 112 | | 0.0640 | 51,400 | 3.96 | 4.0 |
| | Average........... | 0.0640 | 51,525 | 4.21 | 4.0 |
| 113 | Mohawk copper, an- | 0.0660 | 52,300 | 4.63 | 4.0 |
| 114 | nealed at 100°C. | 0.0655 | 52,900 | 6.86 | 5.0 |
| 115 | | 0.0660 | 52,400 | 5.01 | 4.0 |
| 116 | | 0.0660 | 52,300 | 4.63 | 4.0 |
| | Average........... | 0.0659 | 52,475 | 5.28 | 4.2 |
| 117 | Copper Range copper, | 0.0660 | 52,500 | 8.85 | 6.0 |
| 118 | annealed at 100°C. | 0.0660 | 52,500 | 8.85 | 6.0 |
| 119 | | 0.0660 | 53,300 | 7.65 | 4.5 |
| 120 | | 0.0640 | 54,400 | 4.77 | 4.0 |
| | Average........... | 0.0655 | 53,175 | 7.53 | 5.1 |
| 121 | Electrolytic copper, an- | 0.0650 | 49,000 | 9.86 | 4.5 |
| 122 | nealed at 200°C. | 0.0645 | 49,100 | 9.49 | 4.5 |
| 123 | | 0.0650 | 48,600 | 6.12 | 4.0 |
| 124 | | 0.0645 | 50,500 | 5.15 | 6.0 |
| | Average........... | 0.0647 | 49,300 | 7.66 | 4.8 |
| 125 | Mohawk copper, an- | 0.0660 | 52,900 | 9.45 | 6.5 |
| 126 | nealed at 200°C. | 0.0660 | 52,000 | 8.85 | 5.0 |
| 127 | | 0.0660 | 52,100 | 7.45 | 5.5 |
| 128 | | 0.0660 | 52,000 | 7.45 | 5.5 |
| | Average........... | 0.0660 | 52,250 | 8.30 | 5.6 |
| 129 | Copper Range copper, | 0.0640 | 53,100 | 5.81 | 5.5 |
| 130 | annealed at 200°C. | 0.0655 | 52,500 | 12.95 | 4.5 |
| 131 | | 0.0650 | 52,700 | 12.85 | 4.5 |
| 132 | | 0.0660 | 51,800 | 13.13 | 4.5 |
| | Average........... | 0.0651 | 52,525 | 11.18 | 4.8 |

* Calculated values of reduction vary between 49 and 51 per cent.

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

| No. | Brand of Copper and Annealing Temperature | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 133 | Electrolytic copper, annealed at 250°C. | 0.0645 | 49,200 | 20.80 | 9.0 |
| 134 | | 0.0645 | 49,100 | 19.30 | 9.0 |
| 135 | | 0.0650 | 48,500 | 19.05 | 8.5 |
| 136 | | 0.0645 | 48,900 | 18.30 | 9.0 |
| | Average........... | 0.0646 | 48,925 | 19.36 | 8.9 |
| 137 | Mohawk copper, annealed at 250°C. | 0.0657 | 51,800 | 15.30 | 7.5 |
| 138 | | 0.0650 | 51,800 | 14.85 | 7.5 |
| 139 | | 0.0655 | 51,700 | 14.00 | 7.5 |
| 140 | | 0.0652 | 51,700 | 14.25 | 8.0 |
| | Average........... | 0.0654 | 51,750 | 14.60 | 7.6 |
| 141 | Copper Range copper, annealed at 250°C. | 0.0655 | 52,500 | 12.00 | 6.0 |
| 142 | | 0.0660 | 51,500 | 14.15 | 8.0 |
| 143 | | 0.0660 | 53,000 | 14.15 | 8.0 |
| 144 | | 0.0660 | 52,600 | 14.45 | 7.5 |
| | Average........... | 0.0659 | 52,400 | 13.69 | 7.4 |
| 145 | Electrolytic copper, annealed at 275°C. | 0.0643 | 43,000 | 24.40 | 19.0 |
| 146 | | 0.0650 | 43,000 | 24.60 | 19.0 |
| 147 | | 0.0635 | 43,600 | 22.80 | (14.0) |
| 148 | | 0.0650 | 42,500 | 24.60 | 20.0 |
| | Average........... | 0.0645 | 43,025 | 24.10 | 19.3 |
| 149 | Mohawk copper, annealed at 275°C. | 0.0650 | 51,200 | 16.65 | 8.5 |
| 150 | | 0.0658 | 50,700 | 15.65 | 8.5 |
| 151 | | 0.0655 | 50,700 | 15.40 | 7.0 |
| 152 | | 0.0657 | 50,700 | 16.00 | 8.5 |
| | Average........... | 0.0655 | 50,825 | 15.92 | 8.1 |
| 153 | Copper Range copper, annealed at 275°C. | 0.0653 | 52,200 | (9.45) | (5.0) |
| 154 | | 0.0655 | 51,500 | 11.00 | 9.0 |
| 155 | | 0.0655 | 51,000 | 11.00 | 9.0 |
| 156 | | 0.0655 | 51,200 | 11.00 | 9.0 |
| | Average........... | 0.0654 | 51,475 | 11.00 | 9.0 |
| 157 | Electrolytic copper, annealed at 300°C. | 0.0640 | 38,800 | 36.90 | 36.5 |
| 158 | | 0.0640 | 37,600 | 40.20 | 36.0 |
| 159 | | 0.0650 | 35,400 | (44.00) | (52.5) |
| 160 | | 0.0640 | 37,700 | 38.00 | 37.5 |
| | Average........... | 0.0647 | 37,375 | 38.37 | 36.7 |
| 161 | Mohawk copper, annealed at 300°C. | 0.0655 | 50,700 | 10.75 | 11.0 |
| 162 | | 0.0660 | 50,400 | 15.65 | 11.0 |
| 163 | | 0.0660 | 50,000 | (8.85) | 11.0 |
| 164 | | 0.0660 | 50,100 | 14.10 | 11.0 |
| | Average........... | 0.0659 | 50,300 | 13.50 | 11.0 |
| 165 | Copper Range copper, annealed at 300°C. | 0.0660 | 50,200 | 14.50 | 9.5 |
| 166 | | 0.0645 | 49,800 | 13.40 | 11.5 |
| 167 | | 0.0660 | 51,000 | 14.35 | 10.0 |
| 168 | | 0.0655 | 50,400 | 15.00 | 9.0 |
| | Average........... | 0.0655 | 50,350 | 14.31 | 10.0 |

TABLE 3.—*Strength and Ductility of Strips Annealed* 40 *Min. at Different Temperatures after a Reduction of* 50 *Per Cent. by Rolling* (*Continued*)

| No. | Brand of Copper and Annealing Temperature | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 169 | Electrolytic copper, an- | 0.0650 | 34,600 | 44.60 | 46.0 |
| 170 | nealed at 325°C. | 0.0650 | 35,100 | 43.90 | 46.0 |
| 171 | | 0.0630 | 34,600 | 42.00 | 49.0 |
| 172 | | 0.0650 | 35,600 | 43.10 | (43.0) |
| 173 | | 0.0645 | 33,800 | 44.60 | 49.5 |
| | Average........... | 0.0645 | 34,740 | 43.64 | 47.6 |
| 174 | Mohawk copper, an- | 0.0650 | 43,000 | 27.90 | 19.0 |
| 175 | nealed at 325°C. | 0.0645 | 43,500 | 28.40 | 19.0 |
| 176 | | 0.0684 | 43,500 | 30.70 | 18.0 |
| 177 | | 0.0650 | 41,800 | 29.60 | 22.0 |
| | Average........... | 0.0657 | 42,950 | 29.15 | 19.5 |
| 178 | Copper Range copper, | 0.0655 | 37,600 | 39.90 | 42.0 |
| 179 | annealed at 325°C. | 0.0655 | 37,550 | 39.80 | 41.0 |
| 180 | | 0.0655 | 36,800 | 40.20 | 43.0 |
| 181 | | 0.0640 | 38,200 | 39.10 | 42.0 |
| | Average........... | 0.0651 | 37,540 | 39.75 | 42.0 |
| 182 | Electrolytic copper, an- | 0.0650 | 33,900 | 45.90 | 50.0 |
| 183 | nealed at 350°C. | 0.0630 | 34,300 | 45.00 | 51.0 |
| 184 | | 0.0645 | 34,100 | 46.30 | 52.0 |
| 185 | | 0.0650 | 33,900 | 46.40 | 51.0 |
| | Average........... | 0.0644 | 34,050 | 45.90 | 51.0 |
| 186 | Mohawk copper, an- | 0.0663 | 34,500 | 44.00 | 54.0 |
| 187 | nealed at 350°C. | 0.0650 | 34,200 | 45.40 | 54.0 |
| 188 | | 0.0655 | 34,700 | 45.40 | 53.0 |
| 189 | | 0.0655 | 34,700 | 43.30 | 54.0 |
| | Average........... | 0.0656 | 34,525 | 44.53 | 53.8 |
| 190 | Copper Range copper, | 0.0650 | 34,900 | 43.00 | 54.0 |
| 191 | annealed at 350°C. | 0.0655 | 34,700 | 42.60 | 54.0 |
| 192 | | 0.0660 | 35,100 | 43.00 | 54.0 |
| 193 | | 0.0655 | 34,500 | 42.20 | 53.5 |
| | Average........... | 0.0655 | 34,800 | 42.70 | 53.9 |
| 194 | Electrolytic copper, an- | 0.0640 | 34,100 | 44.60 | 51.0 |
| 195 | nealed at 400°C. | 0.0645 | 34,000 | 45.10 | 55.0 |
| 196 | | 0.0645 | 33,900 | 45.00 | 54.0 |
| 197 | | 0.0645 | 33,000 | 45.00 | 51.5 |
| | Average........... | 0.0644 | 33,750 | 44.93 | 52.9 |
| 198 | Mohawk copper, an- | 0.0660 | 34,400 | 44.50 | 53.5 |
| 199 | nealed at 400°C. | 0.0660 | 34,400 | 45.00 | 53.5 |
| 200 | | 0.0660 | 34,100 | 45.50 | 53.5 |
| 201 | | 0.0660 | 34,500 | 44.90 | 54.0 |
| | Average........... | 0.0660 | 34,350 | 44.98 | 53.6 |
| 202 | Copper Range copper, | 0.0655 | 34,700 | 47.60 | 54.0 |
| 203 | annealed at 400°C. | 0.0655 | 34,600 | 46.50 | 53.5 |
| 204 | | 0.0645 | 34,200 | 47.10 | 54.0 |
| 205 | | 0.0657 | 34,400 | 46.10 | 54.5 |
| | Average........... | 0.0650 | 34,475 | 46.83 | 54.0 |

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

| No. | Brand of Copper and Annealing Temperature | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 206 | Electrolytic copper, an- | 0.0635 | 33,500 | 45.10 | 57.5 |
| 207 | nealed at 500°C. | 0.0635 | 33,500 | 46.50 | 56.0 |
| 208 | | 0.0635 | 33,200 | 44.50 | 49.0 |
| 209 | | 0.0640 | 33,500 | 46.00 | 57.0 |
| | Average........... | 0.0636 | 33,425 | 45.53 | 54.9 |
| 210 | Mohawk copper, an- | 0.0640 | 33,200 | 46.80 | 49.5 |
| 211 | nealed at 500°C. | 0.0655 | 33,900 | 49.30 | 57.0 |
| 212 | | 0.0660 | 33,500 | 49.50 | 56.5 |
| 213 | | 0.0650 | 32,800 | 49.60 | 55.0 |
| | Average........... | 0.0651 | 33,350 | 48.80 | 54.8 |
| 214 | Copper Range copper, | 0.0655 | 33,500 | 47.90 | 55.0 |
| 215 | annealed at 500°C. | 0.0645 | 33,400 | 49.00 | 58.0 |
| 216 | | 0.0645 | 33,700 | 45.50 | 54.5 |
| 217 | | 0.0645 | 33,500 | 48.50 | 58.0 |
| | Average........... | 0.0648 | 33,525 | 47.73 | 56.4 |
| 218 | Electrolytic copper, an- | 0.0645 | 33,400 | 46.90 | 54.5 |
| 219 | nealed at 600°C. | 0.0640 | 33,300 | 46.10 | 54.5 |
| 220 | | 0.0640 | 33,300 | 46.40 | 55.0 |
| 221 | | 0.0620 | 33,800 | 45.40 | 55.0 |
| | Average........... | 0.0636 | 33,450 | 46.20 | 54.8 |
| 222 | Mohawk copper, an- | 0.0645 | 33,200 | 48.40 | 55.5 |
| 223 | nealed at 600°C. | 0.0645 | 33,100 | 47.00 | 54.0 |
| 224 | | 0.0645 | 32,900 | 49.60 | 56.0 |
| 225 | | 0.0655 | 33,000 | 49.40 | 56.5 |
| | Average........... | 0.0648 | 33,050 | 48.60 | 55.5 |
| 226 | Copper Range copper, | 0.0655 | 33,100 | 49.40 | 57.0 |
| 227 | annealed at 600°C. | 0.0655 | 33,000 | 48.60 | 57.0 |
| 228 | | 0.0660 | 33,250 | 51.50 | 57.5 |
| 229 | | 0.0655 | 33,200 | 51.10 | 56.0 |
| | Average........... | 0.0656 | 33,140 | 50.15 | 56.9 |
| 230 | Electrolytic copper, an- | 0.0635 | 32,200 | 44.70 | 52.5 |
| 231 | nealed at 700°C. | 0.0630 | 31,800 | 44.90 | 55.5 |
| 232 | | 0.0640 | 32,199 | 45.00 | 55.0 |
| 233 | | 0.0635 | 32,400 | 46.40 | 55.0 |
| | Average........... | 0.0635 | 32,125 | 45.25 | 54.5 |
| 234 | Mohawk copper, an- | 0.0645 | 31,900 | 49.50 | 54.0 |
| 235 | nealed at 700°C. | 0.0647 | 32,200 | 47.90 | 61.0 |
| 236 | | 0.0645 | 32,200 | 49.30 | 55.0 |
| 237 | | 0.0645 | 32,100 | 49.20 | 53.5 |
| | Average........... | 0.0646 | 32,050 | 48.98 | 55.9 |
| 238 | Copper Range copper, | 0.0650 | 32,300 | 50.60 | 55.5 |
| 239 | annealed at 700°C. | 0.0645 | 32,100 | 50.10 | 59.0 |
| 240 | | 0.0643 | 32,000 | 49.70 | 55.0 |
| 241 | | 0.0645 | 31,900 | 49.70 | 57.0 |
| | Average........... | 0.0646 | 32,075 | 50.03 | 56.6 |

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

| No. | Brand of Copper and Annealing Temperature | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 242 | Electrolytic copper, an- | 0.0615 | 32,400 | 44.00 | 49.0 |
| 243 | nealed at 800°C. | 0.0615 | 32,200 | 44.30 | 51.0 |
| 244 | | 0.0600 | 32,100 | 42.80 | 51.0 |
| 245 | | 0.0615 | 32,500 | 44.00 | 50.5 |
| | Average........... | 0.0615 | 32,300 | 43.78 | 50.4 |
| 246 | Mohawk copper, an- | 0.0615 | 32,100 | 44.80 | 52.0 |
| 247 | nealed at 800°C. | 0.0620 | 32,200 | 45.70 | 53.0 |
| 248 | | 0.0630 | 31,800 | 45.80 | 53.0 |
| 249 | | 0.0630 | 31,400 | 46.30 | 51.0 |
| | Average........... | 0.0624 | 31,875 | 45.65 | 52.3 |
| 250 | Copper Range copper, | 0.0620 | 32,200 | 46.60 | 55.0 |
| 251 | annealed at 800°C. | 0.0625 | 32,100 | 46.00 | 55.0 |
| 252 | | 0.0625 | 32,100 | 47.40 | 59.0 |
| 253 | | 0.0613 | 31,400 | 47.10 | 50.0 |
| | Average........... | 0.0621 | 31,950 | 46.79 | 54.8 |
| 254 | Electrolytic copper, an- | 0.0600 | 31,600 | 39.80 | 44.0 |
| 255 | nealed at 900°C. | 0.0615 | 31,200 | 41.40 | 44.0 |
| 256 | | 0.0620 | 32,200 | 41.10 | 46.0 |
| 257 | | 0.0600 | 31,800 | 39.80 | 44.0 |
| | Average........... | 0.0609 | 31,700 | 40.53 | 44.5 |
| 258 | Mohawk copper, an- | 0.0640 | 30,100 | 46.90 | 47.0 |
| 259 | nealed at 900°C. | 0.0620 | 30,600 | 42.40 | 45.0 |
| 260 | | 0.0605 | 30,600 | 44.60 | 46.0 |
| 261 | | 0.0620 | 31,500 | 45.80 | 44.0 |
| | Average........... | 0.0621 | 30,700 | 44.93 | 45.5 |
| 262 | Copper Range copper, | 0.0625 | 30,200 | 48.50 | 54.5 |
| 263 | annealed at 900°C. | 0.0620 | 30,200 | 47.50 | 51.0 |
| 264 | | 0.0610 | 30,500 | 45.60 | 45.0 |
| 265 | | 0.0610 | 30,900 | 46.50 | 48.0 |
| | Average........... | 0.0616 | 30,450 | 47.03 | 49.6 |
| 266 | Electrolytic copper, an- | 0.0600 | 29,700 | 37.16 | 35.5 |
| 267 | nealed at 1,000°C. | 0.0600 | 30,500 | 32.10 | 31.0 |
| 268 | | 0.0565 | 30,900 | 33.30 | 30.0 |
| 269 | | 0.0570 | 32,300 | 31.90 | 42.5 |
| | Average........... | 0.0589 | 30,850 | 33.62 | 34.8 |
| 270 | Mohawk copper, an- | 0.0600 | 30,400 | 40.40 | 43.0 |
| 271 | nealed at 1,000°C. | 0.0600 | 29,800 | 40.40 | 38.0 |
| 272 | | 0.0605 | 29,900 | 40.00 | 37.0 |
| 273 | | 0.0590 | 30,200 | 39.40 | 36.0 |
| | Average........... | 0.0599 | 30,075 | 40.05 | 38.5 |
| 274 | Copper Range copper, | 0.0610 | 28,800 | 40.90 | 35.0 |
| 275 | annealed at 1,000°C. | 0.0580 | 29,700 | 38.82 | 37.0 |
| 276 | | 0.0585 | 29,100 | 38.60 | 36.0 |
| 277 | | 0.0605 | 27,600 | 41.50 | 37.0 |
| | Average........... | 0.0595 | 28,800 | 39.96 | 36.3 |

(2) *Tests after Anneal under Oxidizing Conditions. Second Set of Strips.*

Table 3 shows the results of tests made on strips annealed 40 min. at temperatures ranging from 100° to 1,000°C. The previous history of the strips is given in the sketches of Fig. 1. The methods of annealing are described in the preceding section. Preheating periods are given in Table 1.

While all strips were annealed at gage No. 14 (B. & S.), the thickness is somewhat reduced by oxidation after anneal at high temperatures and, in order that comparisons of this sort may be made, micrometer measurements of thickness are given in the table.

(3) *Tests after Anneal under Reducing Conditions. Second Set of Strips*

Table 4 shows the results of a few tests made on strips annealed 40 min. at 600°, 800°, and 1,000° C., respectively, in an atmosphere of illumi-

TABLE 4.—*Strength and Ductility of Strips Annealed* 40 *Min. at Different Temperatures in an Atmosphere of Coal Gas**

Average thickness, before annealing, 0.064 to 0.067 in.

| No. | Brand of Copper and Annealing Temperature | Thickness, Inches | Ultimate Strength in Pounds per Sq. In. | Percentage Reduction of Area | Percentage Elongation in 2 In. |
|---|---|---|---|---|---|
| 278 | Electrolytic copper, an- | 0.0690 | 29,400 | Broke at shoulder | (22.0) |
| 279 | nealed at 600° C. | 0.0680 | 29,100 | Broke at shoulder | (22.0) |
| | Average........... | 0.0685 | 29,250 | ............... | (22.0) |
| 280 | Mohawk copper, an- | 0.0690 | 29,500 | Broke at shoulder | (30.0) |
| 281 | nealed at 600° C. | 0.0670 | 30,500 | Broke at shoulder | (25.5) |
| | Average........... | 0.0680 | 30,000 | ............... | (27.8) |
| 282 | Copper Range copper, | 0.0670 | 31,800 | 39.20 Good breaks | 41.0 |
| 283 | annealed at 600° C. | 0.0670 | 31,700 | 39.40 Good breaks | 43.0 |
| | Average........... | 0.0670 | 31,750 | 39.30 Good breaks | 42.0 |
| 284 | Electrolytic copper, an- | 0.0710 | 21,500 | 12.35 Good break | 12.0 |
| 285 | nealed at 800° C. | 0.0710 | 17,700 | Broke in several | (9.0) |
| | Average........... | 0.0710 | 19,600 | pieces | (10.5) |
| 286 | Mohawk copper, an- | 0.0700 | 17,600 | Broke at shoulder | (8.0) |
| 287 | nealed at 800° C. | 0.0690 | 19,000 | Broke in jaws | .... |
| | Average........... | 0.0695 | 18,300 | | |
| 288 | Copper Range copper, | 0.0695 | 20,700 | 12.80 Good breaks | 11.0 |
| 289 | annealed at 800° C. | 0.0700 | 21,700 | Broke at shoulder | (11.0) |
| | Average........... | 0.0698 | 21,200 | ............... | 11.0 |
| 290 | Electrolytic copper, an- | 0.0670 | 21,900 | Broke at shoulder | (14.5) |
| 291 | nealed at 1,000° C. | 0.0710 | 22,300 | Broke at shoulder | (14.5) |
| | Average........... | 0.0690 | 22,100 | ............... | (14.5) |
| 292 | Mohawk copper, an- | 0.0690 | 21,300 | Broke in pieces | (16.5) |
| 293 | nealed at 1,000° C. | 0.0700 | 21,700 | 18.15 Good breaks | 17.0 |
| | Average........... | 0.0695 | 21,500 | ............... | (16.8) |
| | | | | | ..... |
| 294 | Copper Range copper, | 0.0690 | 22,100 | Broke in jaws | |
| 295 | annealed at 1,000° C. | 0.0690 | 22,100 | 16.70 Good breaks | 16.0 |
| | Average........... | 0.0690 | 22,100 | ................ | ...... |

* Calculated values of reduction vary between 49 and 51 per cent.

nating gas. The previous history and dimensions of the strips are the same as in the case of the preceding tests.

## Discussion of Tables, Curves, and Microstructure

### (1) *Tests after Cold-Rolling*

The results assembled in Table 2 have been plotted in Figs. 5, 6, and 7. In Fig. 5, the comparative values of tensile strength in the three kinds of metal are shown; in Fig. 6, the comparative values of elongation; and in Fig. 7, the comparative values of reduction of area.

It will be observed that changes in temper of about the same degree are produced by rolling the three kinds of metal through any given value of reduction. In other words, these tests fail to develop any striking differences of rolling temper between the different brands of copper used. The difficulty of obtaining a high degree of accuracy in the ordinary tensile test is a handicap in comparing products which are very similar in nature. Testing discrepancies of several per cent. will ordinarily occur with careful handling of test strips from the same sheet of metal. The uneven distribution of small percentages of impurity in commercial products is a source of irregularities in this respect. As may be seen in the micrographs (Fig. *d*, Plate 4, is typical of this condition), the oxide particles, originally more or less segregated in the cast cake, are drawn out into continuous trains in the rolled product and it is obvious that there will be a closer grouping of the zones, or attenuated lines of weakness in some cross-sectional areas than in others. Furthermore, the presence of oxide interferes with the normal growth of grain on annealing, and, consequently, there will be irregularities of grain size in the copper matrix corresponding to segregations of oxide. Such a condition is shown admirably in Fig. *a*, Plate 4, where coarser grain has formed on both sides of an extended zone or band sprinkled with oxide particles.[9] Such conditions cannot fail to be responsible for variations in breaking load, elongation, and reduction of area among the duplicate test strips.

An accurate average (representing the mean properties of the metal) could, of course, be struck from tests on a large number of strips. There is always a limit in this direction, however, and we decided upon four strips as the largest number which could be handled in this investigation. It will be noted from examination of the tabulated results that considerable variations both of strength and elongation were obtained in a given set of tests. It may be left to the individual reader to form his own opinion as to the actual value of the averages reported.

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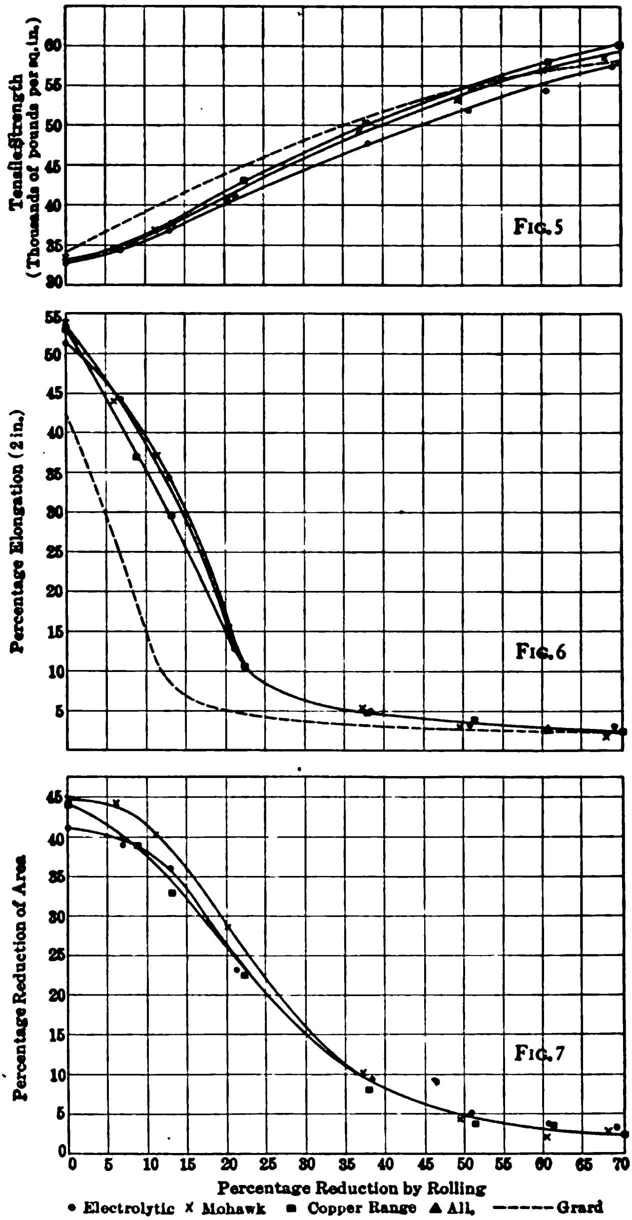
[9] Mathewson and Caesar, in a paper lately sent to the *Internationale Zeitschrift für Metallographie*, have discussed the effect of oxide on the size of recrystallized grain in pure copper.

In rejecting any seemingly abnormal result, we endeavored to be perfectly fair to each kind of metal. In general, where serious irregulari-




FIGS. 5, 6, AND 7.—STRENGTH AND DUCTILITY OF ROLLED STRIPS.

ties developed, an entirely new set of bars was tested and all of the earlier results rejected; in no case were four especially favorable or unfavorable

results selected from a long series of tests, but, in some cases, averages were struck from three tests when there seemed good reason to reject the fourth.

Such differences between the three varieties of copper as appear to be real, although small, may be summarized in the following statements:

The Copper Range product decreases in ductility as measured by elongation more rapidly than the electrolytic product during the early stages of rolling (down to about 20 per cent. reduction of area by rolling), but both products increase in strength about equally, while, during the later stages of rolling, both products decrease about equally in ductility and the Copper Range metal increases in strength more rapidly than the electrolytic metal. The Mohawk copper occupies a more or less intermediate position.

Differences in ductility as measured by reduction of area are also confined to the earlier range of reductions by rolling. While, in principle, the measurement of reduction of area possesses some advantages over the measurement of elongation as a criterion of deformational properties, since the former is more nearly independent of the shape of the test piece, it is far more difficult to make in the case of thin flat metal. Our measurements of elongation are, therefore, more reliable than the corresponding measurements of reduction of area. In the present case, all three kinds of material show practically the same values of reduction of area after the reduction by rolling has passed 35 per cent.

From the general point of view that the Copper Range product takes a harder temper on rolling than the electrolytic product, but does not ultimately become less ductile, it would seem that both might be able to stand the same amount of punishment by rolling, but that the latter might roll a little easier. Since the pressure exerted in rolling cannot be calculated from the results of the ordinary tensile test, and since it would require, in any event, comparative determinations of the yield-point stress intensities at intervals throughout the rolling range in order to determine which of the two materials would require the greatest load to start permanent deformation at any stage, the above statement must remain subject to eventual qualification by actual test.

Where it is desired to finish a previously annealed product with a limited amount of ductility, but increased strength, by rolling through several numbers of gage, it is quite evident that an appreciable increase in strength without sacrifice of ductility may be secured by using Copper Range metal instead of electrolytic metal. Thus, rolling from No. 8 to No. 12, B. & S. gage, a reduction of about 40 per cent., would give the Copper Range material a tensile strength about 1 ton (per square inch) greater than that of the electrolytic material similarly treated, while the elongation in both cases would be in the neighborhood of 5 per cent.

In order that the well-known curves of Grard[10] (electrolytic copper) may be compared with our own, they have been dotted into Figs. 5 and 6. Grard's measurements of elongation were made on a test section 100 mm. long (approximately 4 in.) and, for this reason, show much lower values than our own, measured in 2 in., until the region of slight ductility is reached at about a 35 per cent. reduction by rolling. From this point on, the stretch becomes sensibly uniform along the entire test length, making it of little consequence whether a length of 2 or 4 in. is chosen, and our results are practically coincident with Grard's.

Grard's curve of breaking strength starts somewhat above and ends somewhat below our curves. Thus, our curves show a higher rate of increase of strength with the degree of reduction by rolling. This is doubtless due to differences in the dimensions of the test strips. For reasons previously given, we used test strips whose thickness decreased progressively with the increase in reduction by rolling (minimum ratio of test length to $\sqrt{\text{sectional area}}$, 6½ to 1; maximum ratio, 11½ to 1), while Grard's practice in this respect is not clear. He shows a ratio of 5 to 1 in a sketch of the test piece, but later states that a variable final cross-section was used.

### (2) *Tests after Annealing under Oxidizing Conditions*

The results assembled in Table 3 have been plotted in Figs. 8 (strength), 9 (elongation), and 10 (reduction of area). For general purposes of comparison, Grard's[11] curves from tests on 2-mm. strips of electrolytic copper and Bardwell's[12] curves from tests on No. 12, B. & S. wire, containing from 0.036 to 0.070 per cent. of oxygen and an indeterminate amount of silver, are shown in dotted form.

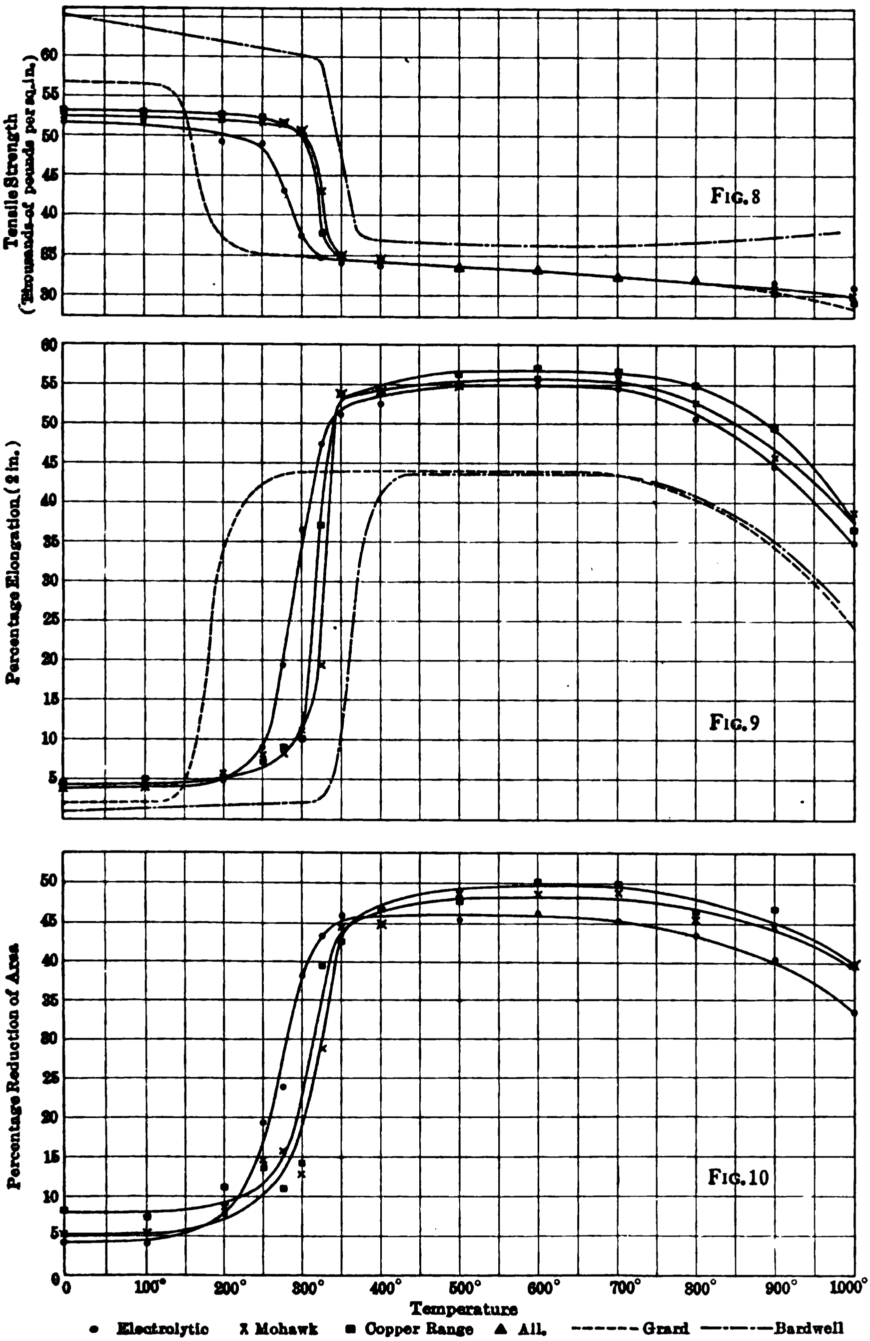
Naturally, one's attention is first drawn to the difference between electrolytic copper, on the one hand, and the arsenical coppers, on the other hand, with respect to the earliest manifestation of an annealing effect. The temperature range of early transformation (annealing period of 40 min.) from the strain-hardened condition to the recrystallized, or annealed, condition is equally narrow in all three kinds of material; it is practically coincident in the two arsenical coppers, but lies some 50° lower in the electrolytic copper.

It is believed that these are the first systematic tests designed to reveal the relative softening (annealing) characteristics of commercially pure copper and commercial copper containing arsenic. The general tendency of arsenic to raise the effective annealing temperature has long

[10] *Loc. cit.*

[11] *Loc. cit.*

[12] E. S. Bardwell: The Annealing of Cold-Rolled Copper, *Trans.*, vol. 49, pp. 753–773 (1914).




FIGS. 8, 9, AND 10.—STRENGTH AND DUCTILITY OF ANNEALED STRIPS.

been known on the basis of fragmentary tests, here and there, under a variety of conditions difficult to bring into any sort of alignment. Thus, F. Johnson[13] quotes from Tomlinson to the effect that samples of rolled arsenical copper containing 0.36 per cent. of arsenic did not soften after 3 or 4 weeks at a temperature approaching 400° F. (205° C.), while samples of pure copper were practically annealed. Further, the arsenical copper lost only 6 per cent. of its strength after 4 hr. at 870° C., while the pure copper lost 44 per cent. (*i.e.*, was presumably completely annealed).

The first statement above is quite in harmony with the conditions shown by our tests, viz., the range of softening by recrystallization from the strain-hardened condition is always carried to lower temperatures by prolonging the anneal. In softening, the arsenical metal lags behind the pure copper (temperature-lag of 50° C. for an annealing period of 40 min. in the case of metal containing 0.3 per cent. of arsenic, neglecting the possible effect of silver) and the above result signifies that the stated temperature of 205° C. lies well within the temperature range of softening based upon an annealing period of several weeks in the case of pure copper, while, in the case of the arsenical metal, it lies below, or barely within, the temperature range of softening based upon a similar annealing period.

It is perfectly clear, without undertaking special experiments involving variation of the annealing period, that arsenical coppers, such as the Copper Range product, constitute a commonly available and satisfactory material—superior to pure copper—when it is desired to preserve a rolled temper in the face of considerable variation of temperature up to some 200° C.

The second statement taken from Mr. Johnson's paper, which seems to indicate a difficulty in annealing the arsenical metal, cannot be brought into harmony with the results of our tests. We find that the separate temperature ranges of early softening for the three kinds of material end at a very nearly common (minimum) temperature of complete softening (some 325° C.) from which point onward, there are only minor differences in the annealing characteristics of the different kinds of metal. It is inconceivable that any of these products should lose only 6 per cent. of their strength by annealing for a period of hours at red heat. Such an effect must be attributed to the somewhat higher arsenic content (of doubtful application), or to other unknown peculiarities of composition.

As far as the present comparisons go, we are able to state that the proper annealing temperature for electrolytic copper (dull red) will produce equally satisfactory results in the case of Copper Range or Mohawk copper.

Outstanding features of comparison between the three varieties of copper in the annealed condition may be summarized as follows:

[13] F. Johnson: The Influence of Impurities on the Properties of Copper, *Metallurgical and Chemical Engineering*, vol. 8, pp. 570–575 (October, 1910).

Only minor differences in strength throughout the entire annealing range of 400 to 1,000° C. were observed. As a rule, after a given anneal, the difference between extreme values of tensile strength (averages) was less than 1,000 lb. per square inch.

The Copper Range product consistently showed the highest values of elongation and reduction of area and the electrolytic product the lowest, with the Mohawk product occupying a more or less intermediate position.

It is important to observe that, in the temperature region of deterioration, which begins in the neighborhood of 700° with all three varieties, the relative ductilities of the Copper Range and electrolytic products are such that the former will stand an extra 100°, or so, of overheating before it drops to the ductility value of the latter. This means that if the metal is allowed to come to a bright red heat in annealing, a Copper Range load may be expected to emerge with about the same ductility as an electrolytic load annealed at the proper temperature of low redness, while an electrolytic load under the same conditions would presumably fall below this standard.

It will be noticed that Grard's curve of tensile strength practically coincides with our curves after an annealing temperature of about 300° is reached, while Bardwell's curve lies somewhat above ours, as would be expected in the case of wires, but rises in the region of high temperature anneals for some obscure reason. The tests from these different sources show marked discrepancies with respect to the temperature ranges of initial softening. Since the location of this range (*zone de detente* in Grard's phraseology) depends upon the period of anneal and the degree of preliminary cold-working, certain discrepancies would be expected, although hardly of the order shown in these comparisons. As may be seen from the initial strength values, both Grard and Bardwell started their tests with material which had been more severely reduced (by rolling or drawing) than our material. Making allowance for the higher strengths commonly shown by wires, it appears that the materials used by both of these authors had received approximately the same degree of cold-working and, under similar annealing conditions, both should show a lower range of initial softening than our material. Bardwell kept his wires 20 min. in the furnace and Grard kept his strips 30 min. in the furnace, as far as we can determine from his general description of procedure. Our material was reduced 50 per cent. by rolling and we used an annealing period of 40 min., actual time of heating at the annealing temperature.

We do not know how far the presence of a small amount of silver might have effected Bardwell's results, but, on a basis of equivalent purity, the temperature ranges of initial softening found by both authors should not be greatly dissimilar and they should lie somewhat behind the corresponding range found by us in the case of electrolytic copper. As matters stand, the range determined in our experiments occupies an

intermediate position. Not enough definite information is at present available to permit an harmonious interpretation of these irregularites. It is particularly to be desired that determinations of the equilibrium positions of the temperature range of initial softening of rolled electrolytic copper for various reductions be made.

Our curves of elongation similarly occupy an intermediate position between those of Bardwell and Grard. Owing to the divergent ratios of test length to diameter (or to $\sqrt{\text{sectional area}}$), the actual values of elongation vary within wide limits. A purely fortuitous coincidence between Grard's and Bardwell's values is noted at annealing temperatures above 400°. This is due to the fact that Grard used a ratio of approximately 22 to 1 (length to $\sqrt{\text{sectional area}}$) and Bardwell, a ratio of approximately 150 to 1 (length to diameter), which choice of dimensions appears to have compensated for the flatter gradient of stretch of the wires, yielding the same percentage values of elongation.

All of the curves show practically the same temperature range of deterioration (*zone de fléchissement*, in Grard's phraseology), viz., from 700° upward.

### (3) *Tests after Annealing under Reducing Conditions*

Bengough and Hill[14] obtained some strikingly destructive effects by annealing certain arsenical mixtures in reducing gases, which led them to seek for an explanation in the constitutional relationships between copper, arsenic, and oxygen. Further tests by the same authors and also by Johnson[15] and others have established the fact that both arsenical and non-arsenical coppers containing oxygen suffer deterioration by "gassing" when given a reducing anneal above moderate red heat, while deoxidized copper is not subject to this disadvantage. It is thus reasonably certain that the effect in question is due to an ordinary reduction of cuprous oxide by hydrogen or other gases which can permeate (*i.e.* dissolve in) solid copper, with formation of (bulky) gaseous products, thereby causing fissures and cavities to develop.

We have endeavored to ascertain the relative destructive effects which occur on annealing the three kinds of copper in hand under typical reducing conditions at several representative temperatures. Accordingly, the test strips were annealed at 600°, 800°, and 1,000° C. in an iron tube through which coal gas was passed in sufficient quantity to burn freely at the discharge end, notwithstanding a certain amount of decomposition, with formation of soot, in the tube.

On examination of Table 4, in which the results of these tests are to

[14] *Loc. cit.*

[15] F. Johnson: A Method of Improving the Quality of Arsenical Copper, *Journal of the Institute of Metals*, vol. 10, pp. 283–4 (1913).

be found, it will be seen that, of the whole lot of strips tested, only one set, viz., that composed of strips from the Copper Range material annealed at 600°, showed a fair retention of ductility. Here, an elongation of 42 per cent. and a reduction of area of 39.3 per cent. replace the normal values of 56.9 and 50.15, respectively, attained after an oxidizing anneal at the same temperature. The other materials after anneal at this temperature gave unsatisfactory breaks and lower values of ductility, as far as could be ascertained. Destructive effects—brittleness and an almost entire lack of elongation—were obtained throughout in the case of reducing anneals at 800° and 1,000°. As far as may be judged from the small number of tests in which good breaks were secured, the anneals at 1,000° were somewhat less destructive than the anneals at 800°. This was doubtless due to the protective influence of a dense coating of carbonaceous material which was deposited on the strips at the highest annealing temperature.

All fractures showed a lack of luster and an approximation to the brick-red color of fractures from "set" copper.

It seemed of considerable interest to determine the oxygen content of the "gassed" strips in the usual way, viz., by ignition in hydrogen. Accordingly, representative samples were sent to the laboratory of the Michigan Smelting Co. for this purpose. Table 5 embodies the results of analyses from this source.

TABLE 5.—*Oxygen Content of Annealed Copper*

| No. of Test Strip (See Table 4) | Kind of Copper | Temperature of Anneal, Degrees Centigrade | Oxygen Content,* Per Cent. |
|---|---|---|---|
| 278 | Electrolytic | 600 | 0.0456 |
| 280 | Mohawk | 600 | 0.0441 |
| 282 | Copper Range | 600 | 0.0402 |
| 284 | Electrolytic | 800 | 0.0035 |
| 286 | Mohawk | 800 | 0.0060 |
| 288 | Copper Range | 800 | 0.0082 |
| 290 | Electrolytic | 1,000 | 0.0054 |
| 293 | Mohawk | 1,000 | 0.0010 |
| 294 | Copper Range | 1,000 | 0.0053 |

In view of the small percentages of oxygen found in the material annealed at 800° and 1,000°, it is obviously not justifiable to attach great importance to the differences which may be observed in comparing the several results. The analytical data are in general agreement with the testing results. In particular, the figures indicate partial deoxidation of the material at 600° and almost complete deoxidation at 800° and 1,000°.

* The initial oxygen content of the different kinds of copper was as follows: Electrolytic, 0.071 per cent.; Mohawk, 0.052 per cent.; Copper Range, 0.055 per cent.

In general, the tests in this section indicate that the Copper Range material is somewhat less susceptible to the evil effects of a reducing anneal than either of the other varieties of copper. On the other hand, they show that none of these coppers will retain its ductility unimpaired after a 40-min. period of anneal under actively reducing conditions at a temperature even as low as 600°.

It is clearly of great importance to regulate carefully the combustion in all furnaces used for heating copper preparatory to hot-rolling, or for annealing copper after cold-working (unless the metal has previously been deoxidized).

### *Microstructure*

Pieces were detached from the ends of the test strips, representing each separate anneal, for microscopic examination. Photomicrographs are reproduced in the accompanying plates in sets of three (electrolytic, Mohawk, and Copper Range copper, respectively, to correspond with the separate anneals at intervals of 100°, beginning with 200° and ending with 1,000°.

Figs. *a*, *b*, and *c*, of Plate 1, taken from strips of electrolytic, Mohawk, and Copper Range copper, respectively, annealed at 200°, are thoroughly representative of the structure shown by the cold-worked metal and by metal annealed at any temperature short of the softening point. The ordinary characteristics of severe deformation, viz., elongation of the grains in the direction of rolling and wavy lines or bands of deformation at right angles to the direction of rolling, are plainly evident in these photomicrographs.

The curves of tensile strength, elongation, and reduction of area (Figs. 8, 9, and 10, respectively) call for far-reaching recrystallization of the electrolytic copper and hardly more than incipient recrystallization of the other varieties at the next temperature of anneal, 300°. Accordingly, Fig. *d*, Plate 1, shows the presence of numerous minute secondary grains along with unaltered fragments of the primary grains—some of them still marked by lines of deformation, while Figs. *e* and *f* show very little evidence of recrystallization at this magnification (×77).[16]

Complete recrystallization has occurred in all cases at 400° and Plates 2 to 5, inclusive, show the growth of grain which takes place as the temperature of anneal is increased by increments of 100° between 400° and 1,000°.

Baucke[17] has published counting data which indicate that grain

[16] A satisfactory examination for the early appearance of recrystallization can only be made at moderately high magnifications. For the sake of uniformity, however, we have prepared all photomicrographs at the magnification which has proved most serviceable in the general work.

[17] H. Baucke: On Some Recent Micrographical Investigations of Copper, *International Association for Testing Materials, 6th Congress (New York)*, Section $II_{14}$, vol. 2.

growth on annealing strain-hardened copper (of unknown purity) increases gradually up to 700°, but much more rapidly in the temperature region, 700 to 1,000° C.

The photomicrographs of the present set show conditions quite in harmony with the above generalization which may thus be extended to include the three varieties of copper used in these tests.

We have not undertaken to prepare counting data from these photomicrographs, believing that the principal elements of difference are such as to render qualitative comparisons more valuable than attempted quantitative ones. To take the extreme cases, it is difficult to compare the grain size of the electrolytic samples with that of the Copper Range samples, because of greater variations in the former material. These major variations are due to the presence of finely subdivided oxide particles which are concentrated in certain portions of the copper matrix. In such regions, the mere presence of the oxide particles hinders coalescence between adjoining grains and reduces the final grain size.[18]

The form and distribution of these oxide segregations, as well as the relative conditions presented by the two kinds of copper, are best discussed with reference to their genesis in the cast material.

Fig. *b*, Plate 6, shows at the left a section through primary dendrites of copper in a dark ground mass of eutectic. Planimetric analysis shows an oxygen content of 0.14 per cent.; equivalent to 1.26 per cent. of cuprous oxide. From the regularity of arrangement, it is probable that most of the light areas are parts of the same dendritic growth, *i.e.*, that they are not separate grains, but are sections from the same grain with a eutectic filling in the interstices.

If we increase the quantity of primary copper at the expense of the eutectic, as may be done by lowering the oxide content, the light-grain sections increase in area, changing their contours somewhat; the eutectic filling becomes attenuated until, with an oxygen content of 0.04 per cent. (rather low for commercial tough-pitch copper), a condition similar to that shown at the right in Fig. *b*, Plate 6, is obtained. The connection between the two portions of this figure is easily seen. The filling in of the interstices with pure copper has obscured the principal features of regularity which could be observed at the earlier stage of dendritic growth, but it is highly probable that most of the light sections seen in the right-hand micrograph are parts of the same grain, *i.e.*, possess the same specific orientation.

The eutectic filling is more massive in some places than in others. Where it is concentrated, an intensive rolling operation will result in a relatively broad band of eutectic, *i.e.*, more or less parallel trains of oxide particles in their copper matrix, and, where it is attenuated, narrow bands will result.

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[18] See Mathewson and Caesar, *loc. cit.*

In the present experiments, we started with a planed cake, 2 in. thick, and ended with test strips 0.064 in. thick. Hence, the cake was elongated to some 30 times its original length as a result of the various rolling operations, and equiaxial patches of eutectic are ultimately replaced by bands of varying breadth and thickness depending upon their original dimensions.

A broad band of this sort is seen in Fig. *a*, Plate 4. It is very noticeable that the copper grains in this region are smaller than in adjacent regions free from oxide. Other bands of varying width may be seen in other photomicrographs of the present collection.

In the case of the Copper Range material, as cast (Fig. *a*, Plate 6), three important structural observations may be made: (1) there is no eutectic, (2) a cored structure (shadowy markings) appears, showing variation in internal concentration with respect to arsenic, and (3) the individual oxide particles, which are located with some regularity in the ultimate interstices corresponding to final solidification between and within the grains, are considerably larger than the eutectic particles of oxide in the electrolytic material.[19]

The early annealing of this metal removes all traces of the cored structure and no indication of variable internal composition was observed in the product as received from the rolling mill.

Some fundamental questions of constitution in the system, copper, arsenic, oxygen, have not yet been answered. It is clear that, up to some 3 per cent., the arsenic exists as a solid solution, probably of copper arsenide ($Cu_3As$) in copper[20] and that arsenic does not deoxidize copper, but exerts a coarsening and scattering effect on the oxide particles,[21] as mentioned above. This latter effect may be due to a greatly increased solubility of cuprous oxide in arsenical copper, whereby the whole amount of oxide (present in tough-pitch metal) remains dissolved until almost all of the copper is frozen, and finally separates out in coarse form from this concentrated solution (as a eutectic containing an unusually high percentage of oxide).

Whatever the constitutional reasons for these facts, it is quite evident that the trains of oxide particles, formed when Copper Range metal is rolled, are always made up of coarse units which do not oppose grain growth as effectively as the highly dispersed form of oxide present in electrolytic copper. Consequently, irregularities of distribution do not,

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[19] These could not, as a rule, be seen as separate particles under the magnification generally adopted. They are closely sprinkled throughout the dark structure element (eutectic) shown in both parts of Fig. *b*, Plate 6. The conclusions reached above were verified by careful examination of the samples in question at adequate magnifications.

[20] Bengough and Hill: *loc. cit.*

[21] Jolibois and P. Thomas: Du Rôle de l'Arsenic dans les Cuivres industriels, *Revue de Métallurgie*, vol. 10, pp. 1264–1270 (1913).

in this case, result in great local variations of grain size, as in the case of electrolytic copper.

The principal conclusions drawn from our structural observations are:

(*a*) From the standpoint that uniformity of recrystallized grain size is a mark of quality in an annealed product, metal rolled from the ordinary Copper Range cake is superior to metal rolled from the ordinary electrolytic cake containing about the same percentage of oxide.

(*b*) With regard to the specific effect of included oxide particles on the properties of the finished sheet, bar, or tube, if a small number of coarse particles are less harmful than a large number of fine particles, the Copper Range product is superior to the electrolytic product. Jolibois and Thomas[22] take this view as a matter of course. We are not prepared to make any definite assertion in this respect. It is not improbable that with one form of oxide distribution the metal may be particularly adaptable to one type of service, while the other form may be preferable when the metal must meet requirements of another type.[23]

A fair sample of tube manufactured from Copper Range cake (by hot-rolling, cupping, and closing in) is illustrated by Fig. *c*, Plate 6.

Typical photomicrographs representing the structure of test strips spoiled by annealing in coal gas are shown in Figs. *d* and *f*, Plate 5. The first of these photos shows a sample of electrolytic copper annealed at 800° and photographed at the usual magnification of 68 diameters. This may be compared with Fig. *a*, Plate 4, representing the same material annealed normally at this temperature.

By careful examination of these photomicrographs, it may be seen that many of the boundaries between grains of the "gassed" metal (Fig. *d*, Plate 5) have a dotted appearance, while, in the normally annealed metal (Fig. *a*, Plate 4), these boundaries are close and sharp (the oxide dots are sprinkled at random without regard to the grain boundaries). Observations of this sort are best made at higher magnifications, as illustrated by Figs. *e* and *f*, Plate 5, taken from Copper Range metal annealed at 800° in air and coal gas, respectively. The first figure shows several trains of unaltered oxide particles (magnification, 450), while the last figure shows an abundance of gas cavities located, for the most part, in the grain boundaries. No unaltered oxide (distinguishable by its color, etc.) could be seen in the samples ruined by drastic annealing in coal gas, *i.e.*, in those annealed at 800° and 1,000°, while, in the samples annealed at 600° some unaltered oxide could be seen, along with the gas cavities resulting from reduction of the remainder.

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[22] *Loc. cit.*

[23] Doubtless, in every case, the metal is improved by eliminating oxide. Arsenical copper can be deoxided with phosphorous, the commonest of deoxidizing agents, without excessive loss of arsenic. It is worthy of note that the producers of Copper Range copper profess to be able to bring the metal to pitch with a lower content of oxide than is commonly attained in the case of non-arsenical copper.

Those who have seen Campbell's[24] recent micrograph of over-poled copper may recall its close resemblance to our last micrograph (Fig. *f*, Plate 5) of copper ruined by anneal in coal gas. Deterioration, in both cases, is due to the development of gas within the grains and along the grain boundaries.

Aside from the comparative observations already described, we have made no special study of the effects of annealing under reducing, or other exceptional conditions. It is beyond the scope of this paper to discuss the results obtained by other authors in this particular field.

Our observations indicate that the oxide contained in any of these brands of copper is decomposed by coal gas, even at temperatures of low redness. Presumably, such components of coal gas as can travel freely in copper at red heat (certainly hydrogen, probably carbon monoxide, and perhaps certain hydrocarbons) effect an ordinary reduction of the oxide, with formation of bulky gaseous products which seem to accumulate in the grain boundaries,[25] but also occur abundantly within the grain substance.

The character of the fractures indicates that the tensile break occurs very largely at the grain boundaries through the gas cavities, whose lining seems to be devoid of luster.

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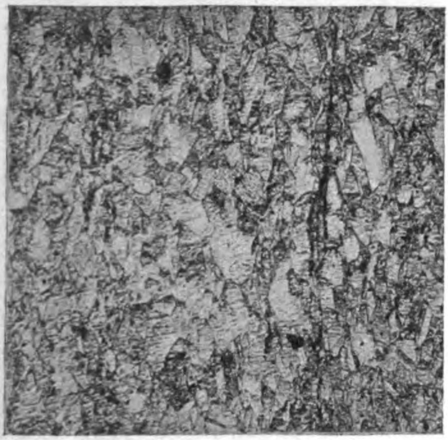
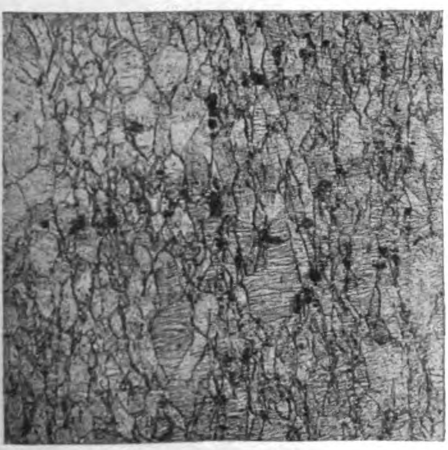
[24] *Loc. cit.*

[25] G. D. Bengough and D. Hanson's tensile tests (*Journal Institute of Metals*, vol. 12, pp. 56–88, 1914) have shown that these boundaries are favored regions of weakness at high temperatures (the grains pull apart at the boundaries), which indicates that gas evolved from within would find its path of least resistance along the boundaries.

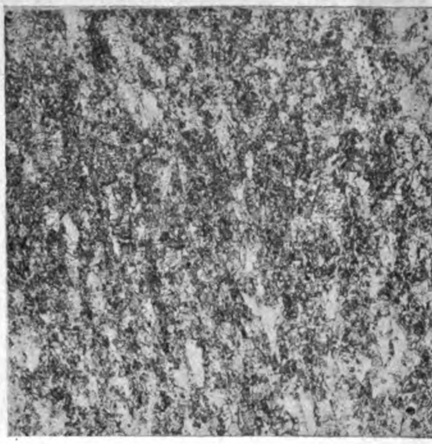
*a.*—Electrolytic Copper, 200° C. × 68.



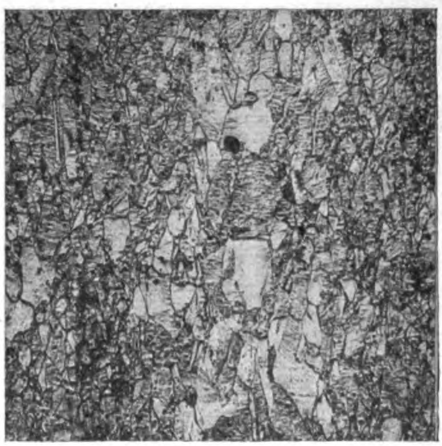
*b.*—Mohawk Copper, 200° C. × 68.



*c.*—Copper Range Copper, 200° C. × 68.



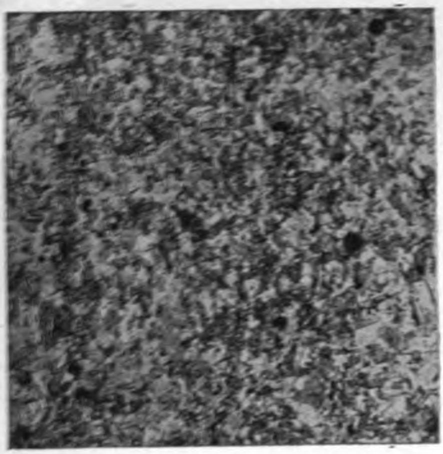
*d.*—Electrolytic Copper, 300° C. × 77.



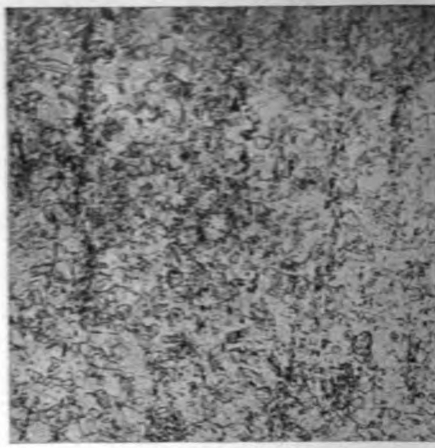
*e.*—Mohawk Copper, 300° C. × 77.

*f.*—Copper Range Copper, 200° C. × 77.

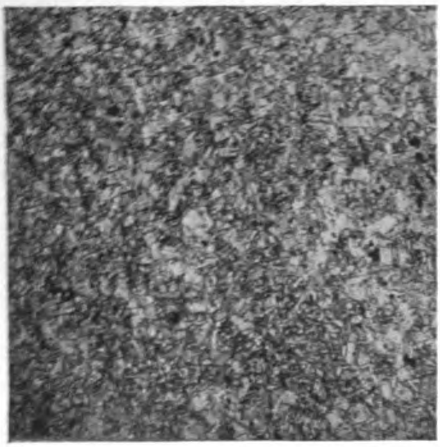
PLATE 1.—COPPER ANNEALED AT 200 AND 300° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



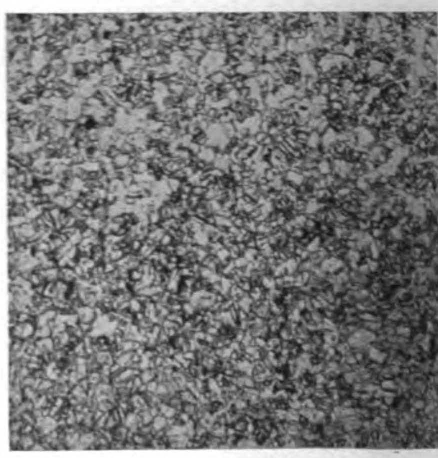
a.—Electrolytic Copper, 400° C. × 68.



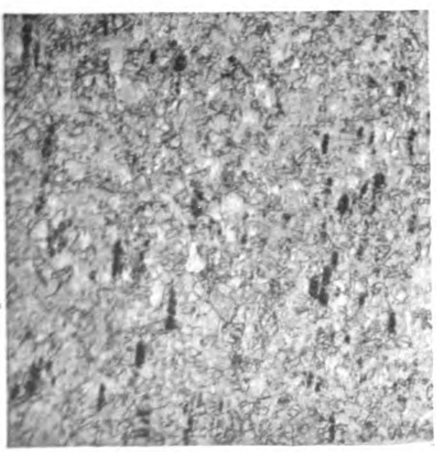
d.—Electrolytic Copper, 500° C. × 68.



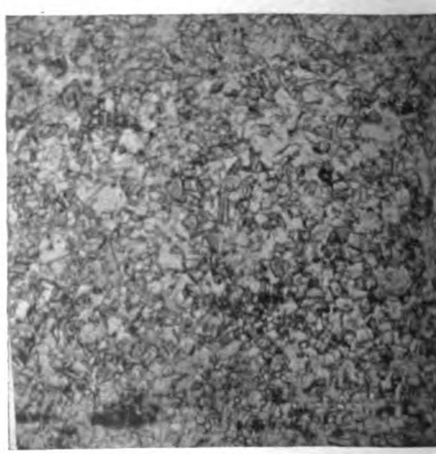
b.—Mohawk Copper, 400° C. × 68.



e.—Mohawk Copper, 500° C. × 68.



c.—Copper Range Copper, 400° C. × 68.



f.—Copper Range Copper, 500° C. × 68.

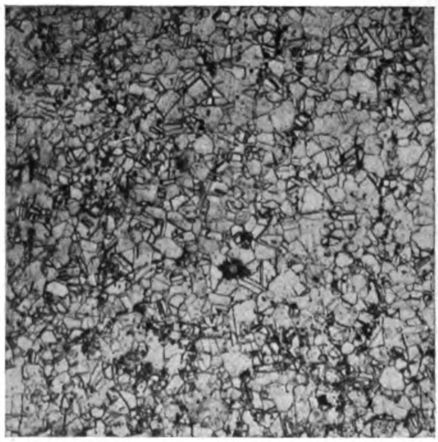
PLATE 2.—COPPER ANNEALED AT 400 AND 500° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.

*a.*—Electrolytic Copper, 600° C. × 68.

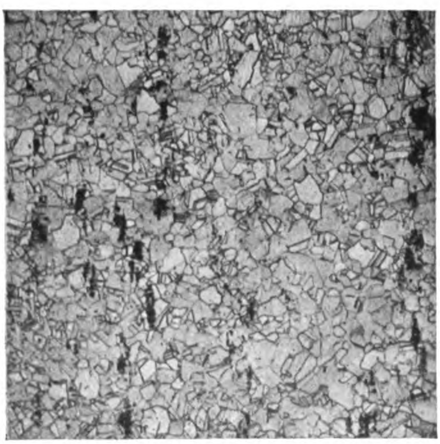
*b.*—Mohawk Copper, 600° C. × 68.

*c.*—Copper Range Copper, 600° C. × 68.

*d.*—Electrolytic Copper, 700° C. × 68.

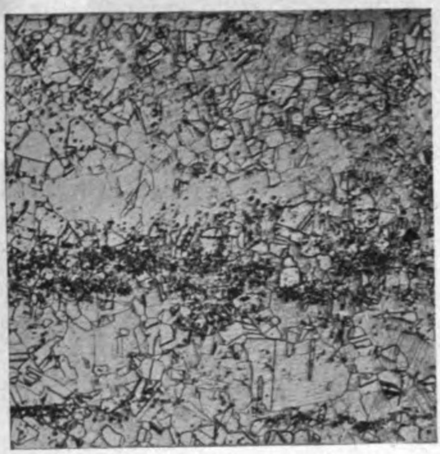


*e.*—Mohawk Copper, 700° C. × 68.

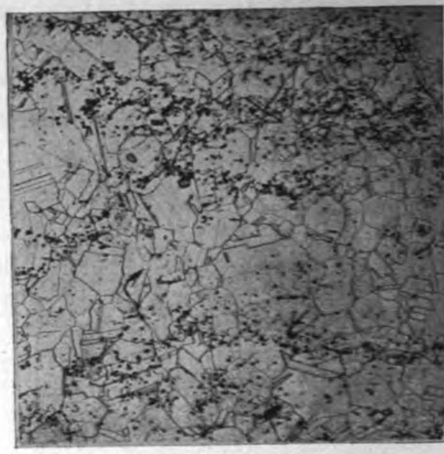


*f.*—Copper Range Copper, 700° C. × 68.

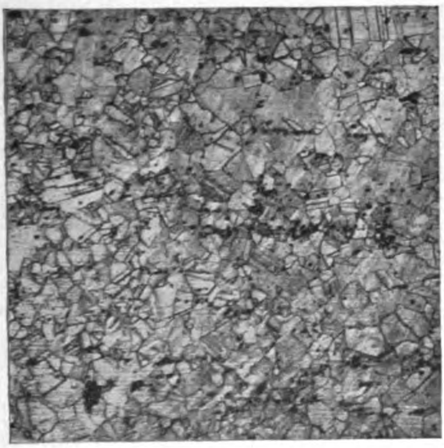
PLATE 3.—COPPER ANNEALED AT 600 AND 700° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



*a.*—Electrolytic Copper, 800° C. × 68.

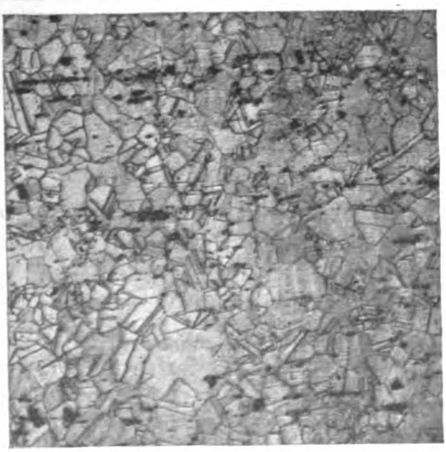


*d.*—Electrolytic Copper, 900° C. × 68.



*b.*—Mohawk Copper, 800° C. × 68.

*e.*—Mohawk Copper, 900° C. × 68.

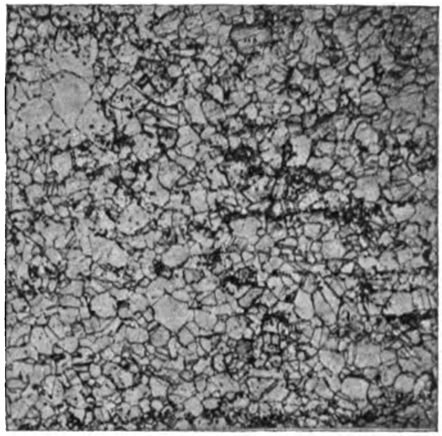


*c.*—Copper Range Copper, 800° C. × 68.

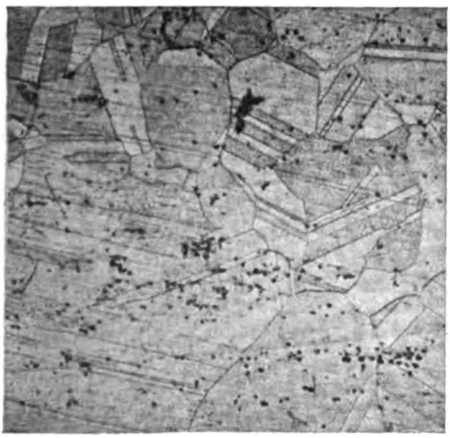
*f.*—Copper Range Copper, 900° C. × 68.

PLATE 4.—COPPER ANNEALED AT 800 AND 900° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.

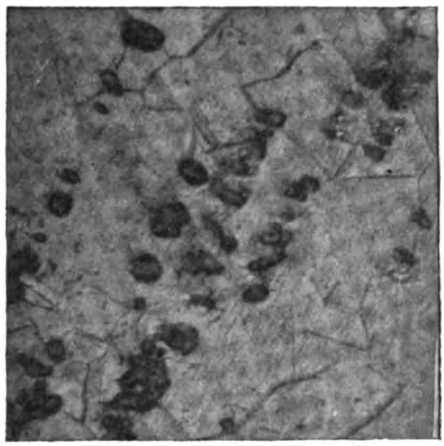
*a.*—Electrolytic Copper, 1000° C. ×68.



*d.*—Electrolytic Copper, Annealed at 800° C. in Coal Gas. × 68.

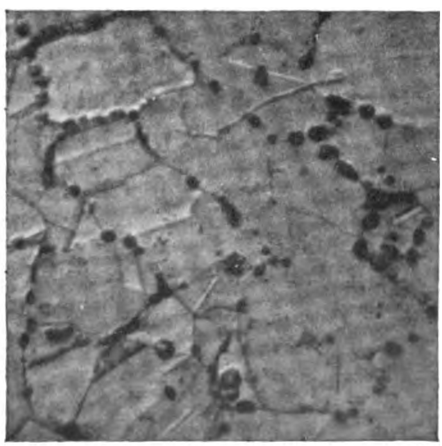


*b.*—Mohawk Copper, 1000° C. × 68.



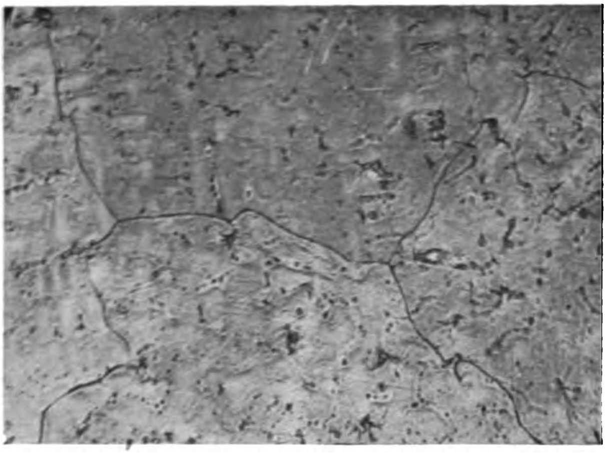
*e.*—Copper Range Copper, Annealed at 800° C. in Air. × 450.

*c.*—Copper Range Copper, 1000° C. × 68.

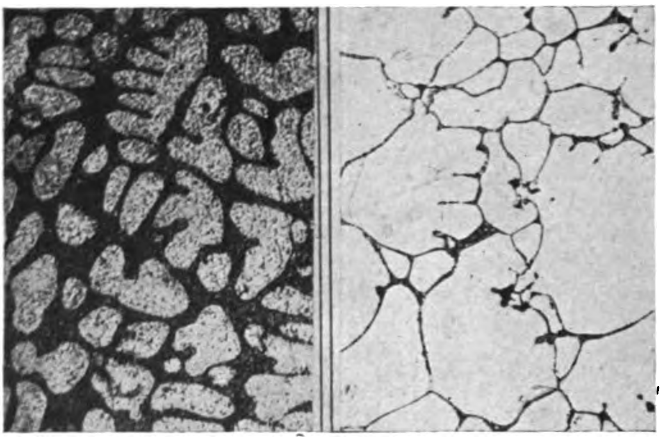


*f.*—Copper Range Copper, Annealed at 800° C. in Coal Gas. × 450.

PLATE 5.—COPPER ANNEALED AT 1000° C IN AIR, AND SAMPLES ANNEALED AT 800° C. IN AIR AND COAL GAS. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.

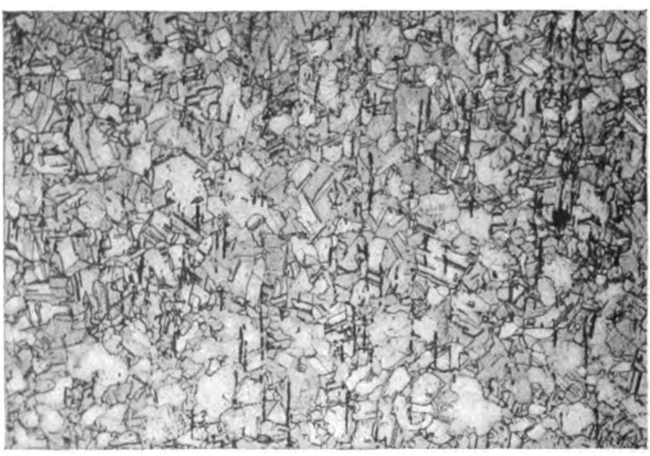


*a.*—Copper Range Copper as Cast. 0.055 Per Cent. Oxygen.



*b.*—Electrolytic Copper as Cast.

0.14 Per Cent. Oxygen. 0.04 Per Cent. Oxygen.



*c.*—Annealed Tube from Copper Range Copper.

PLATE 6.—SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 69.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Tungsten-Molybdenum Equilibrium Diagram and System of Crystallization

BY ZAY JEFFRIES,* B.S., MET. E., CLEVELAND, OHIO

(Arizona Meeting, September, 1916)

In this paper, it is proposed to outline a method for the determination of melting points of those metals and alloys having high fusion temperatures. The application of the method as used to determine the melting points of alloys of tungsten and molybdenum will be given. It will be seen from these results that tungsten and molybdenum form a completely

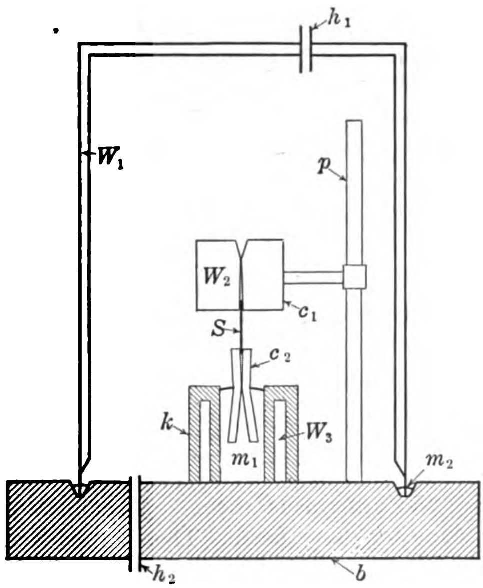



Fig. 1.—Partial Section of Apparatus Used to Determine Melting Points of Tungsten and Molybdenum Alloys.

isomorphous series. This is also verified by the accompanying photomicrographs.

Tungsten and molybdenum have been found to crystallize in the same system, namely, isometric, and the crystal units are cubes. These determinations were made by examining etching pits in the pure metals.

* Case School of Applied Science.

*Description of Apparatus*

Fig. 1 shows a partial section of the apparatus used for determining the melting points of the alloys. $W_1$ is a water-jacketed metal housing with a mercury seal $m_2$ at the bottom. $h_1$ is a hydrogen inlet and $h_2$ a hydrogen outlet. $c_1$ is the top electrode which is water-cooled, as indicated by $W_2$. This electrode is supported by the post $p$, to which it is fastened by means of an adjustable sleeve. $c_2$ is the bottom electrode which floats in a bath of mercury $m_1$. This mercury is contained in the casting $k$, which is water-cooled, as shown by $W_3$. The whole apparatus is mounted on the base $b$ made of electrical insulating material. The electrodes are suitably connected to a current regulator, and the voltmeter and ammeter are properly inserted in the circuit.

*Method of Making Determinations*

Wires ($S$), 2¾ in. between electrode contacts and 0.030 in. in diameter, were inserted in the electrodes and hydrogen was allowed to flow through the inclosed chamber until the air was expelled. Electric current was then passed through the wire, starting with low amperage and gradually increasing until fusion occurred. The time of fusion is easily determined because the circuit is broken and the ammeter ceases to register. The readings of the voltmeter and ammeter are recorded with each increase in electric current so the fusion wattage can be accurately calculated.

Besides using samples of the same size, other conditions must be maintained constant; for example, the flow of hydrogen through the inclosed chamber, the flow of water through each of the water jackets, and the temperature of the water at the intake. The electrical resistance of the electrodes and their connections is so small that it can be neglected in the calculation for drop of potential when compared to the drop of potential between the two electrodes.

*Calibration of the Apparatus using Samples of Known Melting Points*

For calibrating the apparatus wires of pure platinum, pure molybdenum and pure tungsten were used. These wires were all of the same size and same length. The number of watts to fuse each is shown in Table 1.

TABLE 1

| Material | Fusion Wattage |
|---|---|
| Platinum | 144 |
| Molybdenum | 400 |
| Tungsten | 1,800 |

By considering the fusion wattage of tungsten as 100 per cent., the fusion wattage of molybdenum is 22.2 per cent. and of platinum, 8 per cent.

The curve in Fig. 2 is obtained by plotting the following temperatures against the corresponding fusion wattage percentages:

TABLE 2

| Room Temperature, Degrees Centigrade | Per Cent. of Tungsten Fusion Wattage |
|---|---|
| 20 | 0.0 |
| 1,755 | 8.0 |
| 2,500 | 22.2 |
| 3,300 | 100.0 |

The temperature 3,300° C. is recommended as the most probable figure for the melting point of tungsten by Dr. Langmuir,[1] and also by the Nela Laboratories.[2]

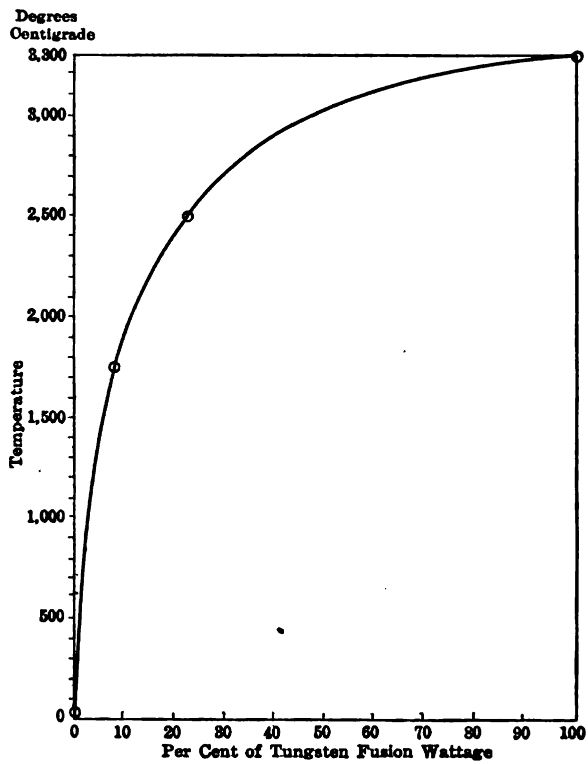



FIG. 2.—PER CENT. OF TUNGSTEN FUSION WATTAGE AT DIFFERENT TEMPERATURES.

If a sample of unknown melting point, having the same diameter and same length as the standardizing samples, be fused in the apparatus, its fusion temperature can be interpolated from its fusion wattage, from

[1] I. Langmuir: Melting Point of Tungsten, *Physical Review*, Series 2, pp. 138–157 (August, 1915).

[2] A. G. Worthing: The True Temperature Scale for Tungsten and its Emissive Powers at Incandescent Temperatures, *Journal of the Franklin Institute*, vol. 181, No. 3, pp. 417–418 (March, 1916).

the curve shown in Fig. 2. A discussion will be found below outlining the possible sources of error in the determination of melting points by this method.

### *Results Obtained with Tungsten-Molybdenum Alloys*

The results are shown in Table 3, and the points on the equilibrium diagram are shown in Fig. 3.

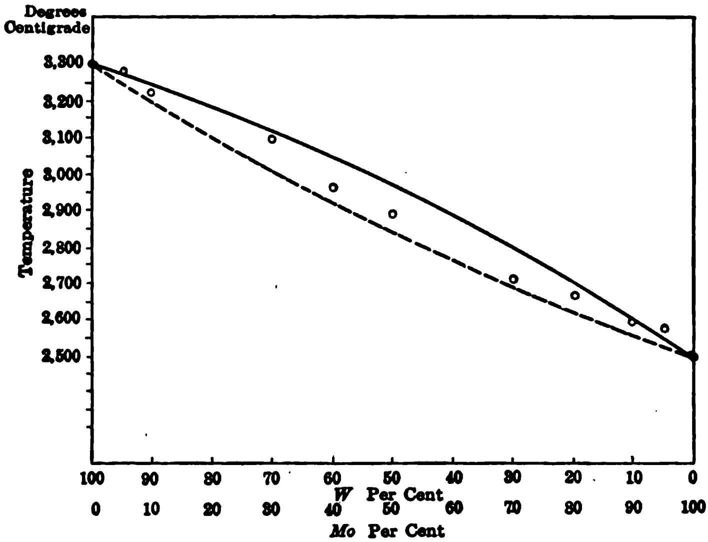



FIG. 3.—EQUILIBRIUM DIAGRAM OF TUNGSTEN AND MOLYBDENUM.

TABLE 3.—*Fusion Results Obtained with Tungsten-Molybdenum Alloys*

| Molybdenum, Per Cent. | Tungsten, Per Cent. | Watts to Fuse | Per Cent. of Tungsten Fusion Wattage | Fusion Temperature, Degrees Centigrade |
|---|---|---|---|---|
| 100 | 0 | 400 | 22.2 | 2,500 |
| 99 | 1 | 410 | 22.8 | 2,510 |
| 95 | 5 | 445 | 24.7 | 2,580 |
| 90 | 10 | 452 | 25.2 | 2,600 |
| 80 | 20 | 509 | 28.25 | 2,670 |
| 70 | 30 | 541 | 30.1 | 2,710 |
| 50 | 50 | 710 | 39.4 | 2,890 |
| 40 | 60 | 797 | 44.2 | 2,960 |
| 30 | 70 | 1,003 | 57.2 | 3,100 |
| 10 | 90 | 1,341 | 74.5 | 3,220 |
| 5 | 95 | 1,650 | 91.7 | 3,280 |
| 1 | 99 | 1,768 | 98.2 | 3,295 |
| 0 | 100 | 1,800 | 100.0 | 3,000 |

The solidus and liquidus curves are drawn tentatively, it being, of course, impossible to determine either of these curves by this method.

The point of breakdown of the alloys under the action of the electric current probably lies between the solidus and the liquidus. Most of the points shown in the diagram in Fig. 3 lie between the proposed solidus and liquidus curves. The fact that two points lie outside of the area included between these two curves may be explained by saying that the accuracy of any of the determinations may be subject to an error of from 30 to 50° C.

As an example to show how the temperature is obtained from the fusion wattage, let us consider the alloy containing 50 per cent. tungsten and 50 per cent. molybdenum. The fusion wattage of this alloy was 710, and the fusion wattage of tungsten was 1,800.

$$\frac{710}{1{,}800} \times 100 = 39.4$$

This alloy fuses, therefore, at 39.4 per cent. of the fusion wattage of tungsten. Referring to Fig. 2, it will be seen that 39.4 per cent. of the fusion wattage of tungsten corresponds to a temperature of about 2,890° C.

The equilibrium diagram suggests that tungsten and molybdenum form a series of alloys which are completely soluble in each other, both in the liquid and solid states, and that the melting point of any alloy of the series will lie between the melting points of the end members. The photomicrographs confirm this indication.

Mennicke[3] observes that two compounds between tungsten and molybdenum occur, namely, $W_2Mo_3$ and WMo. The former would contain 79.3 per cent. tungsten and the latter 65.7 per cent. Samples closely approximating these analyses, as well as those containing more and less tungsten, have solid-solution structures. All of the evidence of the melting points indicates that tungsten and molybdenum form no intermetallic compounds.

### *Microscopic Examination of Tungsten and Molybdenum Alloys*

Fig. 4 is a photomicrograph of nearly pure tungsten at a magnification of 320 diameters. The impurity in this sample is non-metallic. A few globules of this can be seen in the micrograph. This sample contains by analysis about 99.8 tungsten.

Figs. 5 to 7 inclusive are photomicrographs at 320 diameters of the various alloys of tungsten and molybdenum, as indicated in the accompanying descriptions. It can be readily seen that all of these alloys are solid solutions.

Judging from the micrograph (not here reproduced) alone, of an alloy of 20 per cent. tungsten and 80 molybdenum, the black areas might be considered to be due to a second phase; in fact, these regions represent

[3] H. Mennicke: *Die Métallurgie des Wolframs*, p. 262. Berlin, 1911.

areas rich in tungsten. Under high power (1,200 diameters) the separation of the two phases could readily be detected. That they were not in

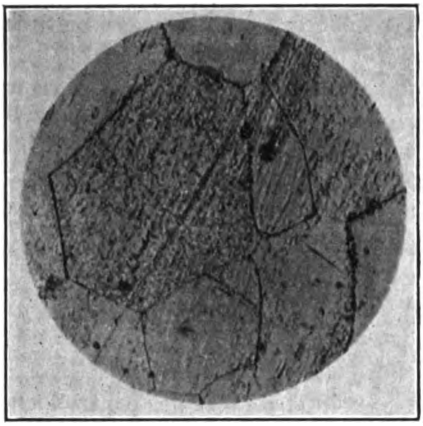


FIG. 4.—PURE TUNGSTEN. × 320.

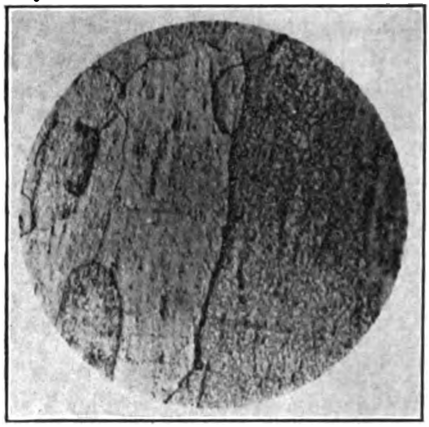


FIG. 5.—TUNGSTEN, 90 PER CENT.; MOLYBDENUM, 10 PER CENT. × 320.

equilibrium was also demonstrated by an additional heating which completely homogenized the alloy. In this connection, it would probably be advisable to mention the manner in which these alloys were made.[4]

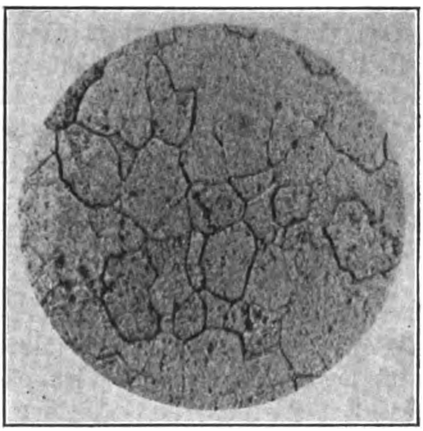


FIG. 6.—TUNGSTEN, 75 PER CENT.; MOLYBDENUM, 25 PER CENT. × 320.

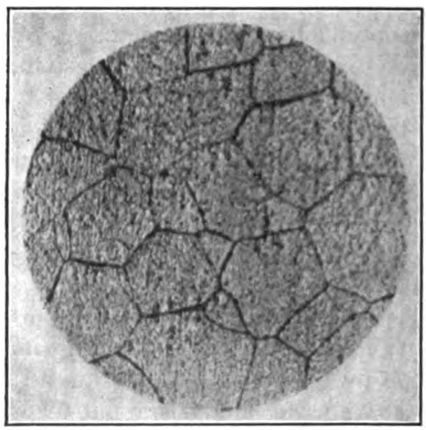


FIG. 7.—TUNGSTEN, 1 PER CENT.; MOLYBDENUM, 99 PER CENT. × 320.

The oxides of both tungsten and molybdenum were obtained in a very pure powdered state, and reduced to powdered metal by hydrogen.

[4] The method of making and swaging these alloys was practically the same as described by R. W. Moore, *Metallurgical and Chemical Engineering*, vol. 12, p. 186 (March, 1914).

The alloys were made up by thoroughly mixing the tungsten and molybdenum powders in the proper amounts. The mixed powders were then pressed into briquets which were sintered at about 1,300° C. in an atmosphere of hydrogen and were then heated by electric current to about 100° C. below the fusion points (which were previously determined with a separate series of briquets of these alloys) for a period of 12 min. The alloys, after receiving the above treatment, were completely crystalline. They may be ground by an emery wheel to a certain diameter, and the fusion points determined on these samples. They may also be heated and rolled or swaged to any desired diameter. The alloys used in these experiments were swaged to 0.030 in. diameter, the original cross-section of the briquets being about 1/16 sq. in.

The micrographs, with the exception of Fig. 6 (which is a section of

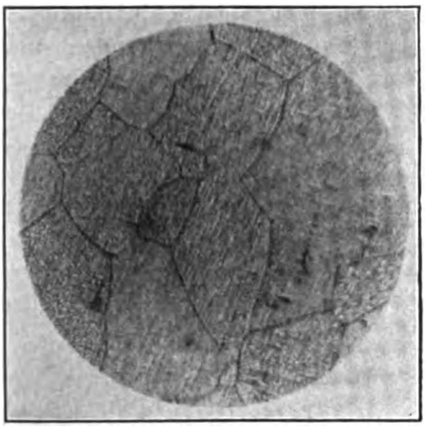


Fig. 8.—Pure Molybdenum. × 320.

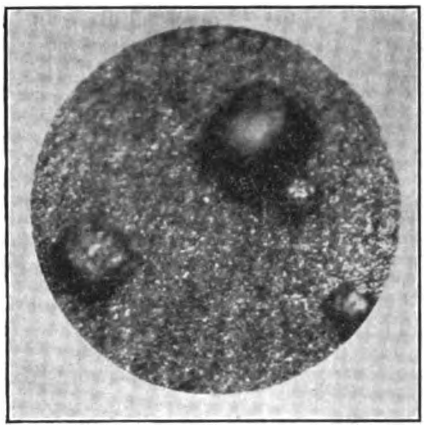


Fig. 9.—Etching Pits in Tungsten. × 775.

0.080 in. wire), represent longitudinal sections of these 0.030 in. diameter wires after fusion. The portions shown in these micrographs represent sections which have not been fused but which have been heated to temperatures very near their melting points. It will be evident that fusion of the wire will take place at a point approximately equidistant between the two electrodes and that adjacent portions of the wires would be heated to very near the melting point. In these sections, therefore, the solution of the tungsten and molybdenum must take place while both metals are in the solid state, or at least while the tungsten-rich portion is in the solid state. In case solution and diffusion have not been complete, two phases, one rich in tungsten, the other rich in molybdenum, will be present. In some of the briquets, the solution was not complete during the first heating, but was complete after the second heating. All evidences of

the existence of two phases were removed when sufficient time was given the samples at temperatures near their melting points.

These samples were also examined after fusion in the parts which had been molten. There was no evidence of a second phase in any of these samples; that is, they were all true solid solutions.

Fig. 8 represents a micrograph of pure molybdenum. By analysis, it contains about 99.9 per cent. molybdenum. Its purity is also suggested by the micrograph.

These alloys are all readily etched by boiling hydrogen peroxide. Etching pits in the various alloys were produced by a comparatively long attack—say, 3 or 4 min. in boiling hydrogen peroxide. All of the evidence from the etching pits points to the conclusion that the crystal units are cubes. The alloys of tungsten and molybdenum, when made up according to the above description, seem to form etching pits very much more readily than either of the pure metals. This might be attributed to the existence of small particles of either the tungsten-rich or the molybdenum-rich portions, which, by the difference in composition, would facilitate solution at that point, thus easily forming the etching pits.

It is of interest in this connection to note that the average grain as seen in the micrographs represents about 1,000 of the original particles of the powdered metals. This free grain growth is positive evidence of free solution of the metals in each other. Had the metals remained as separate phases, each would have impeded the coalescence of the grains of the other.

### *Probable Accuracy of Results*

The quantity of heat represented by the fusion wattage is independent of the quantity of heat actually necessary to raise a mass of metal, similar to that used in these samples, up to its melting point. It depends on the ability of the sample under investigation to dissipate heat in the apparatus. The thermal balance may be expressed by the equation:

Heat added electrically =
heat dissipated by radiation, convection and conduction.

The conditions obtaining within the apparatus are such that if the fusion of tungsten is represented by 1,800 watts, 1,750 watts could be dissipated for a long period of time without fusion of the tungsten.

The probable errors in the determinations are outlined below:

Errors due to:

1. Differences in emissivity of the various alloys under investigation.
2. Differences in their specific heats, heat capacities and heat conductivities.
3. The selective volatilization of molybdenum.

4. Variations in the flow of water through the water jackets, of the temperature of the water at the intake, and of the flow of hydrogen through the inclosed chamber.

5. Slight variations in the diameter of the wires.

6. Possible changes of melting points due to chemical or physical combination of alloys with hydrogen.

7. The personal equation in drawing the per cent. of tungsten fusion wattage-temperature curve.

8. The readings of the electrical instruments.

The differences in emissivity of the various alloys will make but a slight difference in the fusion wattage, for the reason that only about one-twentieth of the total heat dissipated is lost by radiation, and the electrodes combined.

The errors due to No. 2 will be very slight, for the same reason.

Langmuir and McKay[5] report that at 2,800° K., the heat loss from a tungsten filament in hydrogen at 750 mm. pressure, is 10½ times greater than that due to radiation; and at 3,500° K., the heat loss due to hydrogen is 11½ times that due to radiation. The conditions obtaining in the apparatus used by the writer were such that the losses due to the hydrogen are considerably more than is reported by these authors.

Dr. Irving Langmuir[6] suggests errors due to the selective volatilization of molybdenum.[7] The apparent high melting points of the alloys containing 5 per cent. tungsten, 95 per cent. molybdenum, and 5 per cent. molybdenum, 95 per cent. tungsten, may be partially due to this selective volatilization. The enrichment of the alloys in tungsten, due to this cause, could not have been more than 3 per cent. in any case, as was indicated by the diameter measurements of the wires before and after fusion.

The errors due to No. 4 are probably greater than those due to all other variations combined. Errors due to these causes, however, are not necessary if the proper precautions are taken. These precautions would consist of supplying the water for the water jackets from a constant-level and constant-temperature tank. The same conditions should be fulfilled with the hydrogen.

To find out the approximate magnitude of these errors, samples of the same wires were fused on different days, so that the valve-settings for jacket water and hydrogen had to be re-made and re-adjusted.

[5] I. Langmuir and G. M. J. Mackay: The Dissociation of Hydrogen into Atoms, Pt. I, *Journal of the American Chemical Society*, vol. 36, pp. 1708–1722 (1914).

[6] Private communication, March 29, 1916.

[7] I. Langmuir: The Vapor Pressure of Metallic Tungsten, *Physical Review*, Series II, vol. 2, No. 5, pp. 329–342 (November, 1913).

I. Langmuir and G. M. J. Mackay: The Vapor Pressure of the Metals Platinum and Molybdenum, *Physical Review*, Series II, vol. 4, No. 4, pp. 377–386 (October, 1914).

Several determinations showed a maximum variation in the melting point of molybdenum, as interpolated from the curve in Fig. 2, of about 40° C., and a maximum variation in the melting point of tungsten of about 10° C. The results on tungsten seem very favorable when it is considered that some methods for determining its melting point may vary 150° C.

Allowing for other errors, encountered in the determinations, it is reasonably accurate to assume that the maximum errors toward the molybdenum side of the diagram, are about 50° C., and toward the tungsten side of the diagram, 30°C. Allowing for errors of this magnitude, the points in Fig. 3 would fit nicely between the solidus and liquidus curves.

Errors due to slight variations in the diameters of the wires will be very small, owing to the fact that the ability of small wires to dissipate heat in a gas depends largely upon the existence of a relatively thick film of gas surrounding the wire. The thickness of this gas film is dependent largely upon the pressure of the gas, and not upon the diameter of the wire. The fact that these errors were slight was confirmed by an experiment, and was also predicted by Dr. Langmuir[8] from his extensive work along this line.

The suggestion was made by Dr. G. K. Burgess[9] that hydrogen might, by reason of either chemical or physical combination with the alloys, change their melting points. The writer has no knowledge concerning this. It might be well to add, however, that the handl'ng of this type of alloys at high temperatures is done almost entirely in hydrogen.

To ascertain the approximate errors due to the drawing of the curve in Fig. 2, it was plotted independently on two sheets of paper. The maximum difference between the two curves was about 10° C. at about 60 per cent. of the tungsten fusion wattage. To facilitate the interpolation of temperatures from fusion wattages, this curve was drawn on a large piece of cross-section paper, so that the reading errors would be reduced to a minimum.

The wattage, as determined by volts and amperes, was, of course, quite accurate, so errors from this source were negligible.

Concerning the próbable accuracy of results obtained by this method, Dr. Irving Langmuir writes: "I think the method you are using for estimating the temperature of filaments should give reasonably accurate results." He points out, among other things, the desirability of making chemical analyses *after* heating, to determine the actual composition of the alloy at the time of fusion. Dr. G. K. Burgess writes: "This method seems to me susceptible of very considerable accuracy."

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[8] I. Langmuir: The Dissociation of Hydrogen into Atoms, Pt. II., *Journal of the American Chemical Society*, vol. 37, pp. 417–458 (March, 1915).

[9] Private communication, April 6, 1916.

The author is convinced that the method is susceptible of greater accuracy than was obtained; that, with careful attention given to the construction and operation of the apparatus, melting points can be determined within 10° C. The method promises to offer solutions for several equilibrium diagrams of the higher melting-point metals.

*The Crystal System of Tungsten and Molybdenum*

Fig. 9 shows etching pits in tungsten, magnified 775 diameters. As can readily be seen from the micrograph, these etching pits represent sections of cubes cut by a plane parallel to a face.

Fig. 10 shows triangular etching pits which are formed by a plane cutting three faces of a cube.

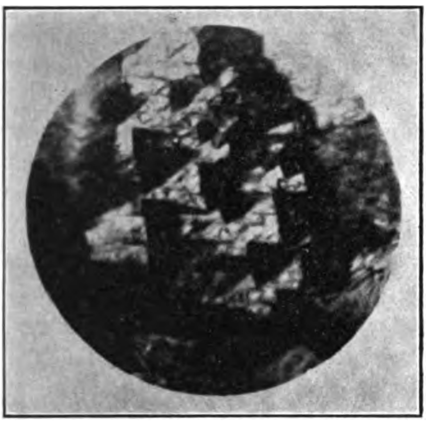

FIG. 10.—ETCHING PITS IN TUNGSTEN. × 775.

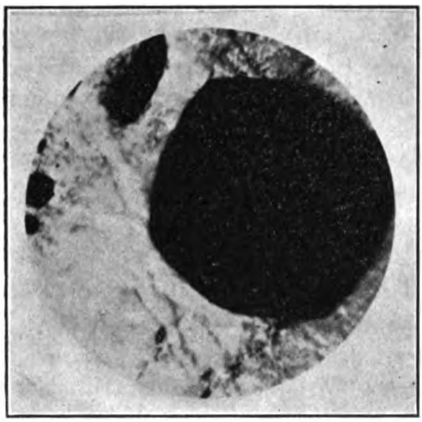

FIG. 11.—ETCHING PITS IN MOLYBDENUM. × 775.

Fig. 11 represents the intersection of the six faces of a cube by a plane. By changing the direction of the illumination, the writer was able to examine the bottom of this etching pit and could easily see that it was a section of a cube.

It was noticed in some of the micrographs that the etching pits instead of having straight line sides had curved sides. Goldschmidt[10] thinks that these curved lines are due to the convection currents set up in the etching solution by the unequal rate of attack at various points. The solution will be used up most rapidly at places where the greatest amount of surface is exposed. This will cause a flow of fresh solvent by diffusion. The corners of the etching pits will offer greater resistance to the flow of convection currents than the adjacent portions, and hence will not

[10] V. Goldschmidt: From the Borderland between Crystallography and Chemistry, *Bulletin University of Wisconsin*, Science Series, vol. 3, No. 2, pp. 21–38 (March, 1908).

receive so much fresh solution. The lines bounding the polygons are thus dissolved at unequal rates and become curved instead of straight.

A great many etching pits in all conceivable positions were examined microscopically. From these examinations, the author has very little hesitancy in saying that both tungsten and molybdenum crystallize in the isometric system and the form of their crystal units is the cube.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Zircon-Bearing Pegmatites in Virginia

BY THOMAS L. WATSON,* PH. D., CHARLOTTESVILLE, VA.

(Arizona Meeting, September, 1916)

## *Introduction*

THE occurrence of zircon in pegmatites of acidic composition is recorded by many observers both in this country and abroad, and they form one of the most important geologic modes of occurrence of the mineral.[1] As is well known, commercially the most important American locality for zircon is near Zirconia in Henderson County, North Carolina, where many tons of the mineral have been obtained from a kaolinized pegmatite dike 100 ft. wide and traced for 1½ miles along the direction of strike N. 50° E.

Zircon may be developed in pegmatites as inclusions in the principal rock-forming minerals, chiefly quartz and feldspar, and as a separate megascopic constituent in the form of grains and crystals not exceeding, as a rule, 2 in. in size, and usually smaller. In the Henderson County, North Carolina, pegmatite, zircon occurs in prismatic crystals with pyramidal terminations measuring up to 30 mm. in diameter, associated chiefly with the feldspar. In some of the apatite veins of Canada, which are closely allied to pegmatite dikes, zircon is reported in crystals upward of 6 in. in length and 2 in. and more in thickness.

Zircon in large masses occurs in Virginia in the well-known pegmatites near Amelia in Amelia County, and near Gouldin in Hanover County. Both localities are in the middle eastern portion of the Piedmont Plateau province, and are separated by a distance of about 40 miles in a northeast-southwest direction. The rocks of both areas, including the pegmatite bodies, are in an advanced stage of decay, and the hard and moderately fresh rocks are concealed beneath a cover of variable thickness of rock decay derived by the normal processes of weathering. Exposures, there-

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[1] For a summary of the properties, occurrence, and uses of zircon, see paper by T. L. Watson and F. L. Hess on Zirconiferous Sandstone near Ashland, Virginia, in *Bulletin of Philosophical Society of the University of Virginia*, Scientific Section, vol. 1, No. 11, pp. 267–292. (1912.)

fore, of even moderately fresh rock in the two areas are rare except along stream courses.

Pegmatite dikes occur in most of the Virginia Piedmont counties, and the detailed study of them now in progress by the State Geological Survey will doubtless add to those of Amelia and Hanover Counties that are zircon-bearing. The particularly interesting features of the zircon noted in the pegmatites of the two Virginia localities are (1) its occurrence in massive forms of unusual size, and (2) its association in the two localities with an entirely different group of rarer minerals, although the pegmatites of each area are of granitic composition. Zircon and apatite are the only rare minerals that have been found alike in the two areas, and, so far as the pegmatite bodies have been worked, zircon is more abundant in the pegmatites of the Hanover County area than in those of Amelia County.

### *Amelia County Area*

The pegmatite bodies occurring near Amelia Courthouse in Amelia County have long been known for the variety of rare minerals found in them, many of which were of unusual size. The dikes have been worked from time to time for a long period of years as a source of commercial mica and feldspar, and to a less extent of minerals for the gem trade.

The country rock is a thinly foliated, moderately dark-colored, fine-grained, biotite gneiss or schist, containing more or less muscovite. Where measured, the foliation strikes N. 25° to 30° E. and dips 40° to 50° N. W. Diabase dikes of Mesozoic age intrude the rocks in places. The pegmatite bodies are dike-like in form and nearly vertical, with the direction of trend doubtful. They cut across the foliation of the schists and the large ones will measure more than 50 ft. across. They are cut by joints, but there is no evidence of schistose structure developed from metamorphism.

The pegmatites are of granitic (acidic) composition, containing feldspar, including the potash varieties, orthoclase and green microcline, and the soda variety, albite, with quartz and muscovite, and a large number of rarer minerals. The principal rock-forming minerals are not uniformly distributed through all parts of the pegmatites, but their distribution is very irregular, first one and then another of these minerals predominating in different parts. The albite, occurring in splendid crystallizations as reticulated platy forms of bluish white to white color and frequently transparent, is of a high degree of purity as indicated in the two following analyses.

The texture of the pegmatites is granular consertal rather than graphic. Miarolitic cavities have been observed in some of the openings made in the pegmatites. One of these was of large size, the walls of

which were lined with crystals of smoky quartz and pure white crystals of albite, some as transparent as glass.[2]

*Analyses of Albite from Amelia County, Virginia*

| | I | II |
|---|---|---|
| $SiO_2$ | 68.44 | 68.22 |
| $Al_2O_3$ | 19.35 | 19.06 |
| $Fe_2O_3$ | ..... | 0.15 |
| CaO | ..... | 0.40 |
| $Na_2O$ | 11.67 | 11.47 |
| $K_2O$ | 0.43 | 0.20 |
| $H_2O$ | ..... | 0.69 |
| | 99.89 | 100.19 |
| Specific gravity | 2.605 | |

I. R. N. Musgrave, *Chemical News*, vol. 46, p. 204 (Nov. 3, 1882).
II. E. T. Allen, *Bulletin No.* 591 *of U. S. Geological Survey*, p. 300 (1915).

The rarer minerals include representatives of five distinct chemical groups: (1) *Haloids*, including fluorite; (2) *silicates*, including garnet (spessartite), black tourmaline, beryl, helvite, allanite, and zircon; (3) *niobates*, including columbite; (4) *tantalates*, including microlite; and (5) *phosphates*, including apatite and monazite. With the exception of fluorite, tourmaline, and zircon, each of the minerals has been analyzed with the results shown below. Some of these minerals have been found only occasionally in the Amelia pegmatites and are very rare. Many of them attained unusual size, such as crystals of beryl 3 to 4 ft. long and 18 in. thick, columbite in crystalline masses weighing 6 to 8 lb., allanite crystals more than 15 in. long, microlite in masses up to 8 lb. in weight, and monazite in masses larger than those of microlite.[3] Zircon has been noted in small crystals and in masses weighing several pounds. Stibnite and galena have been reported, but they are extremely rare and have not been seen by the writer.

Among the rarer minerals found in the Amelia County pegmatites, analyses of the garnet, beryl, helvite, allanite, columbite, microlite, apatite, and monazite have been made and are given below in the order named. The general character and mode of occurrence of these minerals in the Amelia County pegmatites have been fully described by Professor Fontaine.[4] No analysis has been made of the zircon. It is much less abundant than in the pegmatites of the Hanover County area and properly belongs to the more sparingly occurring rare minerals in the Amelia County area.

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[2] W. F. Fontaine: Notes on the Occurrence of Certain Minerals in Amelia County, Virginia, *American Journal of Science*, Series 3, vol. 25, p. 332 (1883).

[3] W. F. Fontaine, *ibid.*, pp. 330–339 (1883).

[4] *Ibid.*, pp. 330–339 (1883).

| | Spessartite | | | Beryl |
|---|---|---|---|---|
| | I | II | | III |
| $SiO_2$ | 36.34 | 35.35 | $SiO_2$ | 65.24 |
| $Al_2O_3$ | 12.63 | 20.41 | $Al_2O_3$ | 17.05 |
| $Fe_2O_3$ | ..... | 2.75 | $Fe_2O_3$ | 2.20 |
| FeO | 4.57 | 1.75 | BeO | 12.64 |
| MnO | 44.20 | 38.70 | CaO | 0.57 |
| MgO | 0.47 | None | $Na_2O$ | 0.68 |
| CaO | 1.49 | 0.94 | $H_2O$ | 2.70 |
| Ign. | Trace | 0.27 | | |
| | | | | 101.08 |
| | 99.70 | 100.17 | | |
| Specific gravity | 4.20 | ..... | Specific gravity | 2.702 |

I. C. M. Bradbury: *Chemical News*, vol. 50, p. 220 (Nov. 7, 1884). See also W. H. Seamon, *Chemical News*, vol. 46, p. 195 (Oct. 27, 1882).

II. F. W. Clarke: *Bulletin No. 60 of U. S. Geological Survey*, p. 129 (1890).

III. R. W. Barker: Analysis of a Beryl from Amelia C. H., Amelia Co., Virginia, *American Chemical Journal*, vol. 7, No. 3, pp. 175–176 (Oct., 1885).

| | Helvite | | | Allanite |
|---|---|---|---|---|
| | IV | V | | VI |
| $SiO_2$ | 25.48 | 31.42 | $SiO_2$ | 32.35 |
| BeO | 12.63 | 10.97 | $Al_2O_3$ | 16.42 |
| MnO | 39.07 | 40.56 | $Fe_2O_3$ | 4.49 |
| FeO | 2.26 | 2.99 | $Ce_2O_3$ | 11.14 |
| Mn | 8.66 | 8.59 | $Di_2O_3$ | 6.91 |
| S | 4.96 | 4.90 | $La_2O_3$ | 3.47 |
| $Al_2O_3$ | 2.95 | 0.36 | FeO | 10.48 |
| CaO | 0.71 | ..... | MnO | 1.12 |
| $Na_2O$ | 1.01 | ..... | CaO | 11.47 |
| $K_2O$ | 0.43 | ..... | $Na_2O$ } $K_2O$ } | 0.46 |
| | 98.16 | 99.79 | $H_2O$ | 2.31 |
| | | | | 100.62 |
| Specific gravity | ....... | 3.25 | Specific gravity | 3.323 |

IV. R. Haines: *Proceedings of Academy of Natural Sciences of Philadelphia*, 1882, p. 101.

V. B. E. Sloan: *Chemical News*, vol. 46, p. 195 (Oct. 27, 1882).

VI. F. P. Dunnington: *Proceedings of Academy of Natural Sciences of Philadelphia*, 1882, p. 103.

| | Columbite VII | | Microlite VIII |
|---|---|---|---|
| $Nb_2O_5$ | 31.40 | $Ta_2O_5$ | 68.43 |
| $Ta_2O_5$ | 53.41 | $Nb_2O_5$ | 7.74 |
| $SnO_2$ | Trace | $WO_3$ | 0.30 |
| FeO | 5.07 | $SnO_2$ | 1.05 |
| MnO | 8.05 | CaO | 11.80 |
| CaO | 1.27 | MgO | 1.01 |
| MgO | 0.20 | BeO | 0.34 |
| $Y_2O_3$(?) | 0.82 | $UO_3$ | 1.59 |
| | | $Y_2O_3$ | 0.23 |
| | 100.22 | $(Ce, Di)_2O_3$ | 0.17 |
| Specific gravity | 6.48 | $Fe_2O_3$ | 0.29 |
| | | $Al_2O_3$ | 0.13 |
| | | $Na_2O$ | 2.86 |
| | | $K_2O$ | 0.29 |
| | | F | 2.85 |
| | | $H_2O$ | 1.17 |
| | | | 100.25 |
| | | Specific gravity | 5.656 |

VII. F. P. Dunnington: *American Chemical Journal*, vol. 4, No. 2, p. 138 (May, 1882).

VIII. F. P. Dunnington: *American Chemical Journal*, vol. 3, No. 2, pp. 130–133 (May, 1881).

| | Apatite IX | | Monazite X | Monazite XI |
|---|---|---|---|---|
| $P_2O_5$ | 41.06 | $P_2O_5$ | 24.04 | 26.12 |
| CaO | 53.94 | $Ce_2O_3$ | 16.30 | 29.89 |
| F | 3.30 | $La_2O_3$ | 10.30 | 26.66 |
| Cl | Trace | $Di_2O_3$ | 24.40 | |
| $Al_2O_3$ | 0.19 | $(Y, Er)_2O_3$ | 1.10 | ..... |
| $Fe_2O_3$ | 0.81 | $SiO_2$ | 2.70 | 2.85 |
| Insol. | 0.63 | $ThO_2$ | 18.60 | 14.23 |
| Ign. | 0.81 | $Fe_2O_3$ | 0.90 | ..... |
| | | $Al_2O_3$ | 0.04 | ..... |
| | 100.74 | Ign. | ..... | 0.67 |
| Specific gravity | 3.161 | | | |
| | | | 98.38 | 100.42 |
| | | Specific gravity | ..... | 5.30 |

IX. G. H. Rowan: *Chemical News*, vol. 50, p. 208 (Oct. 31, 1884).

X. F. P. Dunnington: *American Chemical Journal*, vol. 4, No. 2, p. 140 (May, 1882).

XI. S. L. Penfield: *American Journal Science*, vol. 24, pp. 250–254 (1882).

*Hanover County Area*

The zircon-bearing pegmatites of Hanover County form a part of the recently discovered but fairly well-known rutile area of Goochland and Hanover Counties,[5] which lies about 25 miles northwest of Richmond (see map, Fig. 1). The principal rock of the region is a gneiss of variable composition, chiefly micaceous (biotite and muscovite) and at times

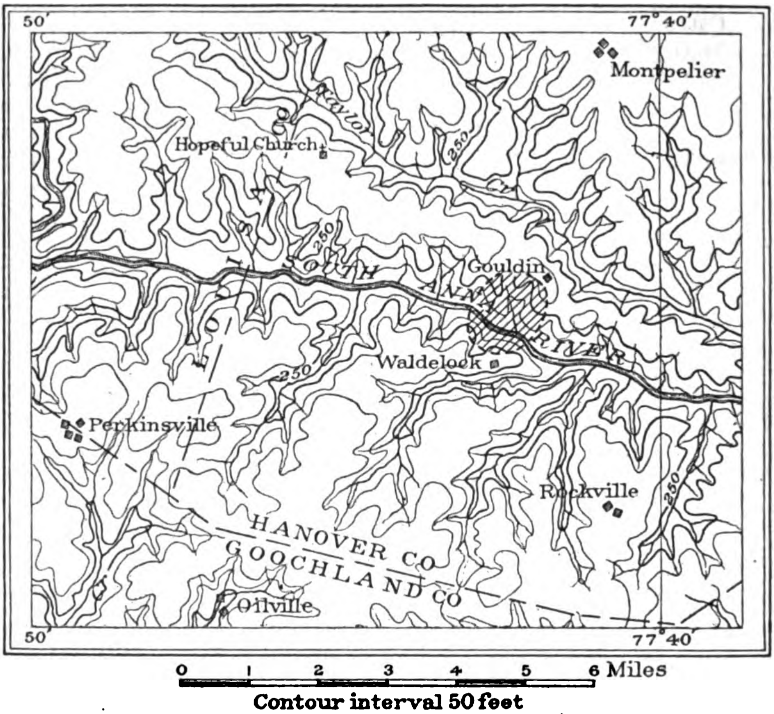


FIG. 1.—TOPOGRAPHIC MAP OF A PART OF HANOVER COUNTY, VIRGINIA, SHOWING LOCATION OF GOULDIN ZIRCON-BEARING PEGMATITE AREA. (BASED ON GOOCHLAND SHEET OF U. S. GEOLOGICAL SURVEY.)

hornblendic, cut by numerous pegmatites, some of which are rutile-bearing, and a variety of basic igneous rocks. Microscopic study of thin sections of the gneiss shows it to conform in composition to an original acidic igneous rock of the granite type. The banded structure is secondary, developed by regional metamorphism.

The pegmatite bodies range up to 4 ft. and more in width, are of light color and granitic composition, and lie mostly in the foliation planes of the gneiss. Like the inclosing gneiss, they have been mashed and

[5] T. L. Watson and S. Taber: Geology of the Titanium and Apatite Deposits of Virginia, *Virginia Geological Survey Bulletin*, III A, pp. 248–261 (1913).

squeezed from metamorphism and are schistose in structure. Much of the feldspar has been granulated and the quartz mashed into lens-like masses. The pegmatites are dominantly feldspathic, containing both potash and soda-lime varieties, with some quartz, and less muscovite and biotite. Neither amphiboles nor pyroxenes occur in the pegmatites of the Hanover or Amelia County areas. Rutile in small grains and masses up to 15 and 20 lb. and more in weight[6] is an important mineral in some of the Hanover County pegmatites, and is found over parts of the area in such quantity as to be of commercial value.

The rutile is an original constituent of the pegmatites and much of it shows granulation and mashing like the fieldspar and quartz. Besides the usual rock-forming minerals of the pegmatites, the rarer minerals are exceptionally few in number. Rutile, the most abundant and important one, is associated with ilmenite which may occur as separate grains and masses or as an intimate mixture or intergrowth with rutile. Occasional apatite has been found, and in places indications of the former presence of pyrite.

In addition to these, zircon has recently been found associated with rutile in the pegmatites near Gouldin in the Hanover portion of the rutile area. (See map, Fig. 1.) The mineral has been found in irregular fragments and masses up to about 2 lb. in weight. One of the larger masses examined by the writer appears to have been broken from a large crystal of the mineral. Like the other constituents of the pegmatites, every specimen of the zircon studied shows mashing and squeezing from metamorphism. The color is irregular even in the same mass, ranging from reddish-brown through grayish to colorless. Although a chemical analysis of the zircon has not been made, laboratory tests carried out on a number of pieces of the mineral show it to be quite pure. The many pieces of the mineral found on the surface, due to the extensive weathering of the pegmatite bodies, encourage the belief that the mineral is by no means a rare constituent of the dikes in this area, and may be found in quantity to be of commercial value.

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[6] F. L. Hess mentions one mass of rutile that was reported to have weighed 200 to 300 lb. *Mining World*, vol. 33, pp. 305–307 (Aug. 20, 1910).

# Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. Howland, printed in *Bulletin* No. 111, March, 1916, pp. 627 to 650.

A. H. Lee, Buffalo, N. Y. (communication to the Secretary*).—The statements and results recorded in Mr. Howland's paper can be accounted for by the use of coke with a low sulphur content (0.60 and under). This allows the furnaces to be run on a small volume of acid slag, which it is presumed was done in Mr. Howland's practice. H. A. Brassert has admitted that this was the largest factor in low fuel consumption in his practice. Where the coke contains from 1.00 to 1.25 per cent. of sulphur the situation is quite different and I doubt whether the results mentioned by Mr. Howland could be obtained under such circumstances.

N. M. Langdon, Mancelona, Mich. (communication to the Secretary†).—Gruner's ideal working (which is but a statement of the law first enunciated by Bell) as given in Gordon's *Translation of Studies of Blast Furnace Phenomena*, "supposes that the reduction of the oxide of iron is effected by the CO only, without intervention of solid carbon."

This ideal, if reached, would mean theoretically perfect or 100 per cent. efficiency of reduction. The workings of the large number of furnaces as given in Bell's and Gruner's great works do not show that this standard of perfection ever was reached. This problem has been worked out by the writer for a number of furnaces, and his results, as well as those given by others, all show a failure to reach perfection. Some furnaces, however, approach nearer to perfection than others. The "ideal" as given by Gruner was not stated as a solved problem in blast-furnace practice, but as an ideal to the attainment of which furnace managers should strive. A study of the factors involved from every angle is interesting and instructive and the paper on this subject contributed by H. P. Howland is no exception to the rule. This article is fairly open to criticism, not because of the unique method of presenting the subject, but for the reason that, as the writer conceives, erroneous assumptions are taken which lead naturally to incorrect conclusions.

Referring to the example of two furnaces compared to show the efficiency of the fuel and, for better illustrating the comparison, rearranging the data as follows:

* Received May 9, 1916.
† Received May 1, 1916.

*Exhibit* 1

| | Coke | Total Carbon | Carbon Used in Iron and Flue Dust | Carbon Burned at Tuyères | Carbon Used for Direct Reduction |
|---|---|---|---|---|---|
| 1 | 2,100 | 1,848 | 100 | 1,486 | 262 |
| 1*a* | 1,800 | 1,586 | 100 | 1,486 | 0 |
| 2 | 2,500 | 2,200 | 100 | 1,486 | 614 |

1. $\left(\frac{1,486}{1,848 - 100}\right) \times 100 = 85$ per cent. efficiency.

1*a*. $\left(\frac{1,486}{1,586 - 100}\right) \times 100 = 100$ per cent. efficiency.

2. $\left(\frac{1,486}{2,200 - 100}\right) \times 100 = 70.8$ per cent. efficiency.

In the above figures no account has been taken of the carbon required for reduction of the metalloids, which is always direct. The general context of the article, as well as the tables and the one case when the amount of carbon required for the reduction of the metalloids is stated, shows it to be small. Therefore, without vitiating the conclusions to be drawn, it may be neglected.

Case (1) shows 85 per cent. efficiency, (1*a*) 100 per cent. and (2) 70.8 per cent. This is correct for the conditions assumed, but a further examination shows 100 per cent. efficiency of carbon burned at the tuyères and the whole of the inefficiency located at the top or reducing zone. Had the reduction of the ore been 100 per cent. inefficient, that is, had the reduction been made with direct carbon only, it would have required 677 lb. of carbon. Letting *a* equal the amount of solid carbon actually used in the reducing zone, the equation $\frac{677 - a}{677} \times 100$ gives the percentage of efficiency in this zone. Applying to the above we have as the efficiency of reduction:

*Exhibit* 2

Case 1. $\frac{677 - 262}{677} = 61.3$ per cent.

Case 1*a*. $\frac{677 - 0}{677} = 100$ per cent.

Case 1. $\frac{677 - 614}{677} = 9.3$ per cent.

These percentages are instructive and show that, in order to secure maximum efficiency in the combustion of carbon in the blast furnace, attention should be directed to the reducing zone.

Quoting, "Table 1 shows clearly that, at least in these furnaces, there is no law governing the relation between coke consumption and the per cent. of the carbon which is burned at the tuyères." For comparison

and discussion, the data of the two examples 5 and 18, are stated and some additional data inserted.

*Exhibit* 3

| | Coke | Carbon of Coke Gasified in Furnace | Carbon Gasified | | Per Cent. Burned at Tuyères | Efficiency of Reduction | Stone Used |
|---|---|---|---|---|---|---|---|
| | | | At Tuyères | Reducing Zone | | | |
| 5 | 2,198 | 1,813 | 1,494 | 319 | 82.4 | 52.9 | 1,003 |
| 18 | 1,699 | 1,376 | 1,155 | 221 | 83.9 | 67.4 | 804 |
| 24 | 1,623 | 1,360 | 1,090 | 270 | 80.2 | 60.1 | 775 |
| 19 | 1,673 | 1,384 | 1,182 | 202 | 85.0 | 70.2 | 780 |

While the carbon burned at the tuyères has a direct reference to coke consumption, yet a comparison of percentages is misleading, for the reason that this carbon is only a part of the total carbon burned and this other part, burned in the reducing zone, is variable according to the efficiency of reduction. The fact that a furnace is working with low coke consumption does not by any means indicate that it is even closely approaching Gruner's ideal. This is shown by 24 in Exhibit 3. It will be noted that the coke, total carbon gasified, and carbon gasified at tuyères are lower than 18, while the carbon gasified in the reducing zone is higher, showing a lower efficiency of reduction. The carbon required at the tuyères of any furnace is governed entirely by the quantity, composition and character of the materials entering the furnace and the pig iron produced. The carbon required in the reducing zone depends upon the character of the metallic oxides, the composition of the pig produced and largely upon the degree of efficiency of reduction. From what has been said, it is obvious that if this dictum is accepted, under similar conditions that furnace which shows the highest percentage of efficiency of reduction and thus approaches Gruner's ideal will show the lowest coke consumption, and naturally that is the condition to be desired.

The high coke of No. 5 is due to a lean ore mixture as disclosed by the amount of stone (1,003 lb.) used per ton of iron, necessitating a larger amount of carbon burned at the tuyères than in the case of either No. 18 or No. 24, also to the low efficiency of reduction (52.9 per cent.) in the reducing zone. The lower coke requirement of No. 18, as compared with No. 5, is due to a better ore mixture and higher efficiency of reduction, 67.4 per cent., as compared with 52.9 per cent. The still lower coke requirement of No. 24 is due to a still better ore mixture as disclosed by the smaller amount of stone used, permitting a decrease in carbon required at the tuyères to only 1,090 lb. as compared with 1,155 lb. in No. 18; and this decrease is sufficient to more than overbalance the increase of carbon due to lower efficiency, 60.1 per cent., as compared with 67.4 per cent., in the reducing zone.

For discussion of the work of furnace No. 19, the following tabulation is made:

*Exhibit* 4

| | Coke | Fe | Total Carbon | Carbon in Pig | Carbon Required | |
|---|---|---|---|---|---|---|
| | | | | | At Tuyères | Reducing Zone |
| 19 A | 1,673 | 2,031 | 1,482 | 98 | 1,182 | 202 |
| 19 B | 1,673 | 2,031 | 1,482 | 98 | 1,182 | 36 |
| 20 | 1,658 | .... | 1,464 | 98 | 1,182 | 184 |

Whether the equation is correct or not as stated in Exhibit 5, column A, the carbon actually used for reduction and the resulting gas is the same as shown in Column B and this latter equation is simpler.

*Exhibit* 5

| | | A | B |
|---|---|---|---|
| Reduction by CO | 1 | $Fe_2+O_3+6CO=2Fe+3CO+3CO_2$ | $Fe_2O_3+3CO=2Fe+3CO_2$ |
| Reduction by C | 2 | $Fe_2+O_3+2C=2Fe+CO+CO_2$ | $Fe_2O_3+3C=2Fe+3CO$ |

The equation 1 A shows 6 CO taken for reduction, only one-half of which was actually used. This unused half, like the bridesmaid at a wedding ceremony, may have performed a very interesting and important part, but, itself, forms no part of the union. This 3CO remains the same after the reaction as before, therefore only 3CO can be considered as having been used, the equation 1 B correctly representing the actual ultimate result.

The reduction of oxide of iron in the blast furnace by solid carbon takes place at a point where the temperature is sufficient in the presence of the surplus of carbon from the fuel to convert any $CO_2$ which may have formed to CO. The ultimate gas from the reduction by solid carbon is CO only and the ultimate reaction is represented by the equation 2 B.

Reconstructing the summary for "assumption A" which is to illustrate reduction by CO and referring to Exhibits 4 and 5, we have according to equation 1 B the results shown in Exhibit 6.

We see that 166 lb. of solid carbon reduced 517 lb. of Fe, and that 487 lb. of carbon from the total of 1,182 lb. burned to CO at the tuyères reduced 1,514 lb. of Fe. The total CO and $CO_2$ gases are the same as shown in the original "assumption A." The ratio by volume of $\frac{CO_2}{CO}$ is 0.646, which is very much lower than the ratio 1CO to $1CO_2$ popularly though mistakenly supposed to be the point of saturation by $CO_2$ at which reduction by CO ceases.

*Exhibit* 6

| | | Fe, pounds | C, pounds | CO, pounds | $CO_2$, pounds |
|---|---|---|---|---|---|
| 19 A | In pig | ..... | 98 | ..... | ..... |
| | Direct reduction of metalloids | ..... | 36 | 89 | ..... |
| | Direct reduction of Fe | 517 | 166 | 387 | ..... |
| | Indirect reduction of Fe | 1,514 | 487 | ..... | 1,786 |
| | Unused | ..... | 695 | 1,622 | ..... |
| | Stone | ..... | ..... | ..... | 343 |
| | | 2,031 | 1,482 | 2,098 | 2,129 |

$$\frac{CO_2}{CO} = \frac{2,129}{2,098} = 1.015 \text{ by weight or } 0.646 \text{ by volume.}$$

Below is "assumption B" reconstructed according to equation 2 B.

*Exhibit* 7

| | | Fe, Pounds | C, Pounds | CO, Pounds | $CO_2$, Pounds |
|---|---|---|---|---|---|
| 19 B | In pig | ..... | 98 | | |
| | Direct reduction of metalloids | ..... | 36 | 89 | |
| | Direct reduction of Fe | 0 | 0 | 0 | |
| | Indirect reduction of Fe | 2,031 | 653 | ..... | 2,394 |
| | Unused | ..... | 529 | 1,234 | |
| | Stone | ..... | ..... | ..... | 343 |
| | | 2,031 | 1,316 | 1,323 | 2,737 |
| | Carbon saved | ..... | 166 | | |
| | | | 1,482 | | |

$$\frac{CO_2}{CO} = \frac{2,737}{1,323} = 2.069 \text{ by weight or } 1.32 \text{ by volume.}$$

The original "assumption B" is supposed to show the result when "all the CO made at tuyères reduces ore and that the excess carbon is used up by $CO_2$," as applied to the same data as "assumption A." Assumption B is not tenable because, under the conditions, we must assume the same amount of carbon burned at the tuyères as in assumption A. Now, to reduce the Fe wholly by carbon from CO requires only 653 lb., leaving the balance of 529 lb. of carbon to pass off as CO, and as all of the Fe has been reduced there is an excess of 166 lb. of carbon in the reducing zone which cannot be burned to either CO or $CO_2$ because there is no oxygen there to unite with it.

The correct statement of the case, assuming the reduction to have been effected with CO only, is shown in Exhibit 7, which shows a saving of 166 lb. of carbon. It may be observed, however, that this saving is

probably impossible because of the dilution of the gas by $CO_2$ which, as shown in the Exhibit, is 1.32. Bell practically states that reduction stops when the proportions are $\frac{CO_2}{CO} = 1.00$.

Assumption G is not well taken as illustrating Gruner's ideal because, in addition to assuming that reduction was by CO only, another unnecessary assumption is injected into the consideration, viz., that 166 lb. more carbon is burned at the tuyères than actually was burned.

Records of heat equations for actual practice of a number of furnaces, collected by the writer, show that for the character of iron given for No. 19, the lowest amount of carbon that can be expected in direct reduction is 156 lb. Assuming this amount for furnace No. 19, we have:

*Exhibit* 8

| | | Fe, Pounds | C, Pounds | CO, Pounds | $CO_2$, Pounds |
|---|---|---|---|---|---|
| 19 C | In pig | ..... | 98 | | |
| | Direct in metalloids | ..... | 36[1] | 89 | |
| | Direct in Fe | 373 | 120 | 280 | |
| | Indirect in Fe | 1,658 | 533 | ..... | 1,954 |
| | Unused | ..... | 649 | 1,514 | |
| | Stone | ..... | ..... | ..... | 343 |
| | | 2,031 | 1,436 | 1,883 | 2,297 |
| | Saved | ..... | 46 | | |
| | | | 1,482 | | |

$$\frac{CO^2}{CO} = \frac{2,297}{1,883} = 1.22 \text{ by weight or } 0.777 \text{ by volume.}$$

[1] Equal 156 lb.

This shows a ratio $\frac{CO_2}{CO}$ of 0.777 by volume, and is probably as near to Gruner's ideal as could be obtained in actual practice.

The statement that "all the furnaces in Table 1 are doing direct reducing" is correct, while the statement that "all those that are burning less than 1,350 lb. of carbon at the tuyères are not making enough CO to reduce all the $Fe_2O_3$" is incorrect. It is also incorrect to say that reduction is three times as efficient when done by the direct method. The oxygen from the iron requires precisely the same amount of carbon for its reduction whether taken from the solid carbon of the fuel or from the carbonic oxide of the gas. When the carbon can be taken from the carbonic oxide produced by the fuel necessarily burned at the tuyéres, it obviously saves an equal amount of carbon from the fuel that, otherwise, would be needed and, therefore, is more efficient than solid carbon. The

extent to which this can be done is the measure of the approach to the ideal.

All furnaces do not reach Gruner's ideal. In fact, it is very doubtful if any do or can. Some, however, come nearer to it than others. They may show very low coke consumption and yet be far from reaching the ideal. The reason for this is not because of a violation of or considerable variation from Gruner's ideal, but because of some other favorable condition. This is shown in Exhibit 9.

*Exhibit* 9

| | Total Carbon Gasified, Pounds | Carbon Gasified at Tuyères, Pounds | Carbon Gasified in Reducing Zone Pounds |
|---|---|---|---|
| Average of first 10 furnaces | 1,800 | 1,522 | 278 |
| Average of last 16 furnaces | 1,384 | 1,146 | 238 |
| Difference................ | 416 | 376 | 40 |

It will be noted that the last 16 furnaces show a decrease of 416 lb. of carbon required, and a decrease of 40 lb. in the reducing zone due to a nearer approach to Gruner's ideal. The great decrease of 375 lb. is in carbon burned at the tuyères evidently due to a better ore mixture, requiring less flux, as shown by the stone used. Less slag is produced, therefore less carbon is required to be burned at this point.

From what has preceded, it will be evident that the factors governing coke consumption will be found in the quantity and quality of the materials entering the furnace, coke, ore, limestone and blast; also in the method of reduction—that method tending to the highest economy, which approaches nearest to Gruner's ideal.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Motor Truck Operation at Mammoth Collins Mine, Shultz, Ariz.

BY WILBERT G. McBRIDE,* B. S., LONG BEACH, CAL.

(Arizona Meeting, September, 1916)

Two Alco 3½-ton motor trucks were used by Young Bros. while operating at the Mammoth Collins mine at Shultz, Ariz. One was equipped with an oil tank holding 1,075 gal. and was used for the transportation of "tops." The other was fitted with a stake body and used to carry machinery, wood, rails, pipe and all classes of miscellaneous supplies. The bodies were made of oak with maple flooring and were attached to the frame of the chassis by U-bolts, to avoid drilling the main members of the frame.

Most of the hauling was done from Tucson, a distance of 47½ miles. During the first 3 months, part of the road was in very bad condition and the tire cost was excessive. After this part was repaired the road was in fair condition, but never good. There were no excessive grades or bad sand, but wagon ruts, too narrow for the truck wheels and of a different gage, caused heavy tire loss; while chuck holes, sharp curves and stones, both imbedded and loose, were objectionable features. During wet weather the trucks could not get sufficient traction to climb some of the hills and were likely to stick in the mud in certain places, so that no attempt was made to run them unless they were on the road when the rain started. This lost time amounted to about 5 per cent. of the total, but, whenever possible, it was utilized in making minor repairs.

The price of gasoline was from 17 to 21 c. per gal. Rubber tires were used throughout. Drivers were paid $4.50 to $5 per shift, and a return trip to Tucson was counted as two shifts even when made in one day. Drivers were provided with a room in Tucson and were paid for all time lost due to causes beyond their control. Trucks were loaded one way only.

Speedometers were placed on both trucks but the excessive vibration soon caused them to fail. For this reason, and because no account was taken of the distance covered in picking up a miscellaneous load or in other minor ways, the mileage given is under the actual distance traveled. Some of the weights had to be estimated, but care was taken to have the number of ton-miles low rather than high, to avoid underestimating the costs. The cost of hauling from Tucson to the mines was $12 per ton

* Consulting Engineer.

with the trucks, while the best possible team price was $15. Teams made one return trip a week, while the truck regularly made one in two days and could always, and many times did, do it in one day. The loss of time due to wet weather would be about half as much with teams as with trucks.

The table of detailed costs given below covered the period from Aug. 21, 1913, to Aug. 15, 1914, the only time in which the trucks were continuously employed. From Aug. 15, 1914, to March 30, 1916, the trucks were used intermittently, but the figures for this period have been excluded as not being representative. If included, they would somewhat lower the cost per ton-mile. Just prior to the close of the period covered by the cost figures, the trucks were overhauled and put in good condition; new rear wheels were put on and new tires secured. The cost of all this was charged to operation. Allowance for extra tires on hand would reduce the cost per ton-mile approximately ¾ c., leaving a net cost of about 25c. With loads on the return trip this cost per ton-mile would be lowered at least 40 per cent.

*Motor Truck Operating Data*

| | |
|---|---|
| Total distance traveled by trucks | 23,000 miles |
| Total work done by trucks | 42,700 ton-miles |
| Average distance covered per gallon of gasoline | 4.5 miles |
| Average distance covered per gallon of lubricating oil | 128 miles |
| Average speed, loaded | 7 miles per hour |
| Average speed, light | 7.8 miles per hour |

*Details of Costs*

| | Total Cost | Per Cent. of Total | Per Truck-mile | Per Ton-mile |
|---|---|---|---|---|
| Wages of drivers | $2,623.32 | 23.91 | $0.1141 | $0.0614 |
| Wages of helpers | 286.50 | 2.62 | 0.0125 | 0.0067 |
| Repairs, labor | 581.74 | 5.30 | 0.0253 | 0.0136 |
| Repairs, lost time | 156.15 | 1.42 | 0.0068 | 0.0037 |
| Oils, grease and waste | 379.17 | 3.46 | 0.0165 | 0.0089 |
| Gasoline | 1,610.49 | 14.68 | 0.0700 | 0.0377 |
| Tires | 2,445.75 | 22.30 | 0.1063 | 0.0573 |
| New parts | 515.08 | 4.69 | 0.0224 | 0.0121 |
| Miscellaneous supplies | 348.82 | 3.18 | 0.0152 | 0.0082 |
| Incidental expense | 226.21 | 2.06 | 0.0098 | 0.0053 |
| Depreciation | 1,796.80 | 16.38 | 0.0781 | 0.0421 |
| Total | 10,970.03 | 100.00 | 0.4770 | 0.2570 |

The advantages of the motor truck over the team and wagon are many—increased speed, ability to work 24 hr. per day when necessary, and lower cost on long hauls—but its adoption by the mining industry

has been slow. Where trucks are used around mines they are usually driven by cheap, inexperienced men, the upkeep and repairs being turned over to the regular mine mechanics. It would be equally good practice to employ a timber framer to make a dining-room table. Just as the niceties of cabinet making are unknown to the timber framer, the exact adjustments and fine workmanship of the high-speed engine and transmission gears of a motor truck are beyond the ken of the mine mechanic, one of the least skilled of his class. If there are enough motor vehicles at the mine, the master mechanic probably turns the work over to one or two men who, in time, become indifferent auto-mechanics, but in the meantime the cost of maintenance soars and often the trucks are condemned. The aim of the makers of all motor vehicles is to secure the maximum of strength and power with a minimum of weight and size. To do this, high-speed engines, the best of materials and the finest of workmanship are employed and parts are reduced to the least possible weight consistent with strength and durability. This is just the reverse of the ordinary American mechanical practice in which reliability is secured by slow speed and large size, the amount of material used and the space occupied being minor considerations. It is, therefore, unreasonable to expect the mechanic trained in one school to understand immediately and adapt himself to the ways of the other. It must also be remembered that no other machine is given the hard use and necessary abuse that a motor truck receives. The road vibration, alone, will loosen nuts and rivets which, if not attended to in time, will cause serious trouble. Where only one or two trucks are used, the drivers should be competent mechanics and should be held responsible for the maintenance of their machines. Where several are used, they should be under the direct supervision of a thorough truck mechanic who is held responsible for operation and given entire control of the drivers and repair work. His constant care will detect and remedy many incipient defects and prevent expensive and annoying breakdowns. With the exception of the time required for periodic overhauling, he should be able to keep the trucks in almost continuous service. This will make possible the employment of cheaper drivers without undue damage being done to the machines.

Motor trucks should not be installed without careful consideration of the roads to be traveled. The difference between the cost of motor truck and team hauling is largely controlled by the quality of the road, and on really bad roads the motor truck is decidedly the more expensive. Many roads are fatal to truck haulage, and considerable experience is required to decide this question without an actual test of some duration. Excessive grades are to be avoided, especially long ones. The ordinary truck will pull over a short 20 per cent. grade with ease, but will give great trouble on a long one of half that rise unless special cooling arrangements are made. Grades greatly increase the tire and gasoline consump-

tion and decrease the life of the machine. Rocky roads, particularly when the rocks are sharp or loose, are very hard on tires. Deep sand is difficult to cross, and for this class of road the caterpillar tractor and the four-wheel-drive truck have distinct advantages. Trucks which drive on the rear wheels only cannot operate in heavy sand. Narrow or rutted roads are objectionable for the larger-sized trucks because they throw all the weight on one of the rear dual tires from time to time, and this overloading is injurious to the rubber. Fairly deep streams can be crossed, but mud is an absolute barrier except to the caterpillar type of tractor. Few dirt roads will stand up under a 7-ton truck, but those of 4 tons, or under, do less damage than the ordinary freight wagon.

Unfortunately, trucks are not designed to suit mining conditions. At Shultz we found it necessary to cut down the gear ratio, increase the size of the wheels and tires and add bumper or auxiliary springs. Had the grades been steeper it would have been necessary to increase the cooling capacity.

For long hauls the motor tractor will probably replace the motor truck. It will operate at a lower cost because the load will be carried on iron tires, and, as the table of detailed costs shows, the rubber tires account for 22.3 per cent. of the total. Tractors travel more slowly than motor trucks, but the tonnage hauled in a trip is much greater. They are also easier on roads, as the load is distributed over several trailers. By using extra trailers, loading and unloading can be done while the tractor is on the road.

The make of a truck is not as important as the care it receives. Almost any standard make will do good work if given careful attention, but none will be satisfactory if not well cared for. Economy should not be sought in the lubricants used; the best oil is none too good for a motor truck. Overloading should be scrupulously avoided. A truck may be made to carry many times its rated load without breaking down but the damage is none the less real because not immediately apparent. High speed, particularly if the road is rough, should be avoided, since it subjects the machine to excessive strains and vibration. Most trucks are now equipped with speed governors, but these are easily tampered with and must be carefully watched. When they are not used, the drivers should be carefully instructed as to the speed limits and compelled to respect them.

Distillate and "tops" are now successfully used on trucks, by the application of a special carburetor. Their use should effect a material saving in the gasoline cost which now amounts to almost 15 per cent. of the total. "Tops" usually sell for 30 to 35 per cent. and distillate for 50 to 60 per cent. of the price of gasoline. With a properly designed carburetor, the available power in the lower-grade fuel will be about the same as in the gasoline, but the carbon deposition will probably be somewhat greater.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Features of the New Copper Smelting Plants in Arizona

BY A. G. MCGREGOR,* B. S., WARREN, ARIZ.

(Arizona Meeting, September, 1916)

DURING the past 5 years, five new copper-smelting plants have been built and put into operation in the State of Arizona. The monthly copper output from these plants averages from 5,000,000 to 18,000,000 lb. Previously, there never was as much activity in copper-smelting plant construction in the same length of time in all the rest of the world.

Naturally, in this amount of work, some new problems were met and new features in plant design and equipment developed. Some of these features are described in this paper.

## CONCENTRATE HAULING, UNLOADING AND SAMPLING SYSTEM

### (a) *Concentrate Car*

The International Smelting Co. has provided at its plant at Miami specially designed cars for hauling the concentrates from the mills of the Inspiration Copper Co. and the Miami Copper Co. The car has a bottom that is readily made tight to hold flotation concentrates and is quickly and cheaply unloaded. Figs. 1, 2 and 3 show views and details of the car.

The main feature of this car is the slot in the bottom throughout its length, which is closed by means of short planks that overlap like ship-lap as shown by Fig. 1. At one end of the car an iron gate operated by a screw and hand wheel is provided, which clamps or presses the planks together, making the bottom tight. A removable tapered plug rests vertically over this gate when the loading of the concentrates at the mill commences.

A concentrate unloading pocket (Fig. 4) 180 ft. long, is provided at the smelting plant, which has a conveyor belt running throughout its length underneath the track. To unload a car, the vertical plug at the end of the car is lifted by means of a chain hoist which is supported by a trolley overhead. The gate at the end of the car is opened approximately 18 in. The removal of the plug leaves a hole down through the concentrates. The concentrates are poked through the hole from the top of the car until it is convenient for a man to stand on the bottom and draw one of the slats in the bottom toward the gate end of the car. The concentrates are then raked down with hoes on their natural angle of repose,

* Mechanical and Metallurgical Engineer.

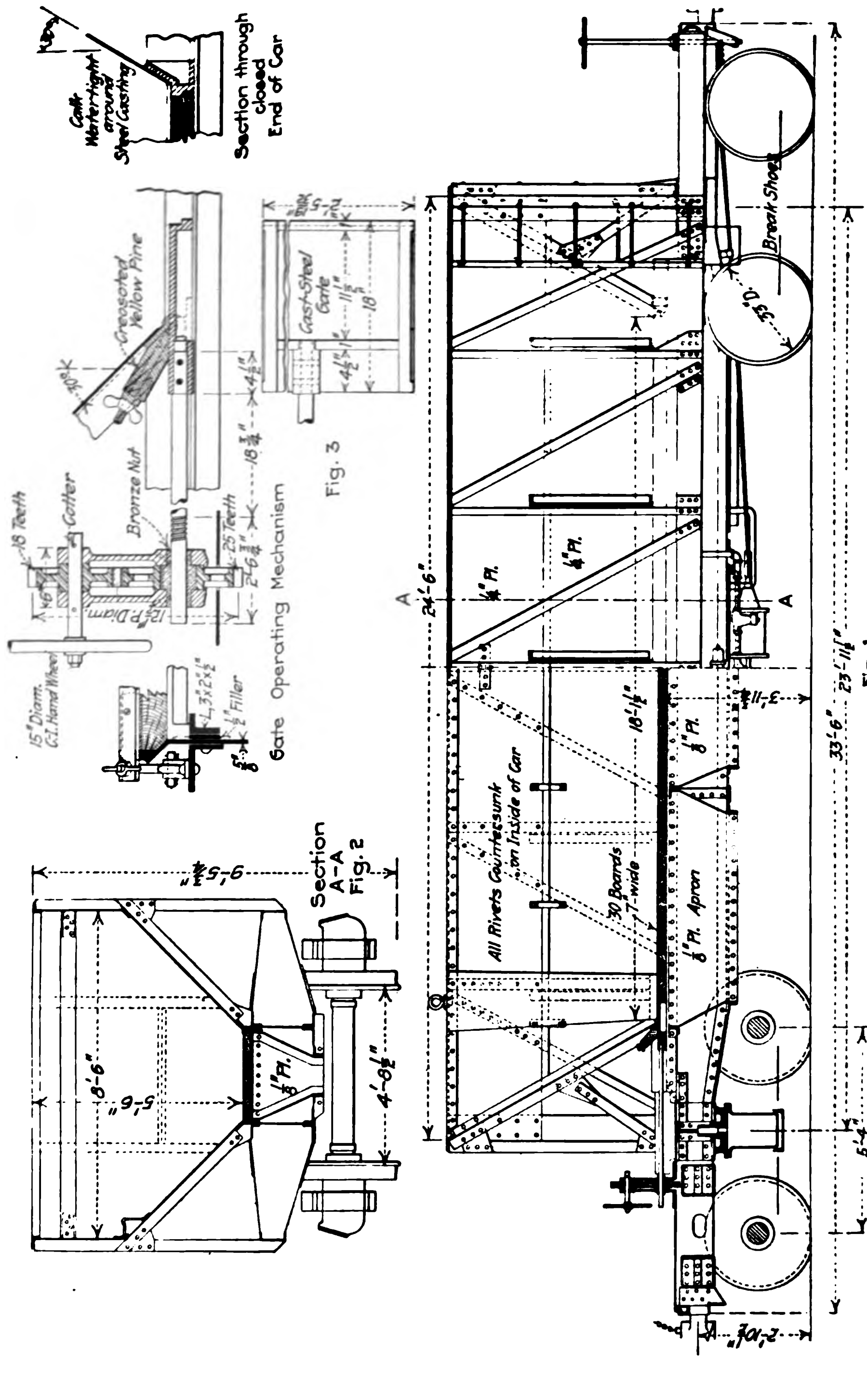


FIG. 1.—6. THE CONCENTRATE CAR. FIG. 2.—SECTION A-A OF FIG. 1. FIG. 3.—GATE-OPERATING MECHANISM OF CONCENTRATE CAR.

which is nearly vertical, until the next slat is partly uncovered, when it is pulled toward the gate end of the car. This operation progresses until the opposite end of the car is reached. The slats and the grooves supporting them are then carefully swept off and cleaned with a broom; the gate is screwed back, pressing the slats together, and the plug is lowered into place. The car is then ready to be taken back to the concentrator for another load.

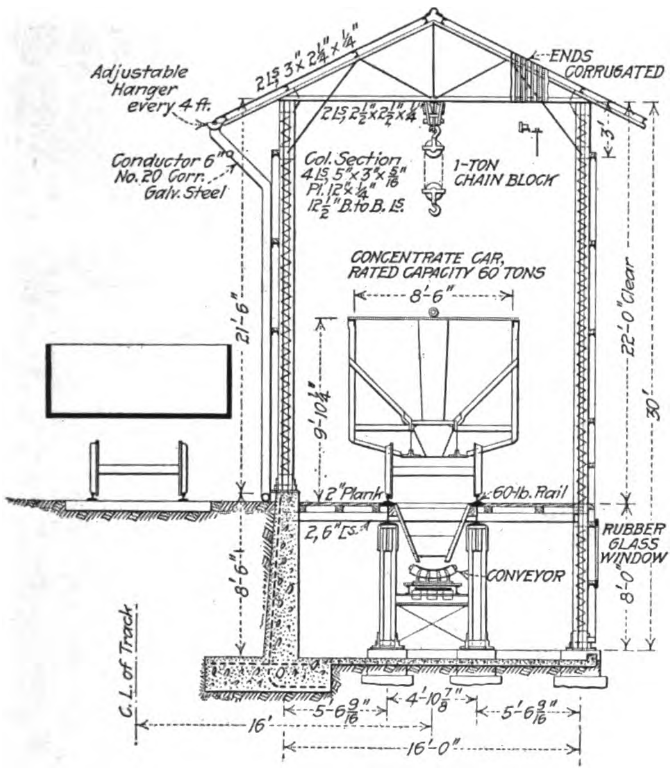


FIG. 4.—SECTION THROUGH CONCENTRATE UNLOADING POCKET.

## (b) *Concentrate Sampler*

Fig. 5 shows diagrammatically the scheme used for sampling the concentrates. It will be noted that the conveyor from the unloading pocket discharges onto a shuttle conveyor (a tripper on the former conveyor could have been substituted for the shuttle conveyor but it would not have handled the sticky concentrates as well). Buckets attached at their ends to chain belts are arranged to cut through the stream of concentrates as they are discharged onto the shuttle conveyor. The chains are driven and supported by sprocket wheels. The arrangement is such that at the highest point in their travel the buckets are in an inverted position. At

this point, they pass under a revolving shaft which has a number of knockers, made of 6-in. belting, attached to it. These knockers slap the bottom of the buckets, jarring loose any concentrates tending to stick to them. This arrangement works satisfactorily on ordinary table and vanner concentrates, but is not satisfactory with flotation concentrates on account of the varying load on the conveyor belt, the flotation concentrates being unloaded from the car in large chunks.

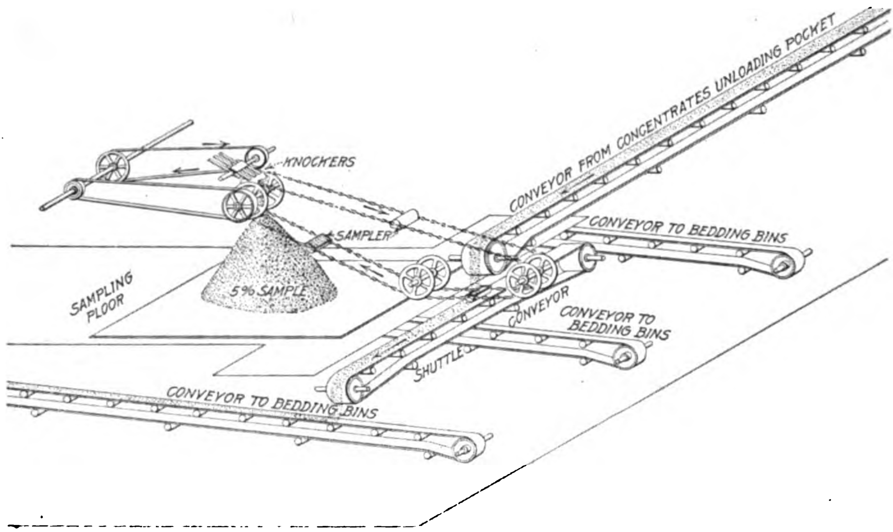


Fig. 5.—Arrangement for Sampling Concentrates.

## (c) *Bedding Bins*

Fig. 6 shows a cross-section of the bedding bins used in connection with the concentrate car and sampler just described. The bins are 150 ft. long and have a total capacity of 9,000 tons of charge. The concentrates are delivered by belts from the car-unloading pocket to any one of the three conveyors passing over the bins. Each belt has a motor-operated tripper which travels back and forth throughout the length of the bin. Each tripper is arranged with two cross-belts, which discharge material on both sides of the tripper, instead of the ordinary chutes that would clog with the sticky concentrates. These cross-belts also permit of a wider top on the ore bed than could be obtained with ordinary chutes. The cross-belts of the tripper as well as the long belts running over the bins and all the other belts handling sticky concentrates are speeded to 400 ft. per minute, which aids in discharging the sticky material from the belt when it passes over the pulley at its head end or the pulley of the tripper. An oscillating deflector is so

arranged that the material passing over the tripper is discharged, first on one of the short cross-belts and then on the other. This is done to provide an even distribution of the material over the top of the beds, making the height of the bed on both sides of the tripper the same. The tripper is reversed at each end of its travel by the reversal of its propelling motor through a magnet-switch control. The bins have "V" bottoms with a slot 2 ft. 4 in. wide running throughout their length. This slot is covered with short planks which are readily removed, and as the reclaiming progresses, they are merely moved back in the slot.

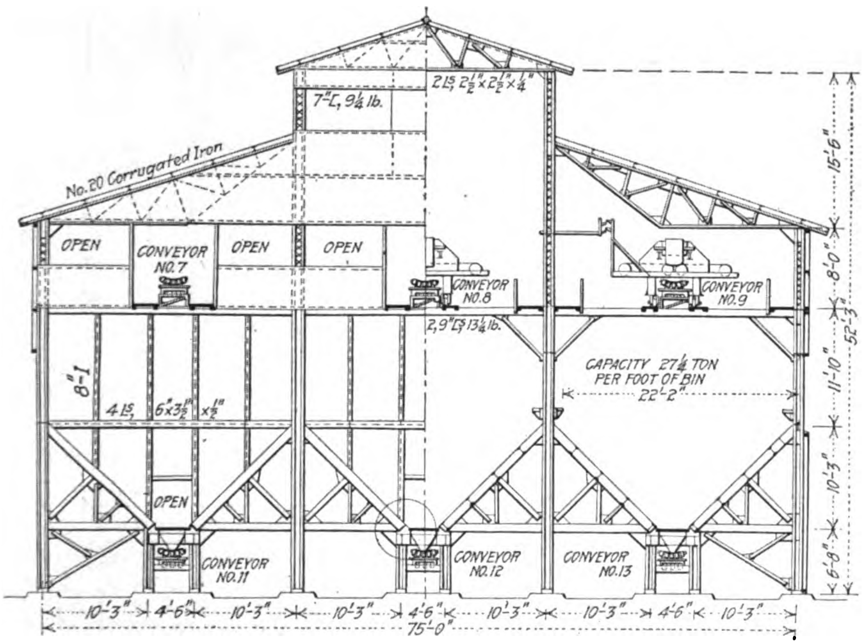


FIG. 6.—CROSS-SECTION OF BEDDING BINS.

The concentrates are raked down with hoes and allowed to discharge, through the opening in the bottom thus provided, onto the conveyor belts underneath. The concentrates are bedded in these bins with a proper amount of limestone and pyrite and the available secondaries, each bin comprising a bed. Storage bins for storing finely crushed lime rock, secondaries, pyrite, etc., are provided and arranged so that the fine material from these bins is delivered by a conveyor and discharged just ahead of the concentrates onto the belts leading up over the bedding bins. The material next to the bedding belts, therefore, consists largely of finely crushed lime rock, which is a further aid in the discharge of the sticky concentrates from these belts.

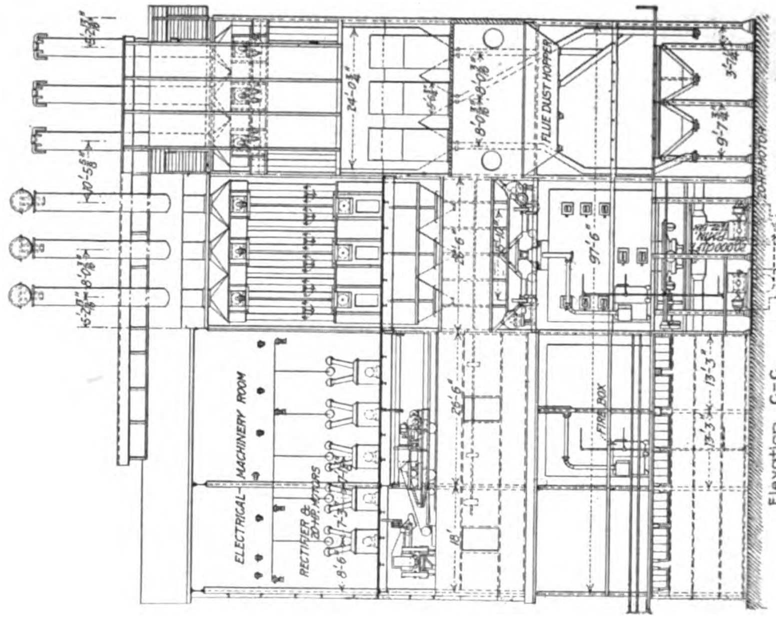


FIG. 8.—SIDE ELEVATION OF ROASTER OR DRYER PLANT, INTERNATIONAL SMELTING CO.

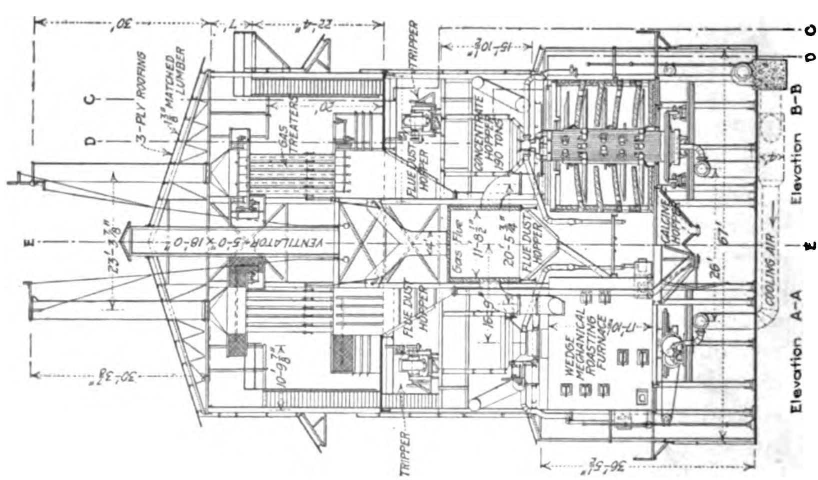


FIG. 7.—END ELEVATION OF ROASTER OR DRYER PLANT, INTERNATIONAL SMELTING CO.

## Roaster or Dryer Plant

Figs. 7 and 8 show end and side elevations respectively of the International Smelting Co.'s roaster or dryer plant. The main features of this plant consist in the method used for conserving the heat and in the placing of the Cottrell fume treaters directly above the furnaces. The roaster furnaces in this plant are used for drying the concentrates and are not used for roasting, as the charge contains no excess sulphur; in fact it has been necessary to have some pyrite shipped in from Bisbee in order to raise the sulphur content of the charge sufficiently to prevent the production of mattes too high in copper for good clean slags.

### (a) *Conservation of Heat in Furnace*

The furnaces are air-cooled. The cooling air, after passing through the furnace arms, is conducted up the central shaft to the top, thence down to the oil fireboxes connecting with the two lower hearths of the furnace. This gives preheated air for the combustion of the oil and conserves the heat absorbed by the cooling air. To further conserve the heat of the furnace, the linings are 12 in. thick. The outer 4 in. of lining, between hearths, is made of "Nonpareil" insulating brick. The manufacturers of these brick claim that 4 in. of their brick is equivalent in heat insulating value to 40 in. of ordinary brick. The tops of the calcine hoppers of these furnaces are insulated from the lower floor of the furnace building, just above them, by an air space. The calcine hoppers have non-conducting linings consisting of 2½ in. of Nonpareil insulating brick and a 1-in. layer of reinforced concrete. The reinforced concrete is to take the wear and erosion caused by the heated material inside the hopper, as the Nonpareil brick will not stand much wear or rough usage.

### (b) *Cottrell Dust-collecting System for Furnaces*

The Cottrell equipment for treating the gases from this plant consists of 12 units of 20 tubes each. The tubes are of lap-welded steel 13-in. in outside diameter, and 15 ft. in length. They are flanged outward, or "Vanstoned" at each end, which reduces the brush discharge caused by sharp corners and makes a good means of connection to the upper and lower diaphragms. Suitable lugs are welded to the pipes against which the hammers strike when vibrating the tubes to shake off the collected dust. The electrical equipment for this installation is so arranged that the voltage used may be varied from 50,000 to 100,000 volts. The treater tubes were planned for a velocity of 5 ft. per second for the gases passing up through them. The recovery of dust by this treater is practically perfect.

The main feature of this installation is in its arrangement which permits the gases to pass directly up from the furnaces through the header flue, the tubes and the stacks, the gases rising continuously from the furnace hearths to the outlet of the stacks.

## Blast Furnaces

Fig. 9 shows a general end elevation of the blast furnaces at the Calumet & Arizona Mining Co.'s plant at Douglas. These furnaces are

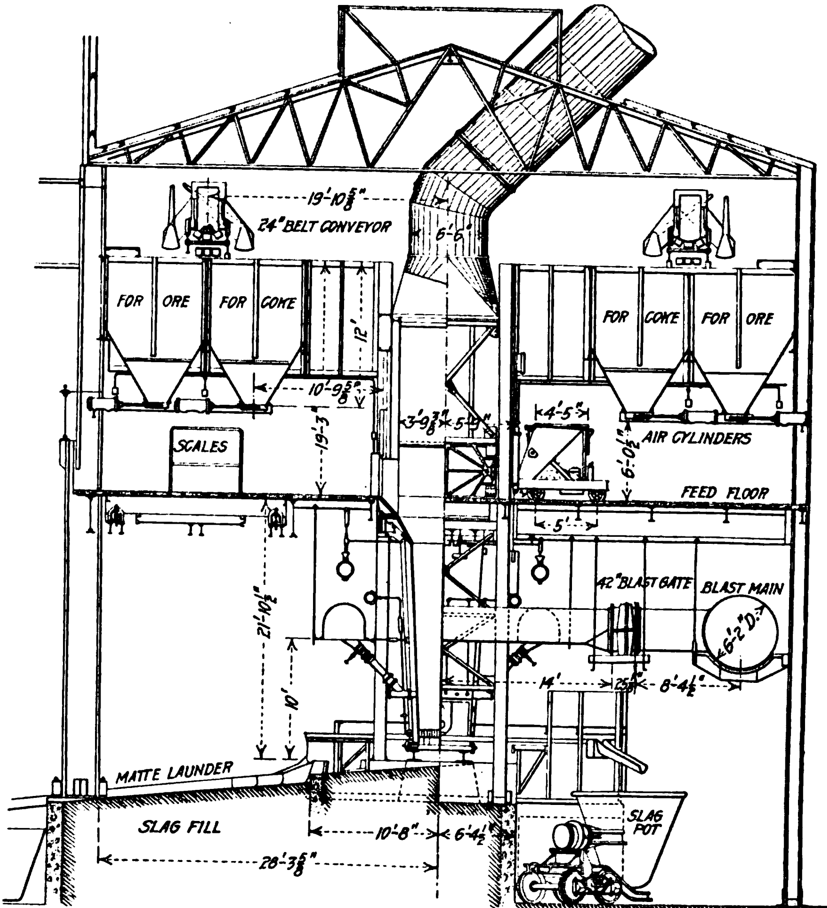


Fig. 9.—End Elevation of Blast-furnace Plant, Calumet & Arizona Mining Co., Douglas.

40 ft. long by 4 ft. wide at the tuyères. The ores and materials smelted in these furnaces are bedded so that the charge coming to the charge bins over the furnaces is thoroughly mixed. A coke bin and an ore bin are pro-

nstallation is in its arrangement wh
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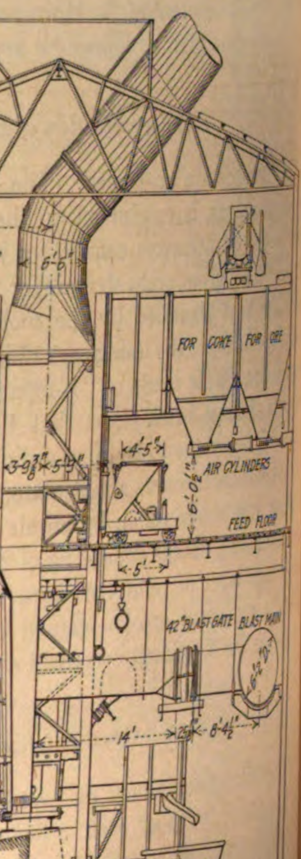
## T FURNACES

elevation of the blast furnaces at t
s plant at Douglas. These furnaces






vided on each side of each furnace. Each furnace has fou
two on each side. The cars are 20 ft. long. When receivi
ore or coke, the cars are resting on track scales. Each car
partments and the bins overhead have a gate corresponding
partment, so that the amount of charge for any part of th
be regulated to suit the conditions. The car is propelled
to the charging position at the furnace, a distance of 7
(according to whether a charge of coke or a charge of ore is
by means of an electric motor geared to two wheels of the
shows a view of the charge car under the charge bins.

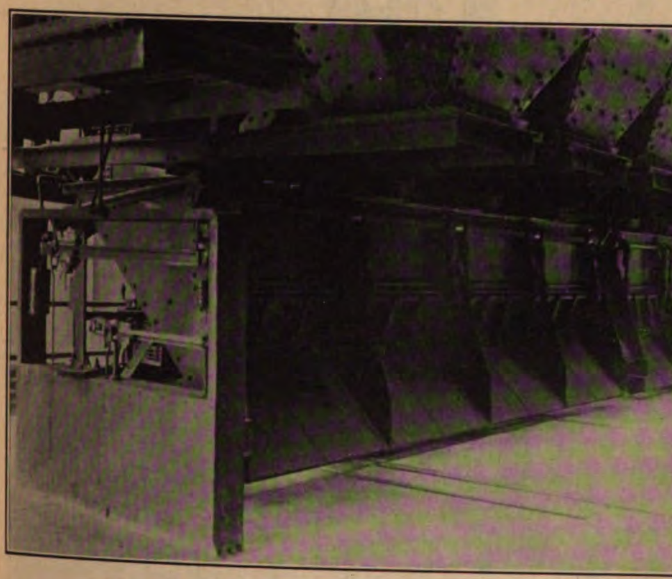


FIG. 10.—VIEW SHOWING CHARGE CAR UNDER BINS.

The furnace side jackets are in one length 16 ft. 6 in. lon
in. wide. It will be noted that the bustle pipe is combined w
or girder which takes the outward thrust of the jackets at the
This simplifies the construction and gives more room aroun
pipe for the water piping, etc. Fig. 11 is a view showing
and bustle pi

webs of the beams and a nut on the threaded end holds a lug against the bottom end of the jacket.

The furnace tops and charge doors are of structural steel in the form of air jackets. An air space is provided between the inner and outer sheets of the furnace tops. The lower end of this air space connects with the outside air while the upper end connects with the inside of the furnace top. The cool air is drawn in at the bottom and discharged at the top into the furnace gases going to the chimney, thus insuring a positive air circulation which keeps the sheets of the air top properly cooled. The furnace charge doors are jacketed in a similar manner.

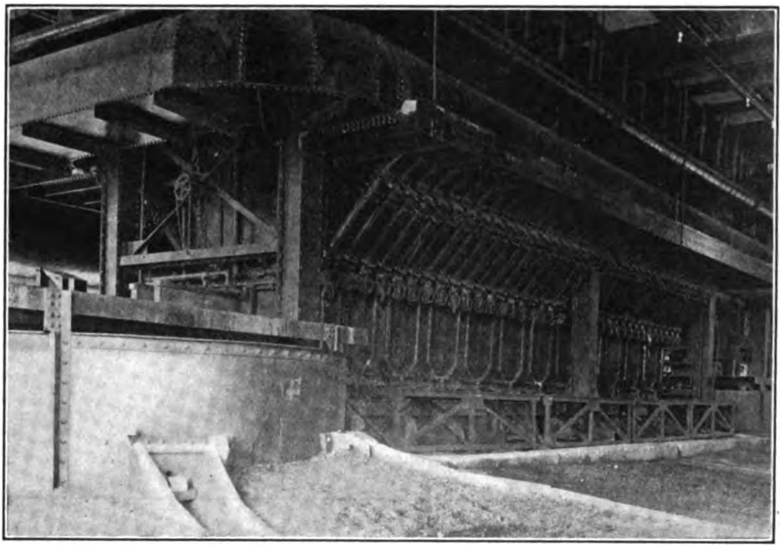


FIG. 11.—VIEW SHOWING TUYÈRES AND BUSTLE PIPE, BLAST FURNACE OF CALUMET & ARIZONA MINING CO., DOUGLAS.

These furnaces have been in use for 3 years, smelting as high as 1,400 tons of charge each per day, and no repairs or replacements have been made in furnace water jackets and no warping has taken place in the air-cooled tops.

## REVERBERATORY FURNACES

Fig. 12 shows a side elevation of a typical reverberatory furnace at the International Smelting Co.'s plant which has several features worthy of mention.

### (a) *Furnace Bottoms*

It was especially desirable at this plant, on account of difficulties at another plant, to avoid any trouble with furnace bottoms in starting up.

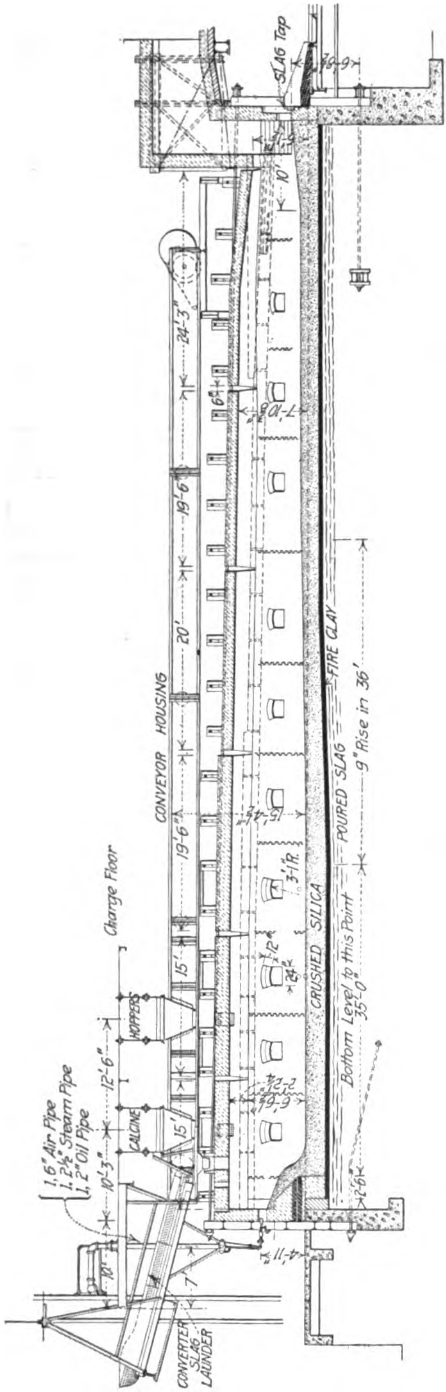



FIG. 12.—LONGITUDINAL SECTION OF REVERBERATORY FURNACE, INTERNATIONAL SMELTING CO.

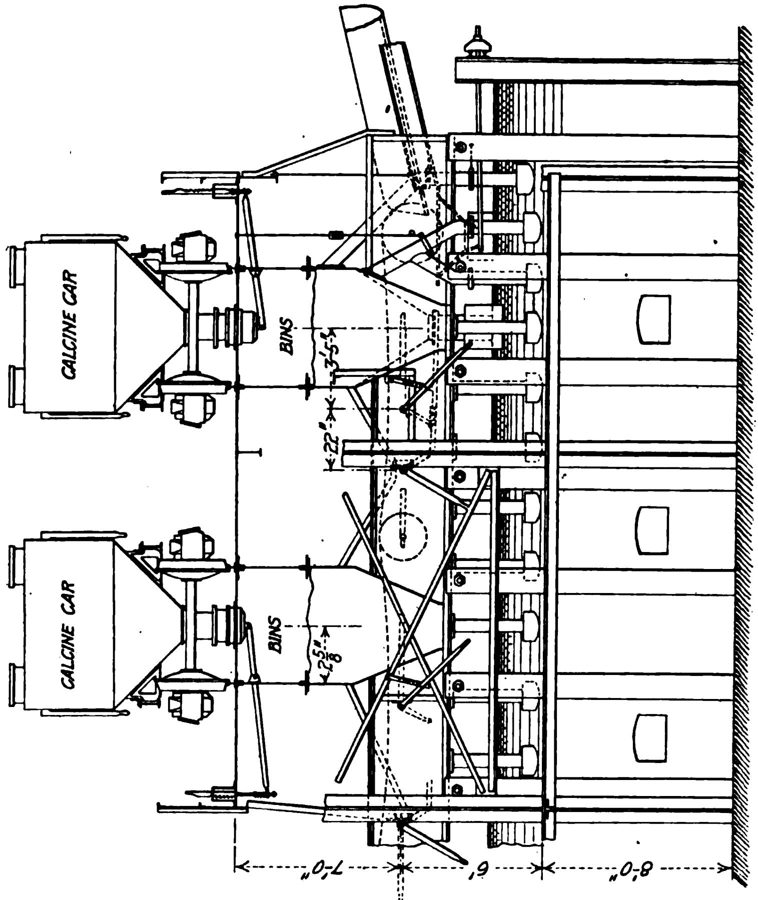



FIG. 14.—SIDE ELEVATION OF REVERBERATORY FURNACE SHOWING CHARGING ARRANGEMENT.

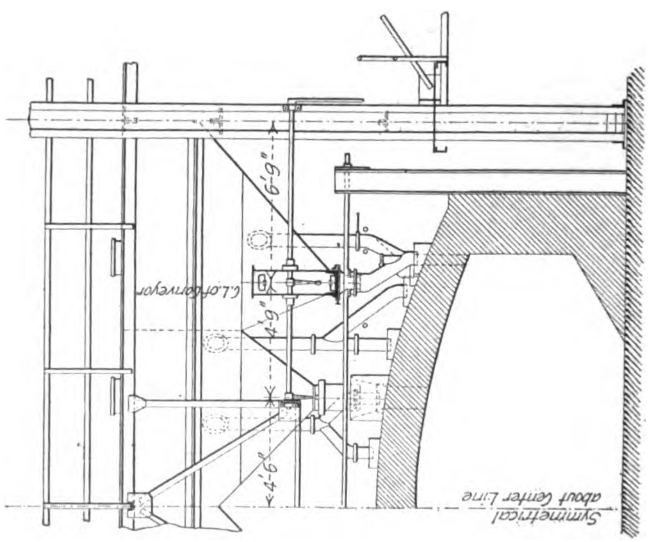



FIG. 13.—SECTION REVERBERATORY FURNACE SHOWING CHARGING ARRANGEMENT.

To accomplish this, broken slag, brought in railroad cars from the Old Dominion smelter, was melted in a small blast furnace, obtained by setting up a number of discarded furnace jackets and a motor-driven blower, on the site for the reverberatory furnaces. The molten slag thus obtained was conducted in launders to the foundations of the reverberatory furnaces. Heavy concrete beams and struts were provided between the furnaces for taking the thrust from the lower ends of the buck stays. Later, the spaces between the concrete struts and the beams were filled in with molten slag obtained from the regular operation of the reverberatory furnaces. The slag bottom was covered first with a 4-in. layer of fire clay and then a 27-in. layer of silica (94 per cent. $SiO_2$) crushed to minus $\frac{1}{4}$ in. These bottoms gave no trouble whatever in starting up.

### (b) *Furnace Fettling and Charging System*

The method of charging, instead of fettling only, along the side walls of reverberatory furnaces, now so successfully used by the Canadian Copper Co. and the Anaconda Copper Mining Co., was developed after the construction of this plant was well along. The plant was laid out for charge tracks running at right angles to the furnaces near the firing end. In order to distribute the charge along the sides of the furnaces, charge hoppers were located under the charge tracks directly over the side walls of the furnaces. Drag-chain conveyors were installed, one over each side of each furnace, which received the charge from the charge hoppers (see Figs. 13 and 14). Under these conveyors, approximately every 30 in., suitable down-spouts with gates are provided so that the charge may be distributed along the side walls of the furnaces throughout their length. The bridge wall and side walls at the firing end of the furnaces are charged by drawing directly from the hoppers through suitable spouts.

The floor over the charge tracks around the skimming end of the furnace is paved with firebrick.

### (c) *Calcine Cars*

Figs. 15 and 16 show end and side elevations respectively of the calcine car used at the International Smelting Co.'s plant. The feature of this car is the arrangement used for reducing dust losses in receiving and discharging a charge. The car is equipped with four sliding sleeves in the top which are spaced to register with the discharge openings of the calcine hoppers at the roaster plant. When the car is spotted under the roaster hoppers in its proper position, the sleeves are forced up against the calcine hoppers by means of levers on the side of the car. The operating lever is of flat spring steel so that the sleeve can be firmly pressed into con-

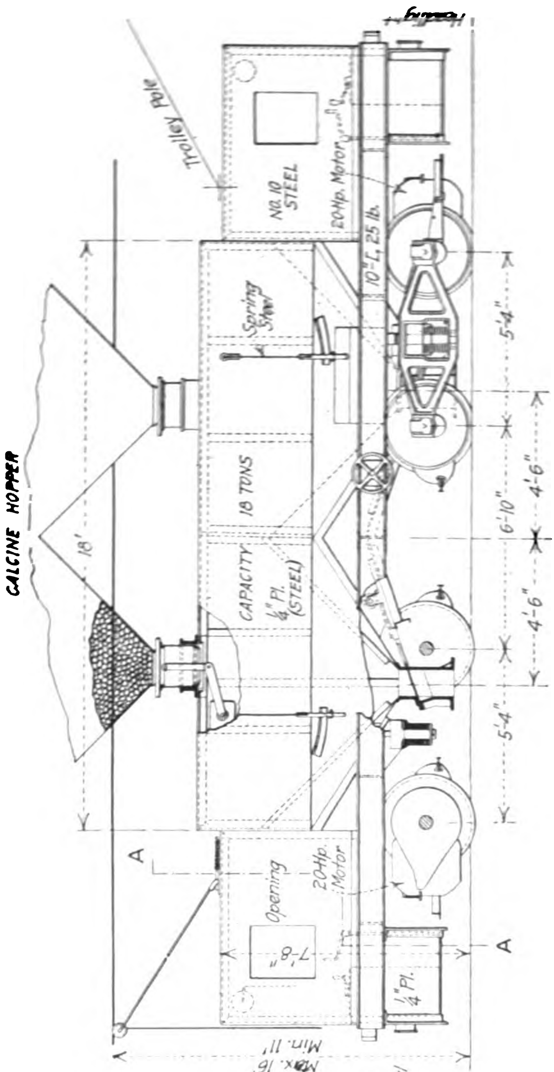



FIG. 16.—SIDE ELEVATION OF CALCINE CAR.

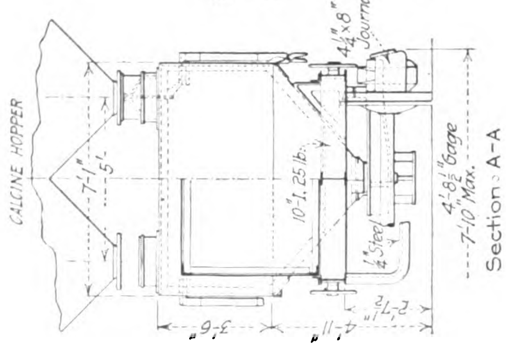



FIG. 15.—END ELEVATION OF CALCINE CAR.

tact with the flange at the bottom of the calcine hopper, and held there by a notch in a quadrant on the side of the car. The spring handle insures a good pressure between the sleeve in the top of the car and the flange of the calcine hopper regardless of the small variations in the distance between the top of the car and the calcine hoppers or any lost motion in the connection of the lever with the sleeve. An air vent is provided in the car made of fine woven wire mesh.

Fig. 14 shows the sleeves in the top of the reverberatory furnace charge hoppers for receiving the charge from the calcine car. These sleeves are similar to the sleeves in the top of the calcine car. By pressing down on the lever, which is operated from the charge floor, the sleeve is forced up against the flange of the discharge spout of the car. Notches are provided in the lever which engage a catch and thus hold the sleeves in their upper position while the car is being discharged. The pressure of the handle is transmitted to the sleeve, as will be seen from the figure, through a helical spring, so that a good contact between the sleeve and the spout of the car is assured regardless of the lost motion and wear in the operating levers and the variations in the distance between the sleeves in the charge floor and the flange on the discharge spout of the car.

### (d) *Baffling in Waste-Heat Boilers*

By reference to Fig. 17, it will be noted that the header flue between the reverberatory furnaces and the waste-heat boilers, and the connections between the header flue and boilers are well up above the furnace floor level, giving ample head room for the convenience and comfort of the workmen; the bottom of the header flue is 10 ft. 6 in. above the furnace floor level. Also, it will be noted that the boiler-room floor level corresponds to the furnace floor level. This is made possible by reversing the baffling in the Stirling boilers so that the gases enter the front of the boiler near the top and leave the boiler at the rear near the bottom instead of as in the standard setting.

This arrangement of baffling gives equally as good results from the standpoint of water circulation in the boiler, priming, etc., as the standard baffling.

## Arrangement of Converter Plants

Southwestern practice is to have the converter plant adjacent to the reverberatory furnaces, and arranged so that the reverberatory matte is received through launders into ladles directly under the converter plant crane. Also arranged so that the converter slag may be poured from ladles by the converter crane back into the reverberatory furnaces through launders extending from the converter aisle to openings provided in the roofs of the reverberatory furnaces. The converter slag is usually

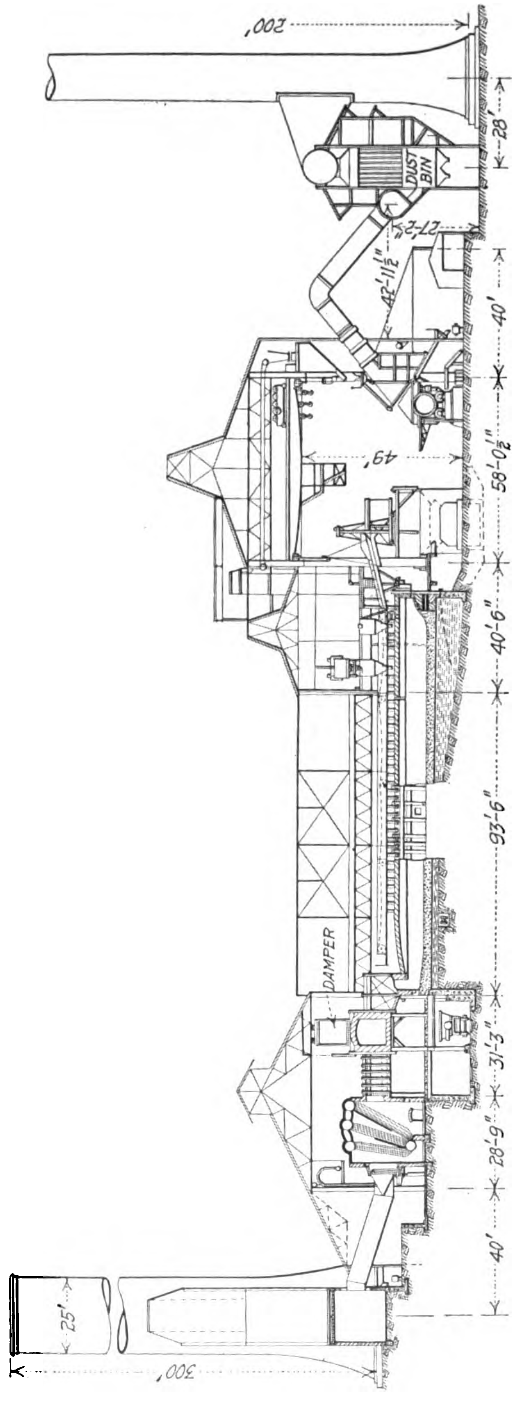



FIG. 17.—TYPICAL CROSS-SECTION OF THE INTERNATIONAL SMELTING CO.'S PLANT.

discharged several feet in front of the bridge-wall of the furnace and midway between the side walls. Fig. 17 is a typical cross-section of the International Smelting Co.'s plant and is typical of the new reverberatory smelting plants in Arizona. Fig. 18 is a plan of the same plant.

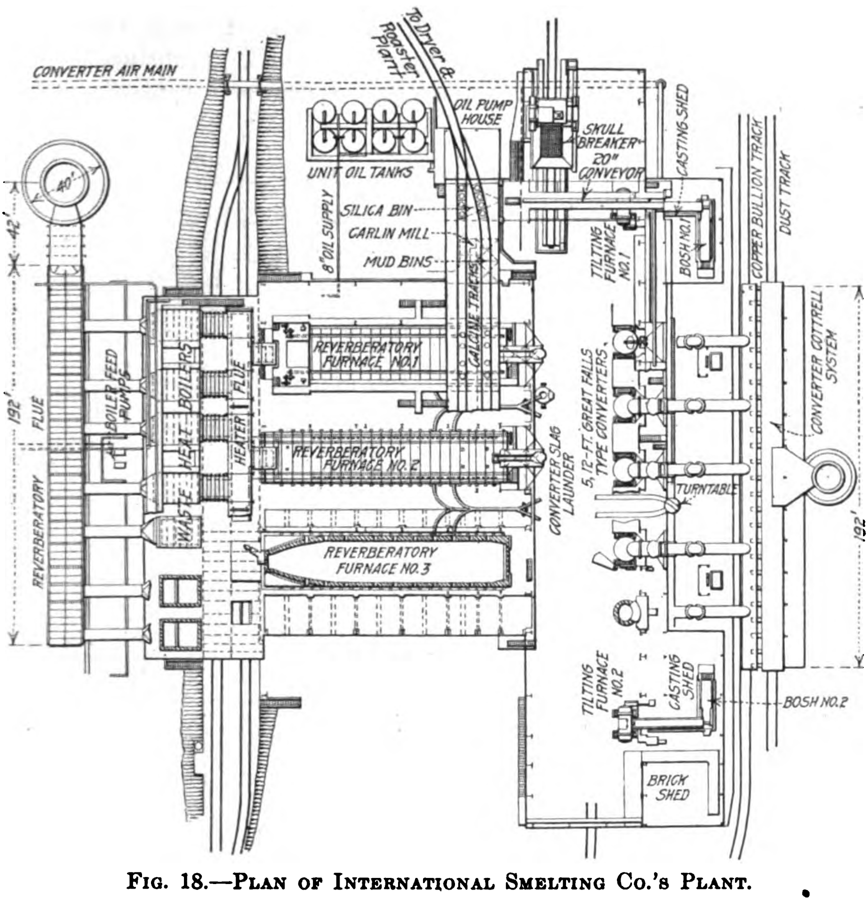



Fig. 18.—Plan of International Smelting Co.'s Plant.

### (a) *Siliceous Ore Charging Device for Converter*

Several of the plants are arranged so that the siliceous ore for the converters is drawn directly from bins overhead. Figs. 19 and 20 show a typical arrangement of bin, measuring hopper and chute for accomplishing this operation. This particular arrangement is installed at the International Smelting Co.'s plant. A weighing or measuring hopper is interposed between the overhead bin and the chute leading to the converter mouth. The hopper is mounted on springs and is connected by levers and links to an indicator readily seen from the operating floor. Any desired amount of charge can be weighed out into the measuring hopper

and discharged into the converter and this can be accomplished entirely through the operation of levers on the main floor of the converter plant.

## (b) *Converter Plant Cranes*

Most of the newer plants are equipped with 12-ft. Great Falls type converters. These converters weigh, when lined, approximately 70 tons. Instead of designing the converter cranes for a capacity to lift

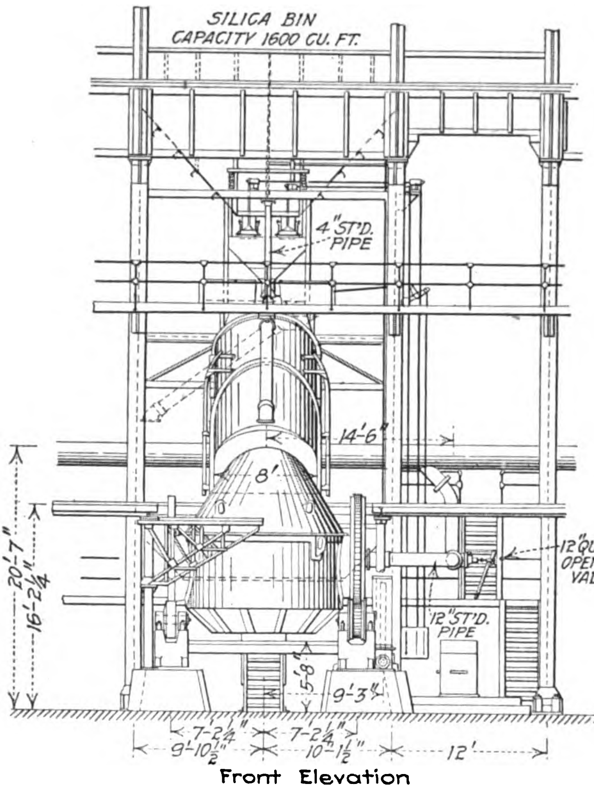



FIG. 19.—CONVERTER PLANT, INTERNATIONAL SMELTING CO.

a converter shell with its lining, as has usually been the practice heretofore, the cranes are designed for a lifting capacity of 40 tons, which is nearer right for the routine work of handling matte, copper and slag in large ladles. Converters with magnesite lining do not have to be moved into and out of their stalls often, but when it is desirable to do this two cranes are used with a lifting rig as shown in Figs. 21 and 22. The converter cranes in all of the Arizona smelting plants are operated by 250-volt direct current. The later cranes are equipped with magnet-switch control.

### (c) *Skull Breaker for Converter Plant*

Three of the Arizona converter plants are equipped with skull breakers and a fourth plant is now being thus equipped. Figs. 23, 24 and 25 show the general arrangement of the skull breaker at the International Smelting Co.'s plant. A hopper about 11 ft. wide by 24 ft. long is provided, the

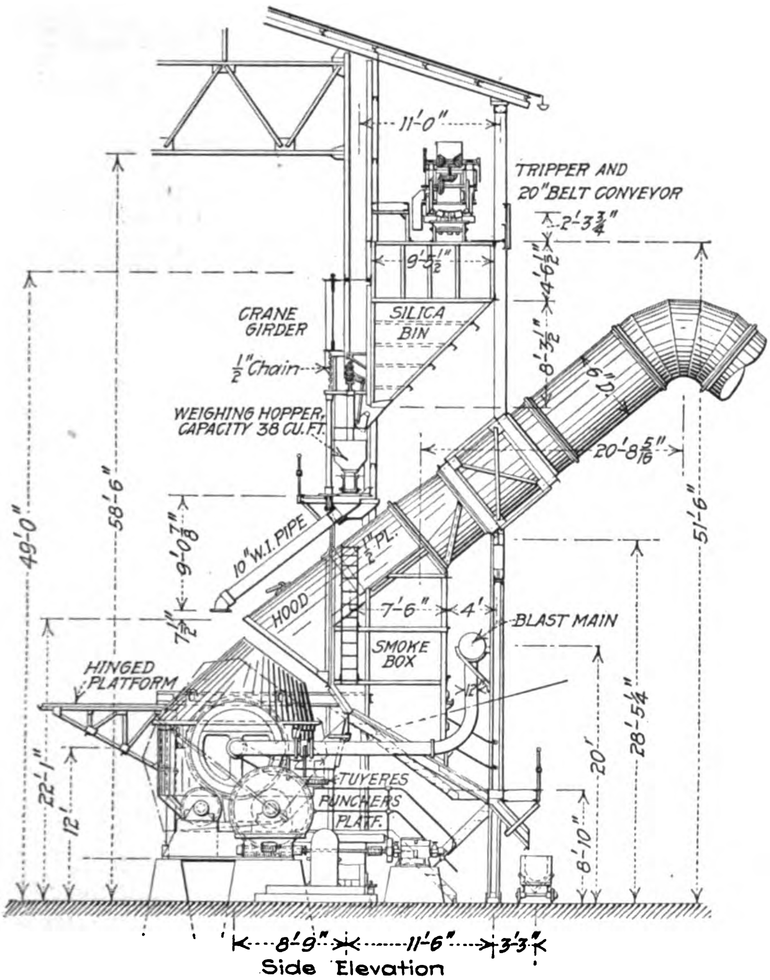



Fig. 20.—Converter Plant, International Smelting Co.

bottom of which consists of cast-steel bars with projecting lugs, making a grating with openings throughout about 9 in. square. A small traveling bridge is provided over this grating, and mounted on this bridge is a trolley which travels transversely to the path of the bridge. In the trolley is mounted a motor-driven hoist which raises a hammer in the form of a long

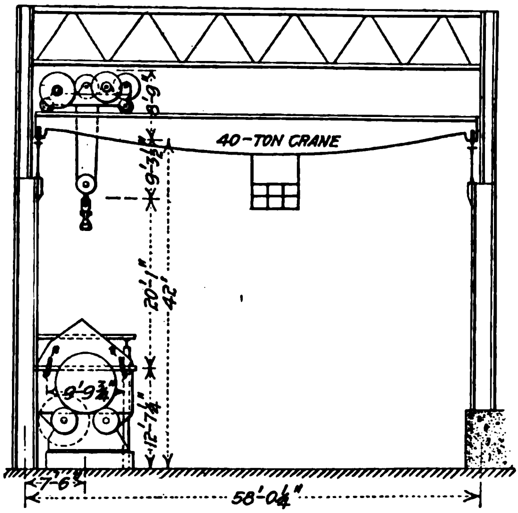



FIG. 21.—METHOD OF LIFTING CONVERTERS, CALUMET & ARIZONA MINING CO.

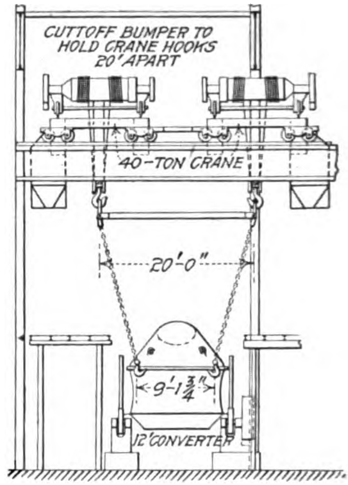
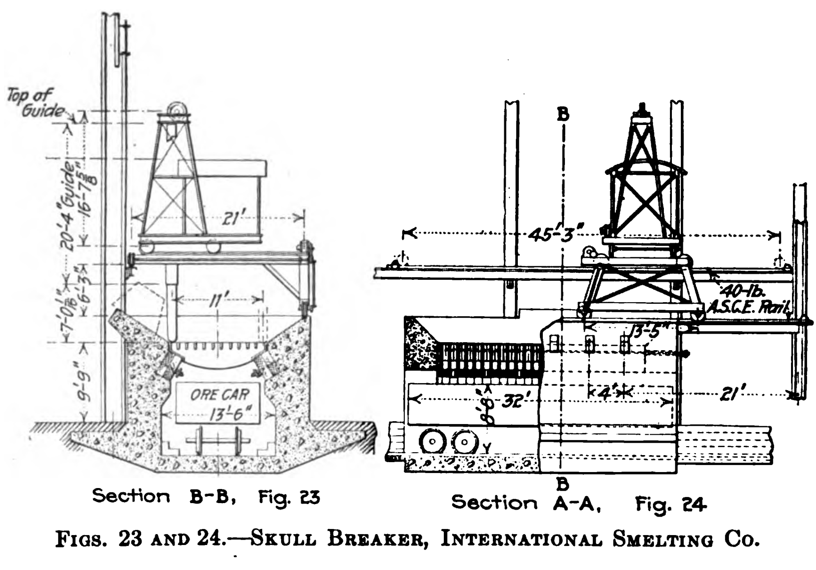



FIG. 22.—METHOD OF LIFTING CONVERTERS, CALUMET & ARIZONA MINING CO.




FIGS. 23 AND 24.—SKULL BREAKER, INTERNATIONAL SMELTING CO.

weight that works in a guide.. The hammer and guide are mounted on a trolley with the hoist. The hoist operator raises the hammer and allows it to drop on the skull in a manner similar to the operation of the ordinary type of pile-driver hammer. When the skulls are broken into fragments sufficiently small to pass through the grating, they drop into a steel railroad car underneath. The bridge and trolley travel permit the hammer to be spotted over any skull on any part of the grating.

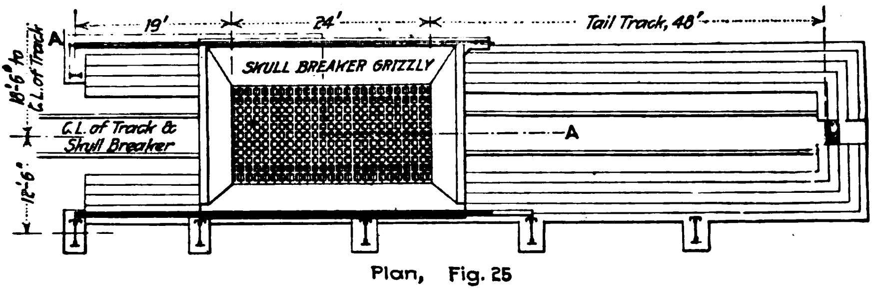



Fig. 25.—Skull Breaker, International Smelting Co. Plan.

## Chimneys

### (a) *The Largest Steel Chimney in the World*

Fig. 26 shows a general elevation with details of the large steel chimney at the plant of the United Verde Copper Co. at Clarkdale, Ariz. It is 30 ft. 9½ in. in diameter inside the steel plates and is 400 ft. 1 in. in height, from the base to the top. It is believed to be the largest steel chimney in existence at this time. It has a brick lining 4 in. thick throughout its height.

### (b) *Slag Foundation for Steel Chimney*

Fig. 27 shows the details of the foundation for a steel chimney at the smelting plant of the Calumet & Arizona Mining Co. at Douglas. The chimney is 305 ft. high from the top of the foundation and is 25 ft. 9½ in. in diameter, inside of the steel shell. It has a hollow-tile lining 4 in. thick throughout its height. The feature of the chimney is in the construction of its foundation. The foundation was cast from molten slag hauled to the site for the chimney in the usual slag pots, instead of concrete or masonry as is usual for chimneys of this type. A template for holding the anchor bolts was made of structural angles and channels supported on a central concrete pier and held from turning or moving by a second concrete pier at the outer circumference. The foundation

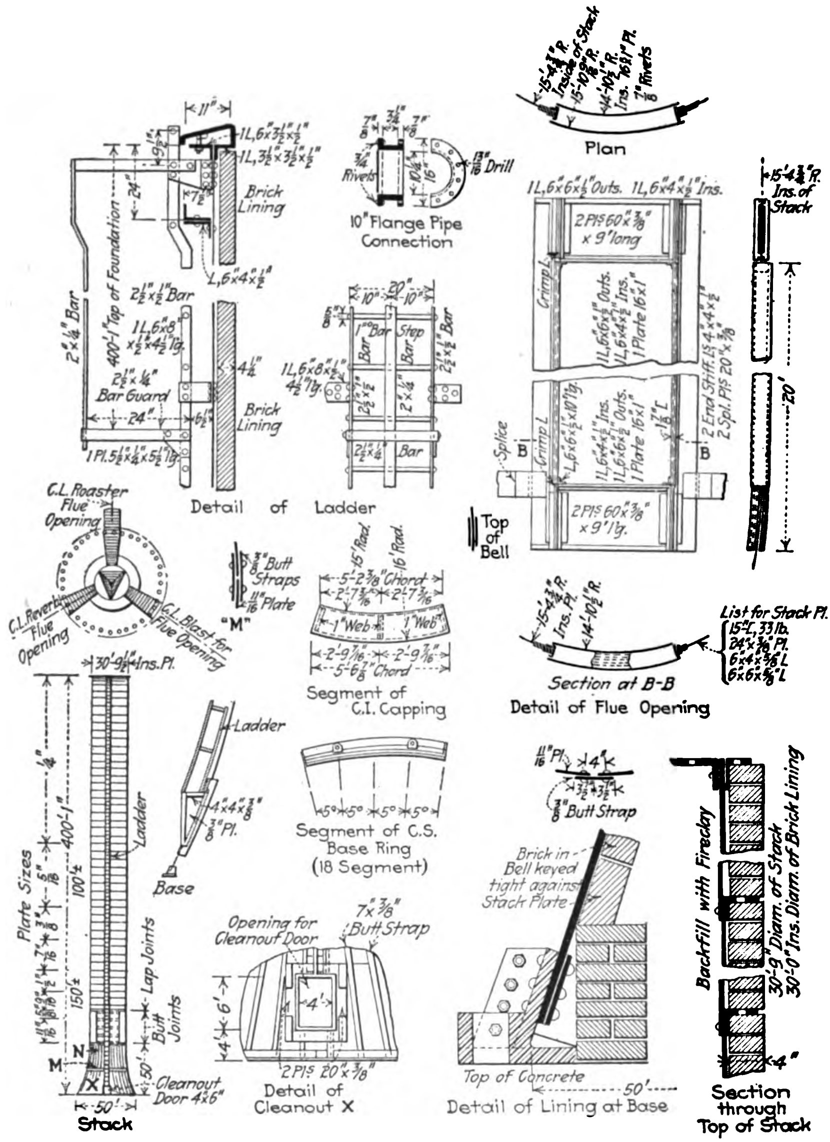


FIG. 26.—GENERAL ELEVATION AND DETAILS OF STEEL CHIMNEY, UNITED VERDE COPPER CO.

bolts were supported at their lower ends on small concrete piers and they in turn supported the template at other points of its outer circumference not supported by the concrete pier, each bolt having at the top a nut on the under side and one on upper side of the template,

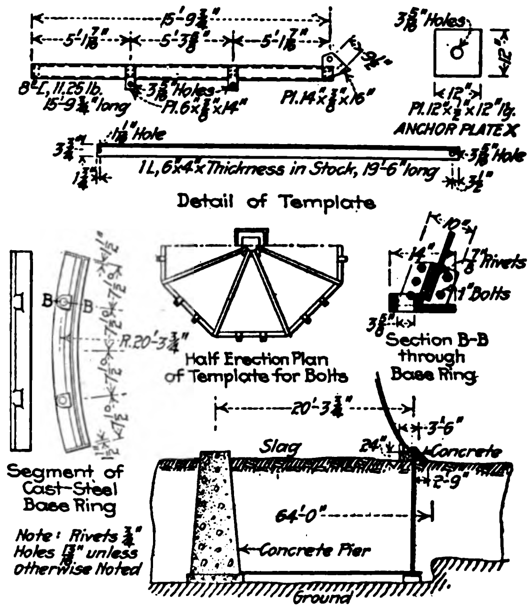



Fig. 27.—Details of Slag Foundation for Chimney of Calumet & Arizona Mining Co., Douglas.

forming a secure support. Large washers were provided for the bottom ends of the anchor bolts over which were laid old steel rails. Blast-furnace slag was then poured over the foundation and adjacent ground, forming the foundation for the chimney. A concrete capping was laid on top of the slag for the cast-steel base-ring of the chimney proper.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The 2,000-Ton Leaching Plant at Anaconda

BY FREDERICK LAIST,* B. S., AND HAROLD W. ALDRICH,† M. E., ANACONDA, MONT.

(Arizona Meeting, September, 1916)

AFTER a series of experiments covering a period of about three years, extending from the spring of 1912 to the spring of 1915, a 2,000-ton leaching plant for the treatment of the accumulated copper concentrator tailing was built and put into operation. During the experimental period, the first step was that of laboratory experiments or beaker leaches. The results on this small scale were so satisfactory that a

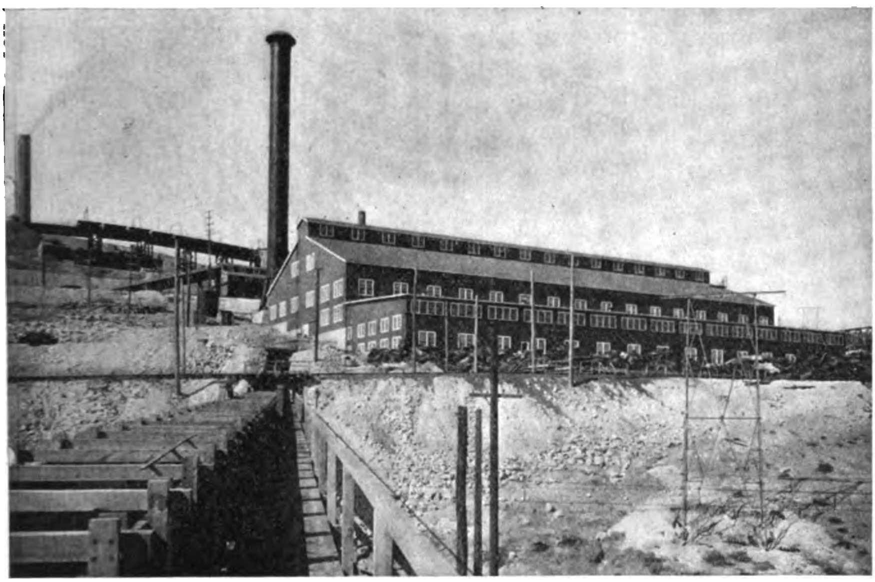


FIG. 1.—LEACHING PLANT AT ANACONDA.

small operating plant, capable of handling 5 tons of roasted tailing per day, was installed. Again, the results proved satisfactory and an 80-ton plant was built and operated continuously from August until February, 1913. In this plant, full-sized roasting and leaching units were

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† Superintendent of Leaching Plant, Anaconda Mining Co.

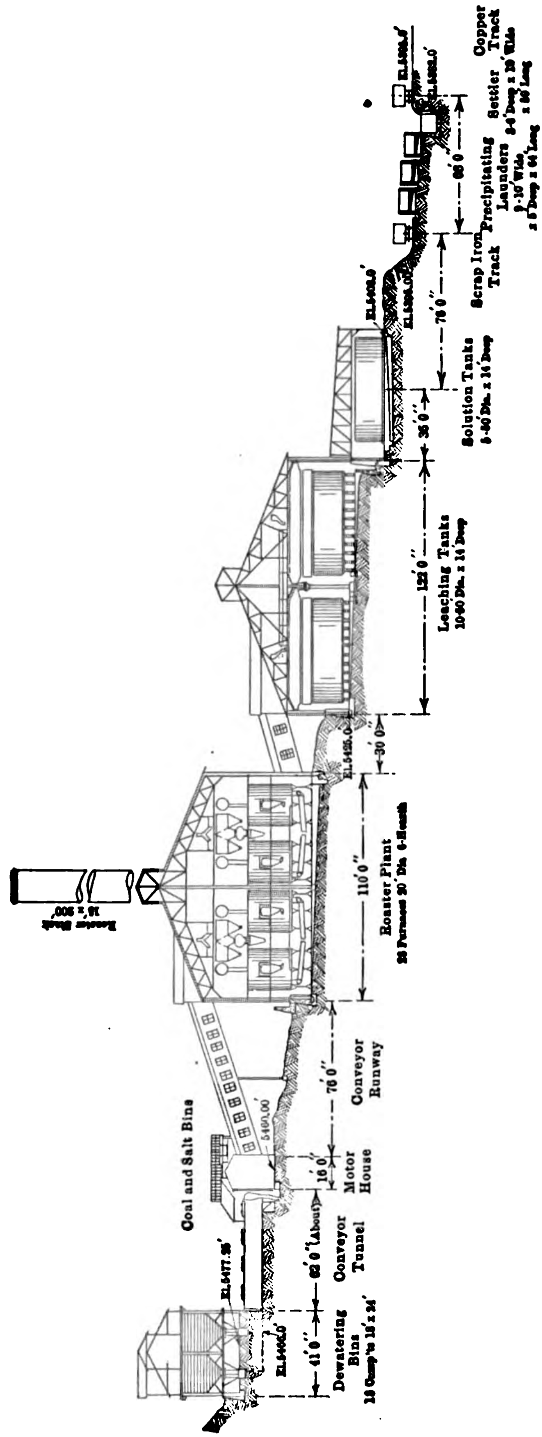


FIG. 2.—GENERAL PROFILE OF 2,000-TON LEACHING PLANT AT ANACONDA.

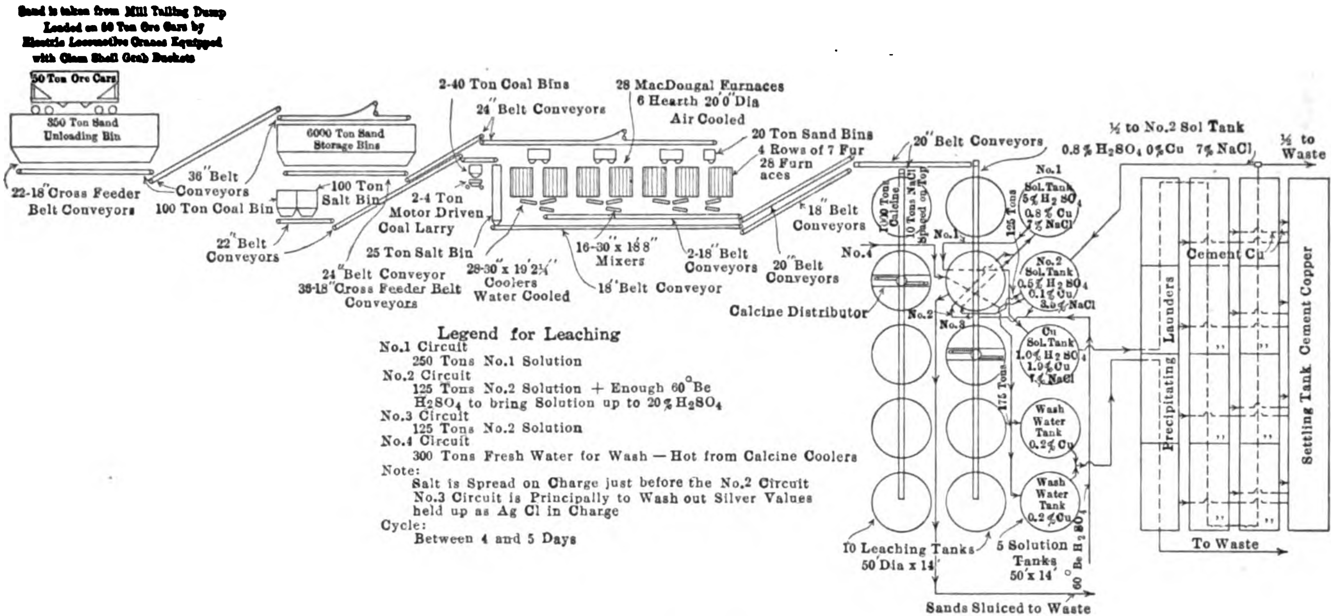


Fig. 3.—Flow Sheet of Leaching Plant.

used. The results obtained by the operation of the 80-ton plant proved that the roasting and leaching of deslimed concentrator tailing could be profitably done, so in the early spring of 1914, the construction of a 2,000-ton leaching plant was begun (Figs. 1 and 2). Operation of the plant began on May 13, 1915. A flow sheet of the plant is shown in Fig. 3.

## Leaching Plant Feed

The accumulated tailing in the New Works dump is estimated at about 20,000,000 tons. The dump consists of the concentrator tailing discharge, over a period extending from February, 1902, to the present date. According to daily samples taken during that period, the copper content is about 0.64 per cent. and the silver 0.48 oz. per ton. The peak of the dump, or the point where the tailing is discharged, carries about 0.75 per cent. copper, while down toward the toe, and where present excavation is taking place, the copper content is only 0.57 per cent. About 3 lb. per ton of the copper is oxidized, the remainder being sulphide.

The following shows an average screen analysis of the tailing in the dump:

*Cumulative Per Cent.*

| +20.0 | +40.0 | +60.0 | +80.0 | +100.0 | +160.0 | +200.0 | −200.0 |
|---|---|---|---|---|---|---|---|
| 22.6 | 64.7 | 82.0 | 89.1 | 93.8 | 97.7 | 98.5 | 1.5 |

An average analysis of tailing being treated at the present time follows:

| $H_2O$ Per Cent. | Cu Per Cent. | Ag Ounces | $SiO_2$ Per Cent. | FeO Per Cent. | S Per Cent. | $Al_2O_3$ Per Cent. | CaO Per Cent. |
|---|---|---|---|---|---|---|---|
| 5.63 | 0.573 | 0.487 | 81.0 | 2.8 | 1.9 | 9.8 | 0.4 |

The dump is being excavated by a Bay City Industrial Works electric hoist, equipped with a 4-yd. bucket. The hoist loads the tailing into 50-ton ore cars. These are hauled to the leaching plant storage bins in trains of 15 cars each. The hoist is capable of hauling 3,000 tons in 8 hr.

## Conveying and Storage Equipment

### *Unloading Pit*

The loaded cars of tailing are spotted by means of a 25-ton electric locomotive operated by the third-rail system. The unloading pit is a steel bin capable of holding 350 tons and is of sufficient length to allow three cars to be unloaded at a time. On top of the pit is a 2-in. grizzly, through which all tailing must pass before going to the storage bins. This grizzly serves to keep out rocks or lumps which would block the

feeders, furnaces, etc. Under the steel bin are 22 short-belt feeders, each feeding from its own gate and running at right angles to the length of the bin. These feeders discharge onto a 36-in. belt running lengthwise, which in turn delivers the sand to another 36-in. belt traveling on an 18° angle to the top of the storage bins. The system will handle over 3,000 tons of tailing in 8 hr.

### *Storage Bins*

The storage bins will hold 6,000 tons of sand tailing. This gives between two and three days' supply for the leaching plant, in case of railroad troubles or difficulties due to cold weather. The bins are of substantial wood construction and are inclosed, the walls and roof of the building consisting of wood sheathing covered with corrugated iron. The bins are arranged in a double row and are hopper-bottomed. The tailing is distributed over the bins by means of a 36-in. belt and movable tripper.

Underneath are 36 gates, each with its hopper from which an 18-in. belt feeder delivers to a 24-in. belt running the full length of the building, in the center. By means of other belts, traveling through a tunnel under the railroad tracks, and up an 18° incline, the feed is delivered to the top of the furnace building.

There are also small coal and salt storage bins from which a 22-in. belt system conveys coal and salt to bins in the roaster building, the salt being drawn from there to the leaching building as it is required. All belts run at approximately 400 ft. per minute.

## Roasting Equipment

The furnace building is 232 by 110 ft. and is of steel and concrete construction (Fig. 4).

There are 28 McDougall-type, six-hearth furnaces arranged in four rows of seven each. The furnaces are 20 ft. in diameter, each being equipped with two fireboxes, diametrically opposite, the flame entering over a fire bridge directly into the third hearth, the top hearth being designated as the first. The grate dimensions of each firebox are 3 by 4 ft. (Figs. 5 and 6).

Each furnace has a 20-ton feed hopper, to which the tailing is delivered by means of two 24-in. belts each equipped with a movable tripper. The furnaces are fed by 14-in. belt feeders drawing from these hoppers, the amount of feed being controlled by gates which are operated by means of a screw adjustment, the feed dropping through a hole in the top arch, directly onto the top floor (Fig. 7). During the time the feed is on the two upper floors, it is dried and heated; as it drops

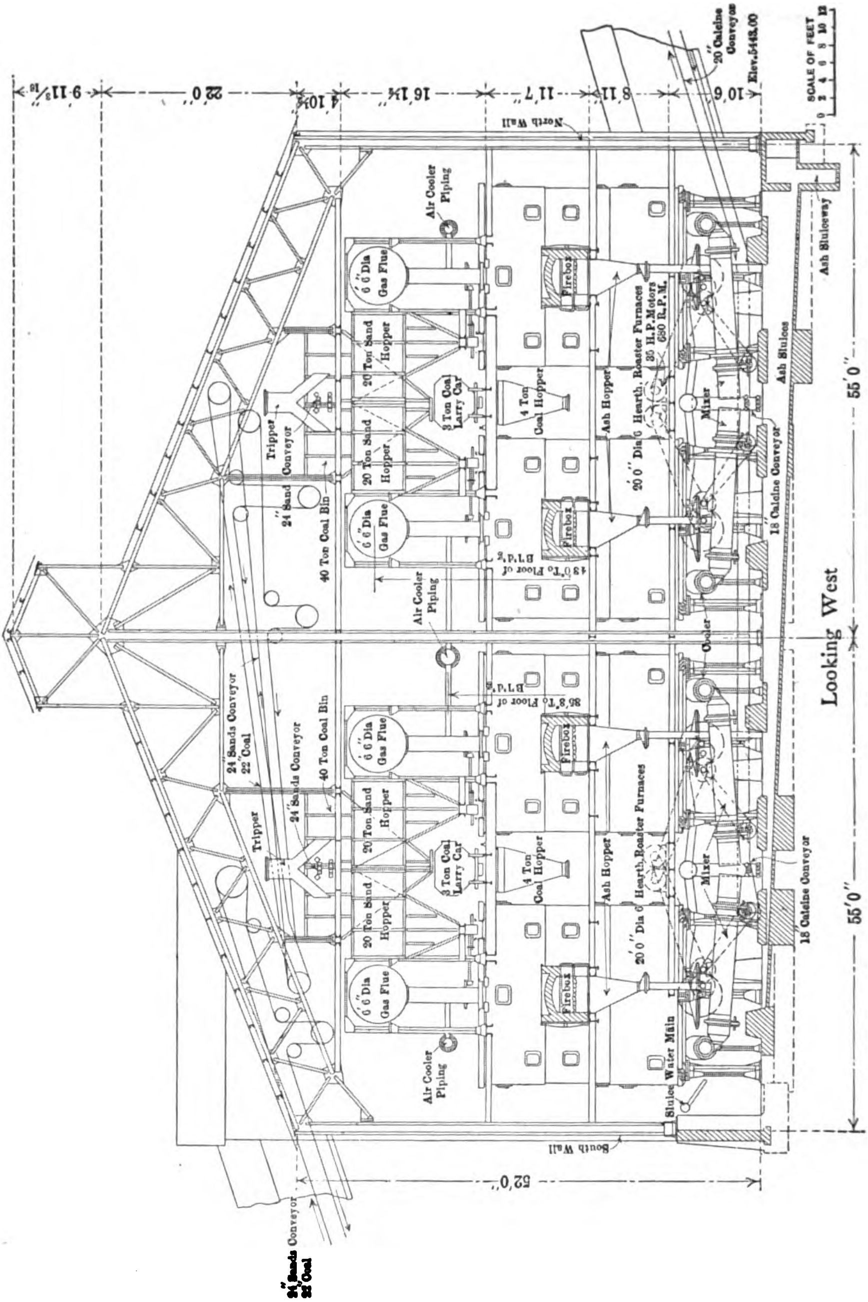


FIG. 4.—CROSS-SECTION THROUGH ROASTING PLANT.

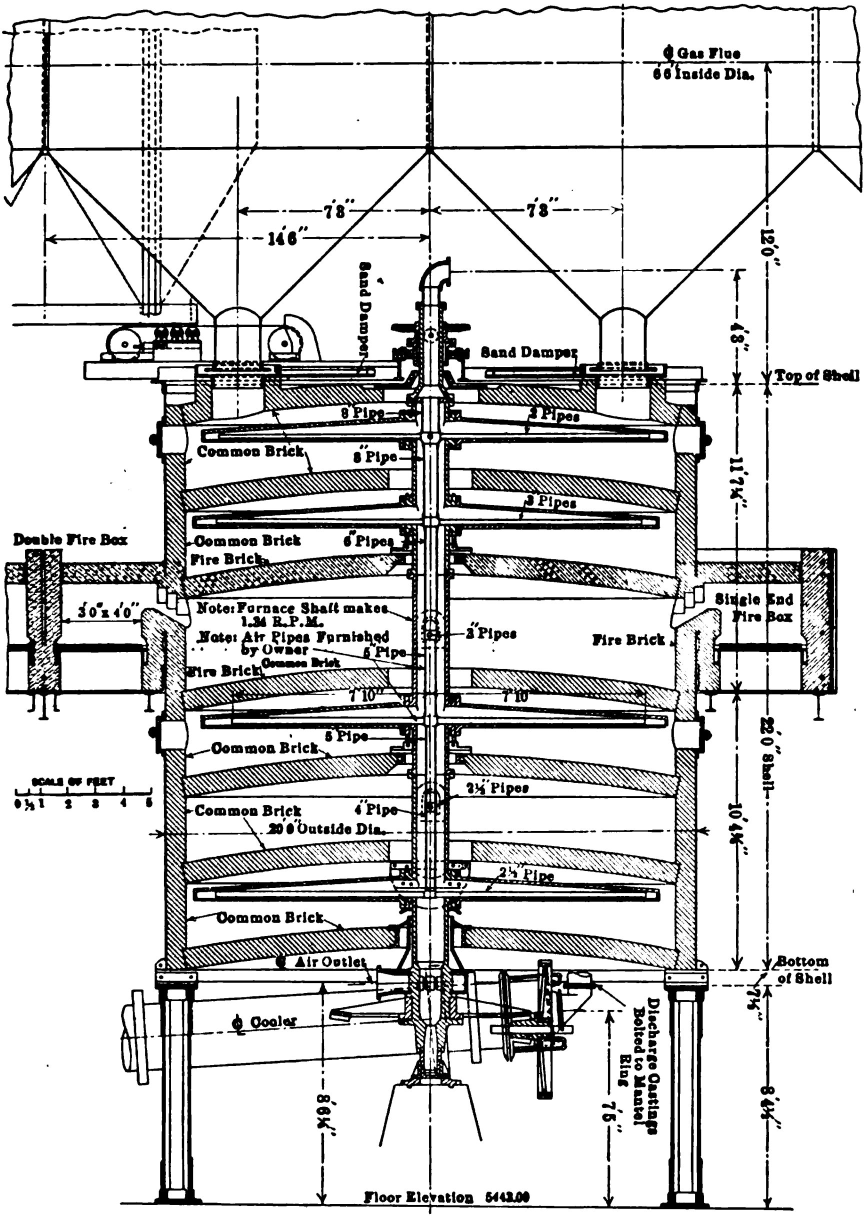


FIG. 5.—VERTICAL SECTION THROUGH McDOUGALL FURNACE.

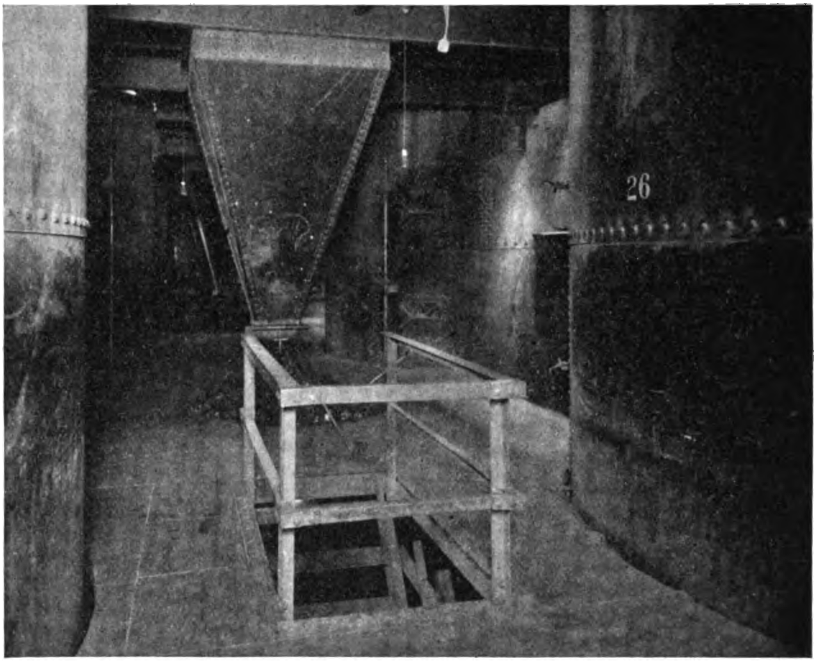


FIG. 6.—FIRING FLOOR OF ROASTER PLANT.

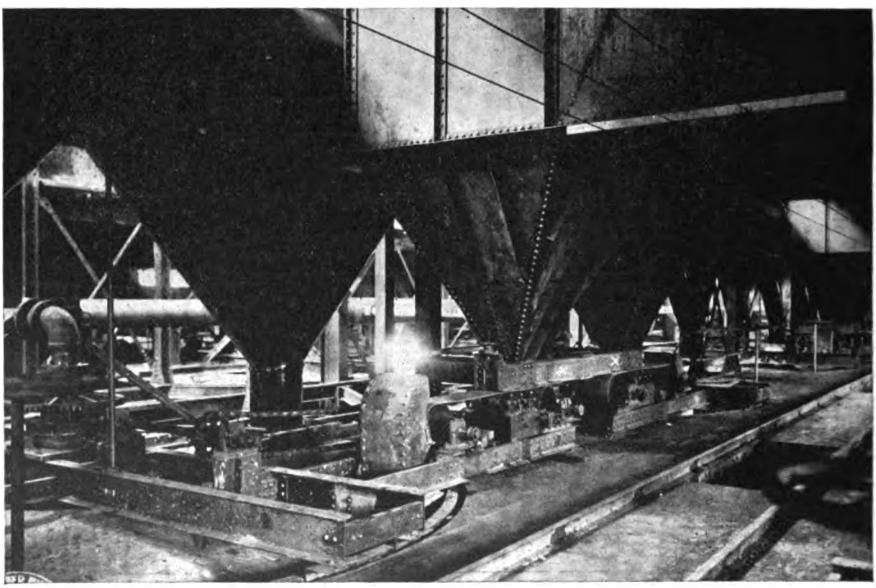

FIG. 7.—FEED FLOOR OF ROASTER SHOWING FEED HOPPERS, FLUE CONNECTIONS AND FEEDING MECHANISM.

to the third or fired floor, the sulphur ignites. The three lower floors are kept hot by the combustion of the sulphur in the tailing.

Four flues run the length of the building, one over each row of seven furnaces. Each furnace has two opposite connections, from the top hearth to the flue. All four flues lead into a balloon flue with a downtake of 45°. The balloon flue enters the 15 by 200-ft. steel stack with a 45° uptake. The stack is unlined. In the bottom of the balloon flue is a 6-in. screw conveyor which delivers the flue dust to the belt-conveyor system which receives the calcine from the furnaces.

The furnaces are air-cooled, the air being furnished by four No. 11 Buffalo blowers each direct-connected to a 50-hp. motor. The air intake is at the top of the furnace shaft and the discharge at the bottom. The hot air does not enter the furnaces, but is delivered to the leaching and solution buildings for heating purposes by a suitable piping system. When it is not needed for this it is discharged into the atmosphere.

Each furnace is equipped with a cylindrical cooler, 30 in. in diameter and 19 ft. long. The cooled calcine enters a mixer or concrete-lined steel cylinder, at the head end of which a very small stream of water is added to settle the dust. The mixer discharges a moist warm calcine to an 18-in. conveyor belt, and by a system of conveyors the calcine is delivered to the leaching building.

The ashes from the fireboxes drop into launders and are sluiced out through the main tailrace.

## Leaching Plant Equipment

The leaching building is 293 by 122 ft. and is of steel and wood construction. It contains 10 redwood tanks each 50 ft. in diameter, and 14 ft. deep. The average charge to a tank is about 1,000 tons of calcine.

The tanks are equipped with an ordinary filter bottom, made of 1¼-in. slats resting on 2 by 4-in. pieces. Above this are two layers of heavy cocoa matting and on top of the matting is a grating, made of 1¾ by 3½-in. material, with 6-in. square spaces. The grating fills with calcine 3½-in. deep and serves to keep the force of the sluicing water from tearing the matting. The acid solutions rot the cocoa matting, but if not disturbed, it will hold its shape and be an efficient filtering medium long after it is too much decomposed to handle. All the steel tank hoops are covered with lead pipes to protect them against leaks of acid solutions.

The tanks are in two rows of five each. A 20-in. conveyor belt travels over each row, and by means of a tripper, the calcine is dumped into a suitable distributor, which spreads it over the tank (Fig. 8).

Each tank has three lead pipes 4 in. in diameter and one 4-in. iron pipe entering at the top. The lead connections are for strong and dilute acid solutions and the iron pipe is for wash water. Above the level of the

leaching tanks an iron storage tank is provided for holding the stock of concentrated acid. Its capacity is about 120 tons of 60° Bé. acid. All concentrated acid, used to raise the acid strength of any solution, is added to the solution as it goes on the charge in the leaching tanks.

There are seven 10-in. sluicing gates in the bottom of each leaching tank, one in the center and six spaced equidistant from each other in a circle about half way between the center and the circumference of the tank. These discharge into launders which connect with the main tailings launder.

"Acimet" valves and lead piping are used throughout for handling dilute and concentrated acid solutions.

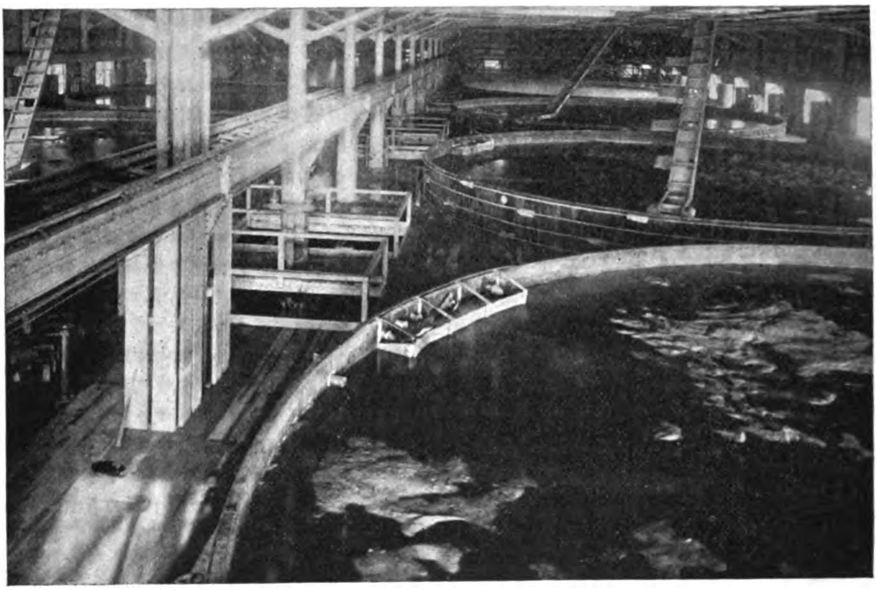

FIG. 8.—VIEW OF LEACHING TANKS AND DISTRIBUTING CHUTES.

The floors of both the leaching and solution buildings are of concrete, and are painted with an asphalt-tar mixture for acid proofing. These floors slope to a gutter which connects to a pump sump and in this way any overflow or leakage of solution is saved and returned to the system.

The solution tank building is a leanto off the leaching building and contains five solution-storage tanks. These are 50 ft. in diameter and 14 ft. deep. Solutions drain from the leaching tanks to the storage tanks and are pumped to the top of the leaching tanks, from the solution tanks, by means of vertical shaft, direct-connected, hard-lead, centrifugal pumps.

## Precipitating Division

The precipitation of the copper and silver is accomplished with scrap iron. The precipitating launders are of concrete, each about 250 ft. long and, having a section of 4 by 8 ft. available for containing iron. Each launder is partitioned off into four sections by concrete walls. Any of the 12 sections may be bypassed for the purpose of cleaning up. In the bottom of the launders is a heavy wood grating, upon which the iron rests, leaving a space about 6 in. under it, for accumulation of any cement copper which may drop off the iron. In the side of each section, at the bottom, are four 6-in. holes, toward which the concrete bottom slopes. These holes discharge into launders which carry the copper to a settling tank. There it is washed and stored and finally excavated with a clam-shell bucket, loaded into standard railroad cars and, at present, shipped to the briquetting plant before blast-furnace treatment. An electrically operated Brown hoist, equipped with a lifting magnet, handles the scrap iron and loads the copper.

## Details of Operation

### *Roasting*

The tailing is subjected to a simple oxidizing roast, no particular care being taken to obtain a large amount of sulphate. The sulphur content of the feed is about 2.2 per cent. and that of the calcine about 0.6 per cent. One-third of the total sulphur in the calcine is in the form of sulphate. When too hot a roast is attempted in order to decrease the total sulphur content, a certain portion of the copper is rendered insoluble in all ordinary acids, with the exception of hydrofluoric.

A pyrometer is inserted over the fourth floor of every furnace, and by means of these, the firemen are able to keep constant control of the temperature. The best results are obtained by keeping the fourth floor at about 500° C. The hottest hearth in the furnace is the third or fired floor, and averages about 535° C.

The water for the calcine cooler system enters at about 40° C., is discharged at 65° C., and is piped to the solution building, where it is used to heat the circulating solutions. It is then pumped back and used again in the coolers. The calcine after passing through the coolers, and after the addition of 1 per cent. moisture, while going through the mixers, has a temperature of about 45° C. During the conveying from the roasters to the leaching tanks, this temperature is lowered to 40° C.

### *Leaching*

The leaching is done by continuous downward percolation, no circulation or upward percolation being used. The percolation rate will vary

from 3 in. per hour with the first solution to as high as 10 in. per hour with the wash water. As nearly as possible, all solutions and wash waters go on the charge at 40° to 50° C.

It requires about one-fourth of the weight of calcine, in weight of solution, to saturate a charge thoroughly.

There are five solution tanks: One for storage of No. 1 solution, one for No. 2 solution, one for copper solution, and two for wash-water.

| | Cu Per Cent. | $H_2SO_4$ Per Cent. | NaCl Per Cent. |
|---|---|---|---|
| No. 1 solution tank | 0.8 | 5.0 | 7.0 |
| No. 2 solution tank | 0.1 | 0.5 | 3.5 |
| Copper-solution tank | 1.9 | 1.0 | 7.0 |
| Wash-water tanks | 0.2 | 1.0 | 1.0 |

After a tank is charged with calcine and leveled, 250 tons of No. 1 solution is added as fast as the charge will absorb it. The drain valve is always open, so, as soon as the solution reaches the bottom of the tank, it commences to drain to the copper-solution storage tank as copper solution. From the copper-solution tank there is only one outlet, which is to the precipitation launders. After traveling through the launders, two-thirds of the solution is returned to the No. 2 solution tank and the balance wasted. This waste is necessary to keep impurities such as iron and aluminum sulphates from building up in the system.

When the No. 1 solution has all been added to the leaching tank, the solution is allowed to drain until none shows on top of the calcine, when 1 per cent. of the weight of the charge, of common salt (NaCl), is spread over the calcine. On top of the salt is then added 100 tons of solution from No. 2 solution tank, but with additional strong acid to bring it to 20 per cent. $H_2SO_4$. Following the 20 per cent. acid solution, 150 tons of No. 2 solution is added, but without additional strong $H_2SO_4$. This scheme gives a zone 4 or 5 ft. in depth of very strong chloridizing solution, traveling down through the charge. There is about 8 per cent. of ferrous and ferric iron in solution, which, with the salt, forms ferric chloride, in itself a very corrosive reagent, even dissolving a considerable amount of unroasted sulphide. This chloridizing action also gives the silver extraction, as without it very little silver is recovered. The 150 tons of No. 2 solution which follows the 20 per cent. acid is for the purpose of washing out silver chloride and dissolved copper which may have been held in the calcine. It carries very little copper or acid, but is fairly high in salt content, and therefore better than a clean water wash. Following the last acid solution, about 300 tons of hot, clean water is added.

The two portions of No. 2 solution, one at 20 per cent. acid, and the other at 0.5 per cent. acid, after percolating through the charge, drain to the No. 1 solution tank.

The wash water, less a quantity sufficient to make up for the discarded solution, drains to the two wash-water tanks. The balance goes to the No. 2 solution tank and adds enough to make up the amount of solution discarded from the precipitating division each day.

### *Precipitating*

The practice here is too old to necessitate much explanation. The main advantages in the practice at this plant over the usual practice are the large launders, which make it possible to put in large and odd-shaped pieces of iron, and the presence of salt in the solutions which prevents the copper from plating on the iron, and makes a soft spongy cement copper which is easily washed off with a hose, leaving the iron clean for more precipitation. It is never necessary to remove the iron for cleaning. The silver is recovered by precipitation on the precipitated copper.

## Results

The resulting cement copper carries about 70 per cent. copper. The following data are taken from the reports for the month of October, 1915. This is a representative month, but it is certain that the results will be improved upon, after longer operation.

| | |
|---|---|
| Sand tailing treated, tons | 70,401.00 |
| Copper in feed, per cent | 0.575 |
| Silver in feed, ounces per ton | 0.45 |
| Copper in tailing, per cent | 0.082 |
| Silver in tailing, ounces per ton | 0.14 |
| Sulphuric acid (60° Bé.), pounds per ton of feed | 64.90 |
| Coal, per cent. of feed | 3.30 |
| Salt, per cent. of feed | 1.52 |
| Iron, pounds per pound of copper | 2.00 |

The plant during this month made an extraction of about 80 per cent. of the copper and 60 per cent. of the silver. This is less than indicated by assays of heads and tailings owing to various plant losses of which the largest is dust from the roasting furnaces amounting to about 4.5 per cent. of the copper in the feed.

*Analyses of Feed and Tailing*

| | Cu, Per Cent. | Ag, Ounces | $SiO_2$, Per Cent. | FeO, Per Cent. | S, Per Cent. | $Al_2O_3$ Per Cent. | CaO, Per Cent. |
|---|---|---|---|---|---|---|---|
| Feed | 0.575 | 0.45 | 81.3 | 3.0 | 2.1 | 9.4 | 0.4 |
| Tailing | 0.082 | 0.14 | 84.7 | 2.4 | 0.4 | 8.7 | 0.4 |

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# Mining and Milling Practice at Santa Gertrudis

BY HUGH ROSE,* PACHUCA, MEXICO

(Arizona Meeting, September, 1916)

THE properties of the company lie within the Pachuca district, State of Hidalgo, Mexico, connected by three railway lines with Mexico City, 55 miles southwest, and by two lines with Vera Cruz, 250 miles southeast.

The ores were formerly divided by sorting into two classes, smelting and milling, the former averaging about 2 oz. gold and 335 oz. silver, the latter 0.12 oz. gold and 23 oz. silver. The smelting ores were sold to custom plants, principally the American Smelting & Refining Co. at Aguascalientes. The milling ores were treated by the patio process at the Guadalupe Hacienda, at Pachuca. This patio was probably the largest in existence at the time and continued in active operation up to March, 1910.

In January, 1910, the mines and patio were sold to English interests, represented by Camp Bird Limited. Two new companies were formed, the Compañia de Santa Gertrudis, S. A., to operate the mines, and the Compañia Beneficiadora de Pachuca, S. A., to build and operate a custom cyanide milling plant. (Fig. 1.)

## Geology

The historical geology of the Pachuca district is described by Fred J. Pope as follows:

1. The most recent sedimentary rocks in the district belong to the Cretaceous period.

2. The Cretaceous rocks were penetrated by large intrusions of andesite (referred to hereafter as primary andesite) which occur as elliptical dome-shaped masses, striking northwest-southeast. These andesite bodies are from 8 to 15 miles long and from 6 to 10 miles wide.

3. A period of erosion during which the contour of the andesite domes was changed.

4. Andesite flows (secondary andesite) which rest unconformably on the primary andesites.

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* General Manager, Compañia de Santa Gertrudis, S. A.

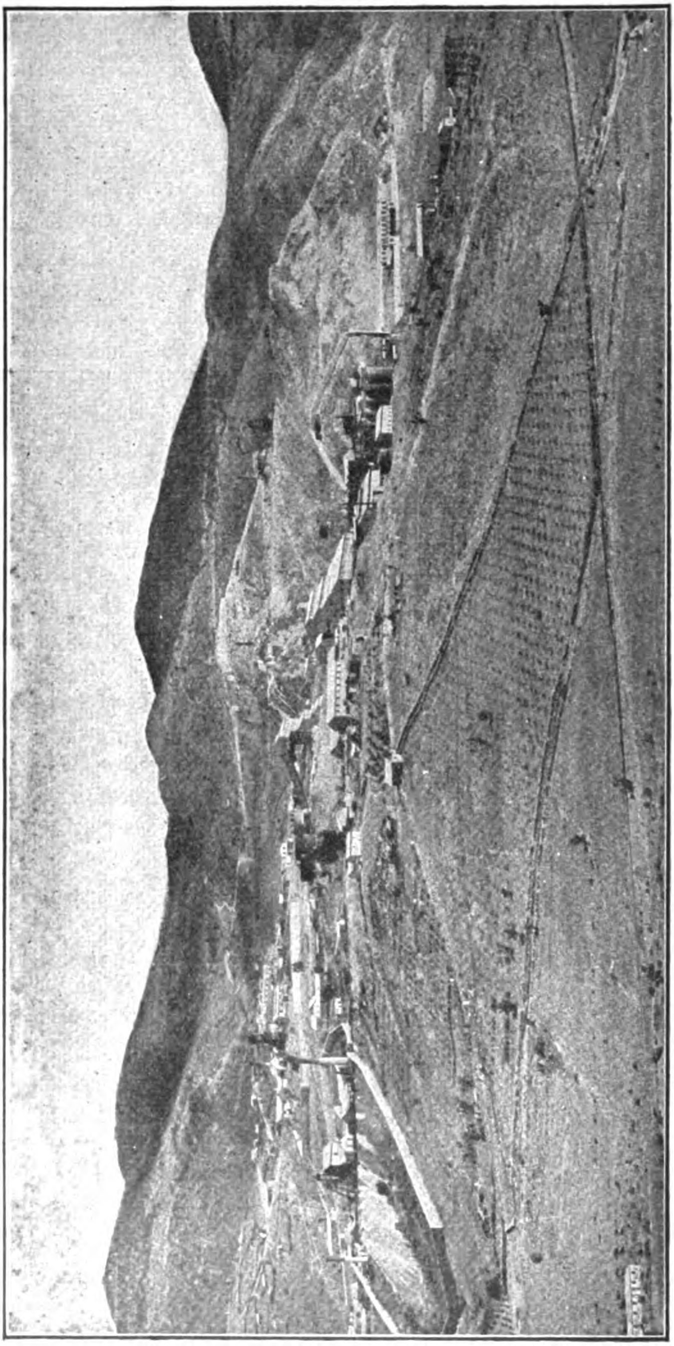

FIG. 1.—SANTA GERTRUDIS MINE AND MILL.

5. A second period of erosion, again exposing areas of primary andesites.

6. A series of rhyolite flows which partially covered the primary and secondary andesites.

7. A third period of erosion which exposed large areas of both the primary and the secondary andesite. During this period there were many intrusions of quartz-porphyry dikes which cut the primary and secondary andesites and the rhyolite. For the most part they do not affect the economic tenor of the veins. In occasional instances, however, the veins are locally impoverished where they cut these dikes. Relatively to the extent of the orebodies, these dikes are very small and practically have no economic bearing.

8. A period of areal faulting, probably due to subsidence and adjustment caused by the large, dome-shaped masses of primary andesite cooling at depth. During this period fissures were formed, rock between the walls was brecciated and there was a heavy silicification, but practically no economic values were brought into the veins. It was essentially a period of fissuring, brecciation and silicification.

9. A second period of faulting with movements relatively much less than during the first period. It was during and immediately subsequent to this secondary faulting that the veins received their economic values.

10. Erosion and minor faults.

## *Country Rock*

In discussing the magmatic variations of the primary andesite, Mr. Pope further states:

The primary andesite in which the Santa Gertrudis vein occurs is not a homogeneous rock. There are local variations due to differentiation in cooling. For the most part the andesite has a slight grayish-green color, is porphyritic, and breaks with a rough surface. This is the prevailing normal type and averages approximately 65 per cent. silica. A more basic variation averages a little over 60 per cent. silica, is porphyritic, dark green in color, breaks with a conchoidal fracture and is very tough. Between these two types there are all gradations and one often graduates imperceptibly into the other. Both occur at all levels to the bottom of the mine, a depth of 2,000 ft.

Operations in the Pachuca district have demonstrated that the tough basic phase of the primary andesite is not favorable to the formation of ore and that all the large orebodies are found in the normal type. This is due to the fact that after the primary faulting and silicification, the basic phase was as tough or tougher than the original rock and thus resisted

the secondary fracturing and subsequent introduction of economic values.[1]

### *Nature and Formation of the Veins*

The veins are not simple fissures filled with silica and economic minerals, but in a general way may be described as crushed zones varying from a few feet up to 50 ft. and in some instances over 100 ft. in width. Usually both foot and hanging walls are fairly well defined, but in many places the dividing line between ore and wall rock can be determined only by systematic sampling.

As a rule the hanging wall is heavy, due to fracturing and to slips or faults parallel to the vein, and in consequence must be carefully supported when mining the ore.

### *Ore Occurrence*

The veins of the district are notable for the size and persistence of the oreshoots.

As a rule, the oreshoots do not outcrop but are discovered at varying depths of one hundred to several hundred feet from the surface, and in the upper levels are shorter and often have a distinctive pitch, generally to the east. Usually, several of these shoots combine in depth to form the main great oreshoot of the mine. In the Santa Gertrudis vein a number of the smaller shoots were thus worked to a depth of from 800 to 900 ft. At about this horizon they joined together forming an orebody which persisted for another 600 to 700 ft. in depth, having a length in the several adjoining properties of about 3,000 ft. and a width averaging 18 ft. The grade of this great oreshoot was not less than $20 per ton.[2]

At a horizon of about 1,800 ft. from the surface the veins, though still strong, become rapidly impoverished and at 2,000 ft. little payable ore is to be found. Below this horizon explorations are being conducted to discover the possible existence of a new zone of mineralization, but such work is not sufficiently advanced to permit any forecast of results.

The ores generally are clean, carrying very small quantities of lead, zinc or iron, and little or no manganese, all of which are unimportant economically. The principal economic mineral is silver sulphide—argentite. The commercial values are confined to gold and silver, which are

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[1] Mr. Pope's report was made after a brief examination undertaken at the time of the purchase of the properties by the Camp Bird interests. Subsequent developments over 5 years have confirmed his findings in general. Several ore-bearing veins occur in basic andesite areas but oreshoots are small and of relatively small economic importance. In the main North Vein, wherein the principal oreshoots are located, the lean parts of the vein within the ore-bearing horizon usually occur where the vein passes through the tough basic phase.

[2] U. S. currency and dry short tons used throughout unless otherwise specified.

present in the proportion of 15 per cent. and 85 per cent. respectively. By weight, for each 1,000 oz. of silver there are usually 5 oz. of gold present.

## Mining Systems and Methods

The mining systems in general use are:

(*a*) Overhead stoping without timbering, with filling.

(*b*) Overhead stoping with timbering and filling.

(*c*) Overhead stoping with shrinkage; with and without subsequent filling.

In mining virgin ore in place, system (*a*) is most generally adopted, being used where the hanging wall is too heavy to employ shrinkage stoping, but sufficiently strong to hold until the stope can be filled. Fig. 2 illustrates the details of this system. Foot-wall drifts (counter-drives) in the harder andesite are kept open after the sill floor of the vein has been stoped out and filled. Crosscuts from the counter drive are run through the vein every 165 ft. and heavily timbered with chutes for drawing the ore from the stopes. These chutes are carried up with cribbed timbers as the stope advances and are divided into two compartments, one for ore and one for manway. Between each two crosscuts a raise is put up to provide entrance for the waste used in filling. Spreading of this waste is done with wheelbarrows.

Throughout the Pachuca district the re-mining of the old areas has become of great importance. The old fills are generally payable and in most cases ore in place is also found on the walls. At first sight it might appear that such territory would be cheaper to mine than virgin ground. But this is not the case, for both the hanging wall and the ore are heavier, making it necessary to adopt system (*b*) with resulting slower extraction of ore at higher cost. Fig. 3 illustrates the details of this system which in the main follow those of system (*a*) with the addition of square-set timbering. As the timbers are soon covered by the waste filling, light sets of 8 in. square material or of equivalent round posts are sufficient. Sets measure 7 ft. 5 in. high by 5 ft. by 5 ft. in the clear.

System (*c*) is applicable only in one small stope in the Santa Gertrudis mine, and hence need not be discussed in detail. In general it is used more commonly in the Real del Monte district than on the Pachuca side, the walls in the former district being much harder and more secure.

Levels were formerly driven 75 to 100 ft. apart. More recently the distance has been increased to 165 ft. (50 m.) vertically. It is questionable whether a greater distance than this could be used economically because of difficulty of holding open the timbered chutes.

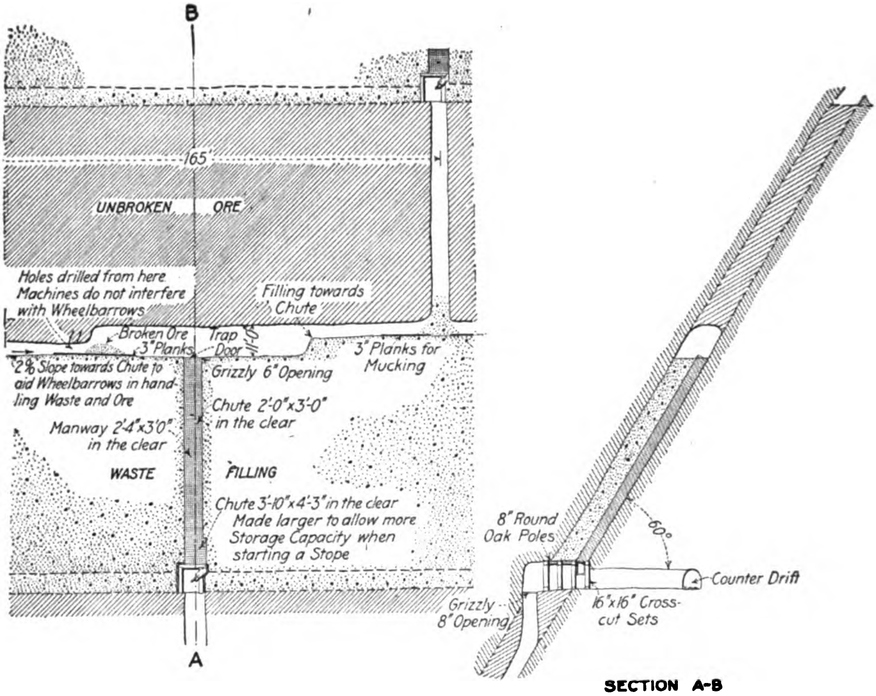



**Fig. 2.—Overhand Stoping without Timbering and with Filling**

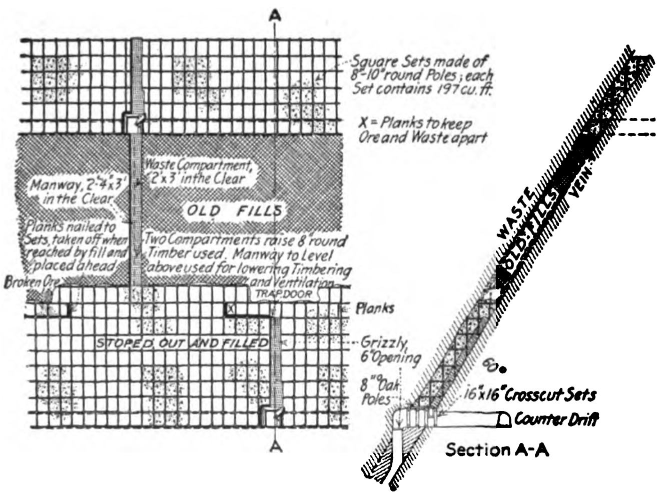



**Fig. 3.—Overhand Stoping with Timbering and Filling.**

*Ore Breaking*

Ore breaking is entirely overhand, using air-hammer drills and 7/8 in. 40 per cent. gelatine powder. Filling for the stopes formerly obtained from caving stations in the hanging wall, is now derived partly from development headings. To provide excess requirements a quarry is being opened on the surface. A central raise in hard foot-wall rock is being put up to tap this supply. This raise will deliver or receive waste at each level.

*Development*

Shaft sinking is generally done against considerable water, 200 to 400 gal. per minute; 3 1/8-in. machine drills are used. Electric firing of rounds has been tried, but with only partial success, due to the breaking timbers which must be carried close to the bottom. Electric time fuses have been used with better success and when intelligently handled are an improvement upon ordinary fuse firing.

Of special interest was the construction of a two-compartment shaft (San Francisco shaft No. 2) adjacent to the existing three-compartment San Francisco shaft No. 1. A pillar of rock, 20 ft. wide, was left to protect the latter shaft, which was kept in operation; sinking and raising were carried on coincidently from several different levels. The shaft raises were timbered in two compartments, a light cage being used in the manway to handle men and tools. A small electric hoist at the foot of the raise served the cage, the hoisting rope being carried over a sheave swung from a movable beam at the top of the raise timbers. Sinking was done by usual methods.

Work was commenced in March, 1912, and the shaft completed to a depth of 1,768 ft. from the collar to the 19th level, with pockets at a number of levels, and put into operation in May, 1913. Steel timbering, bought cut to proper lengths but fabricated locally, was installed throughout. No. 18 galvanized corrugated sheet lagging was used, but only where necessary, being slipped into place on the outside of sets and held secure by small steel clips, riveted to the sheets and fitted over the flange of the I-beam wall plates. The cost of the shaft without pockets was:

| | Per Foot |
|---|---|
| Raising and sinking | $24.62 |
| Timbering (steel) | 20.33 |
| Loading and tramming | 0.65 |
| Hoisting (waste to surface) | 6.23 |
| Explosives | 2.92 |
| Lighting | 0.59 |
| Miscellaneous | 0.27 |
| | $55.61 |

All drifting and crosscutting of importance is done with 3⅛-in. machine drills, using the horizontal bar mounting, two machines to a bar, the muckers cleaning up the breakage while the top part of the round is being drilled.

Raising is entirely accomplished with air-hammer drills.

Winzing is generally by hand and only a relatively small amount of it is done, raising being generally cheaper and more rapid. Air-hammer drills are occasionally used in winzing in hard ground.

*Average Development Costs per Foot*

| | Shaft Sinking (against Water) | Drifting and Crosscutting | Raising | Winzing |
|---|---|---|---|---|
| Labor | $59.25 | $3.75 | $2.50 | $4.75 |
| Timber | 8.45 | 0.10 | 0.35 | 0.10 |
| Explosives | 3.25 | 1.70 | 0.70 | 0.60 |
| Lighting | 0.70 | 0.05 | 0.06 | 0.06 |
| Tracking and piping | 1.00 | 0.60 | 0.12 | |
| Power | 15.60 | 0.50 | 0.50 | 0.55 |
| Maintenance, machine drills | 0.50 | 0.05 | 0.10 | |
| Tools | 0.45 | 0.10 | 0.10 | 0.10 |
| Tool sharpening | 1.50 | 0.60 | 0.60 | 0.10 |
| Hoses, pump fittings, etc | 1.50 | 0.05 | 0.05 | |
| Lubricants | 0.25 | 0.02 | 0.02 | |
| Supervision | 1.05 | 0.30 | 0.30 | 0.30 |
| Totals | $93.50 | $7.82 | $5.40 | $6.56 |

## *Tramming*

Tramming underground is by hand and electric haulage. Four main levels are equipped with electric haulage using 2.5-ton Jeffrey 250-volt trolley locomotives. A battery locomotive was tried without success, the equipment being insufficient to handle the load against varying grades and track curvatures. In the locomotive haulage, a train is made up of 10 U-dump cars each of 20 cu. ft. capacity, equal to about 0.9 dry ton of Santa Gertrudis broken ore. The cars are equipped with Whitney roller-bearing wheels and, in general, are giving good satisfaction. The center of gravity is sufficiently low to require complete filling for easy dumping and to allow the cars to be emptied over the shaft pockets without stopping the train. Waste, when required, for stope filling, is loaded into the empty cars from the waste chutes on the return trip. On hand-tramming levels, the ore is passed, wherever possible, to electric-haulage levels for transportation to shafts. Tracks are 20 in. gage, with 16- and 20-lb. rails.

### *Hoisting*

Men and material are handled in San Francisco shaft No. 1, by two double-deck cages operating in and out of balance. These are served by an Allis Chalmers double-reduction geared electric hoist, using a three-phase, 50-cycle, 440-volt, 300-hp. induction motor with grid type resistance, at a rope speed of 1,000 ft. per minute; also in San Juan shaft, two single-deck cages operating in balance and served by a converted 150-hp. electric hoist, with a rope speed of 700 ft. per minute.

Ore and waste are handled in San Francisco shaft No. 2 by two 5-ton skips operating in or out of balance. These are served by a Fraser and Chalmers single-reduction, herringbone-geared hoist using a three-phase, 50-cycle, 440-volt, 350-hp. induction motor with liquid controller type resistance, at a rope speed of 1,000 ft. per minute, and a capacity of 800 tons per 8-hr. shift; also in San Guillermo shaft, one 3.5-ton skip operating counterbalanced and served by a duplicate of the hoist used on San Francisco No. 1, the rope speed being 1,000 ft. per minute and the capacity 300 tons per 8-hr. shift. This compound shaft is of interest because of its unusual characteristics. It is of old-style masonry from collar to 9th level. It is heavily timbered through the vein to the 13th level and has a twist of 30° between the 13th and 15th levels to bring it parallel to the vein. Between the 15th and 16th levels a 50-ft. radius compound converts the shaft to 60° incline at which it continues to the 20th level, a total vertical and inclined length of 2,090 ft. Except in the heavy part through the vein, little trouble has been experienced in the operation of this shaft, despite its small size in the upper section.

Fig. 4 shows details of construction of San Francisco No. 2 skip, of special interest being the trunnion wheels, the duplicates of which have given no trouble in use in San Guillermo compound shaft. These wheels are also utilized in dumping the skip, as shown in Fig. 5. The loop in the dumping path engages the bottom wheel, thereby preventing the skip from falling back into the shaft in case the safety detaching hook should not hold during an overwind. This dumping path is designed to take care of a skip for vertical or inclined service.

Fig. 6 shows electric-hoist brake-release contact brought into play by being struck by the ascending bale of the skip cage. The hoist brakes, air-operated, are under control of a solenoid connected to the brake weights. When the electric contact is made, the solenoid drops these weights independently of the engineer, thereby applying the brakes.

Plough-steel ropes of flattened-strand construction are used, 1 in. in diameter on men and materials hoists and 1⅛ in. diameter on the ore hoists. Ropes are systematically inspected, following South African requirements. After 2 years' service, new ropes are put on, even though the old ropes show no undue wear.

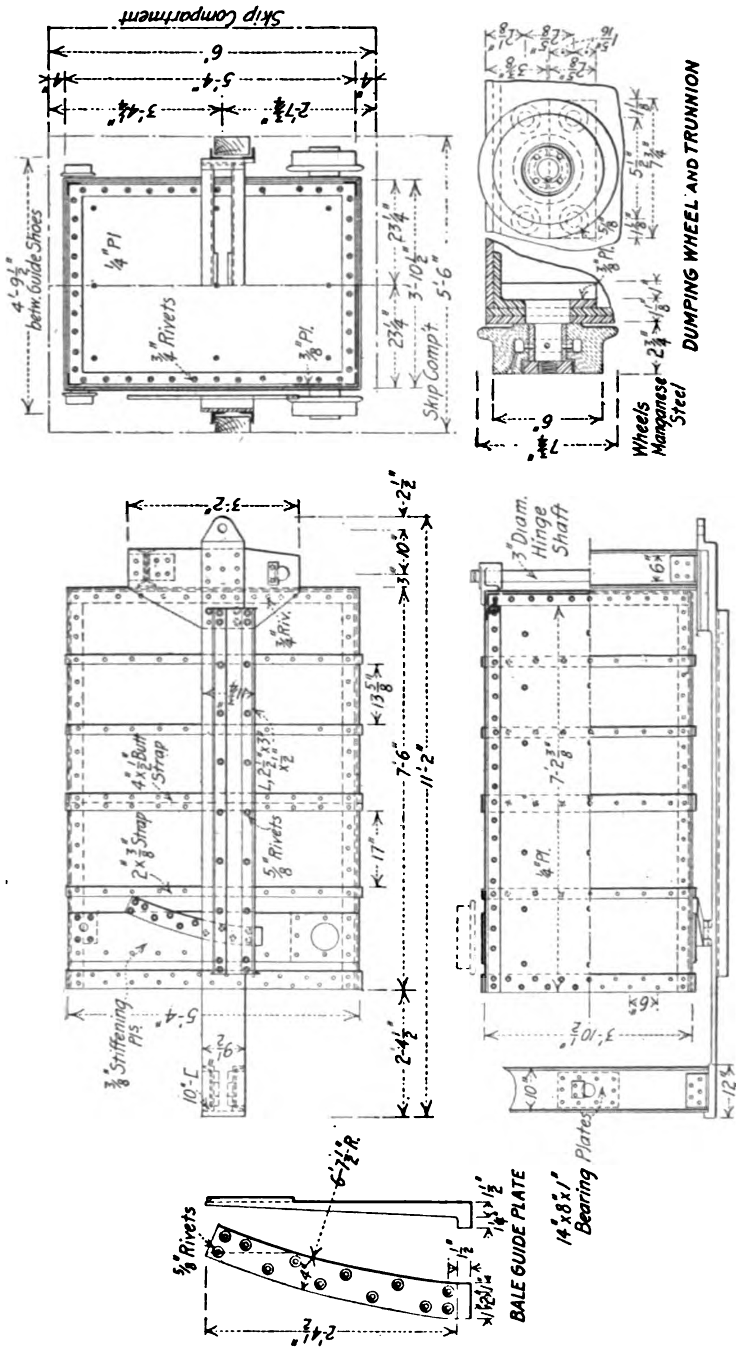

Skip Compartment
4'-9½" betw. Guide Shoes
¼" Pl
¾" Rivets
⅜" Pl.
Skip Compt
DUMPING WHEEL AND TRUNNION
Wheels Manganese Steel
3" Diam. Hinge Shaft
¾" Riv.
4"x½" Butt Strap
2"x⅜" Strap
⅝" Rivets
⅜" Stiffening Pls.
¼" Pl.
14"x8"x1" Bearing Plates
BALE GUIDE PLATE
⅝" Rivets

FIG. 4.—SKIP

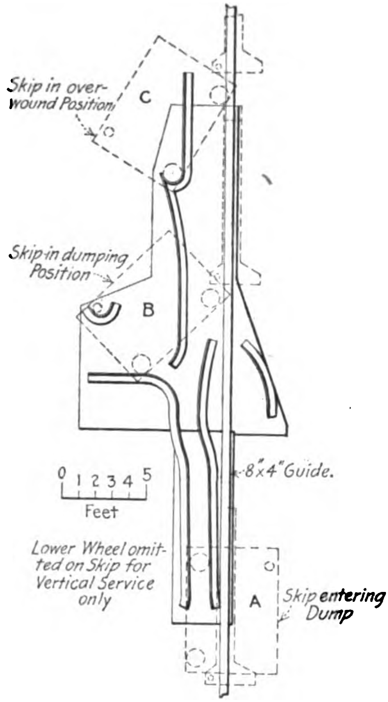



FIG. 5.—DUMPING PATH IN HEADFRAME.

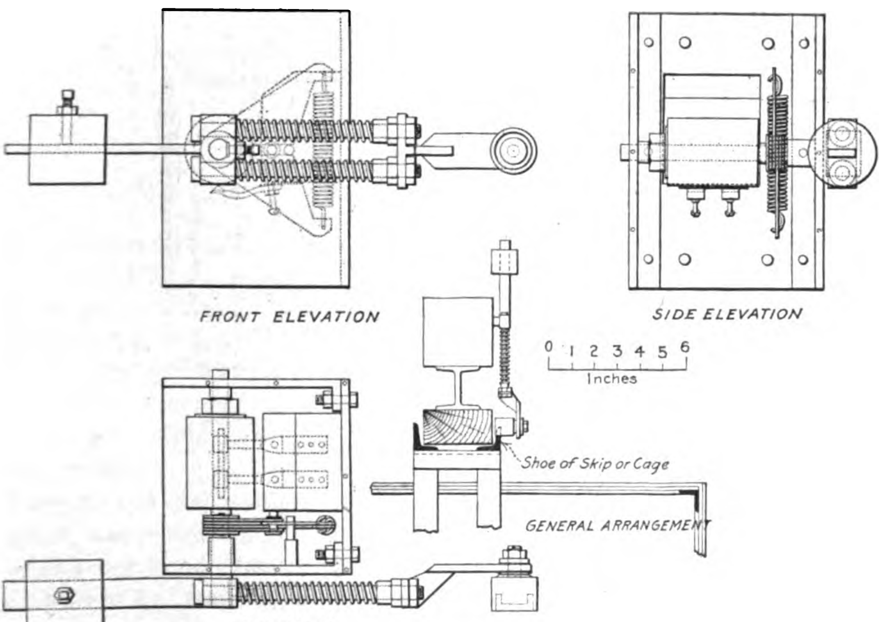



FIG. 6.—ELECTRIC-HOIST BRAKE-RELEASE CONTACT.

Pockets serving the skips are located on all working levels. Fig. 7 shows the characteristic arrangement of the grizzly on top of the pocket and the tracking layout; Fig. 8 shows the standard loading station in general use. Because the ore is free-running and unsually fine, no

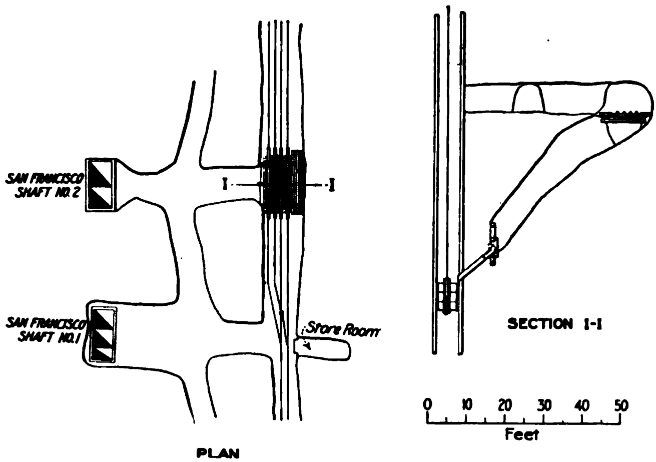



FIG. 7.—STATION AND POCKET LAYOUT.

measuring chutes are required, skip loading being accomplished accurately and with minimum loss of time by manipulation of the gates. With ample ore supply, *i.e.*, pocket fairly full, the loading time is from 15 to 20 sec.

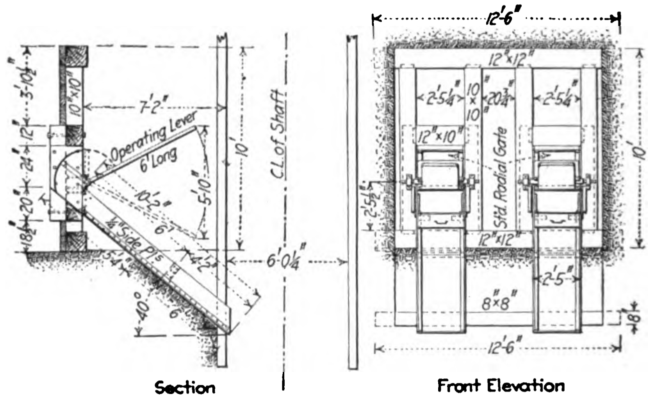



FIG. 8.—SHAFT-LOADING STATION.

Signals from cage or skip tender to hoist engineer are by electric bells, the same system being used by persons calling at any level. Bells are Holtzer Cabot polarized loud-ringing type, 300 ohms resistance, operating on 50- or 60-cycle, 110-volt alternating current. Current is taken from a 1,000-volt power circuit and transformed to 110 volts. The

three-wire system is used for the bells of each shaft compartment. Where telephones are also installed, both bell and telephone wires are in one wire-wrapped, lead-covered cable containing several reserve wires for emergency use. No. 18 size wires are used. Fig. 9 shows the waterproof pull switch used in signaling, which has given satisfactory service.

### *Mining Drainage*

The main pump station is located on the 18th level near the San Guillermo shaft. The 10-in. discharge column of Mannesman drawn-steel tubing with bolted forged-steel flanges and round copper gaskets, is carried horizontally on concrete pillars to the San Francisco shaft No. 1 and thence vertically to the tunnel level, a total length of 2,067

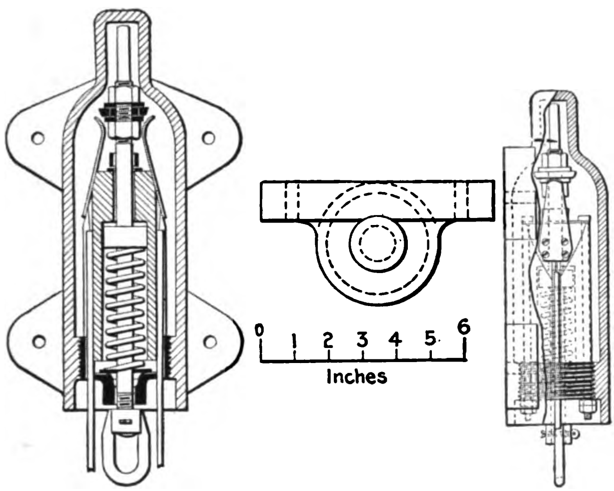



Fig. 9.—Signal Switch.

ft. and a vertical lift of 1,328 ft. The equipment of the station consists of three 500 gal. per minute, horizontal, duplex, double-acting Prescott high-duty pumps, each driven through one reduction of herringbone steel gearing by a three-phase, 50-cycle, 1,040-volt, 250-hp. General Electric induction motor, operating at 480 r.p.m.

No water-hammer difficulties have developed during 3 years' service and the equipment in general has given excellent satisfaction. A traveling crane made the installation easy and assists materially in repair work. The pumps were installed under a guarantee of 85 per cent. efficiency based on power delivered to motor pinion. The motor efficiency was measured at the factory. A series of tests made by the local staff, after the pumps had been in operation for over a year, gave the results shown in Table 1.

The old main pump station was at the 15th level and consisted of two eight-stage Sulzer centrifugal pumps each direct connected to 480-

hp. induction motors of German make, no-load speed, 1,200 r.p.m., capacity 3,600 liters per minute each. These motors were designed for 60-cycle current and at 50 cycles failed to handle the water satisfactorily because of reduction in speed. One of these units is now installed on the 18th level and, being connected to the suction of the pump remaining on the 15th level, the two now serve as a reserve unit of about 1,000 gal. per minute capacity.

Water from the 20th level is pumped to the 18th level by a 1,000 gal. per minute Alberger centrifugal pump, electrically driven. In shaft sinking below the 20th level, No. 10 Cameron sinking pumps are used with air at 80 lb. pressure, and as the lift becomes too great for the air pumps, electric centrifugal station relay pumps are installed.

TABLE 1.—*Results of Test on Pumping Equipment*

| No. of Pump | Water Pumped, Gallons per Minute | Static Head, Feet | Static Head, Pounds | Dynamic Head, Feet | Dynamic Head, Pounds | Theoretical Water Horsepower | Input to Motor, Horsepower | Motor Efficiency from Curve | Input to Pump, Horsepower | Pump Efficiency, Sump to Discharge | Combined Pump and Motor Efficiency |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 480 | 1,328 | 573 | 1,371 | 591 | 160 | 193 | 93.1 | 180 | 89.2 | 83.0 |
| 2 | 481 | 1,328 | 573 | 1,371 | 591 | 161 | 199 | 93.1 | 186 | 86.6 | 80.7 |
| 3 | 469 | 1,328 | 573 | 1,371 | 591 | 157 | 192 | 93.1 | 179 | 88.9 | 81.6 |
| 1 & 2 | 950 | 1,328 | 573 | 1,388 | 599 | 317 | 408 | 93.1 | 380 | 83.6 | 77.8 |
| 1 & 3 | 940 | 1,328 | 573 | 1,388 | 599 | 314 | 400 | 93.1 | 372 | 84.4 | 78.5 |
| 2 & 3 | 939 | 1,328 | 573 | 1,388 | 599 | 314 | 403 | 93.1 | 375 | 83.6 | 77.8 |
| 1, 2 & 3 | 1,421 | 1,328 | 573 | 1,418 | 612 | 475 | 650 | 93.1 | 605 | 78.4 | 73.0 |

Motor data, 480 r.p.m., 250 hp., 1,040 volts.

Pump data, 49.6 r.p.m.; length of stroke, 3 ft.; original diameter of plunger, $4\frac{5}{8}$ in.; diameter of plungers at time of test, $4\frac{9}{16}$ in.; calculated displacement of plungers at $4\frac{5}{8}$ in. diameter, 519.4 gal. per minute; at $4\frac{9}{16}$ in. diameter, 505.5 gal. per minute.

Temperature of water, 87° F.

Weight of water 8.304 lb. per U. S. gallon.

NOTE.—After the tests a small stick was found under one valve in pump 3.

Average gallons per minute pumped, 815; per day, 1,173,600; average pumping cost, including auxiliaries per day, $208.24; per gallon, 0.0177 c.

## *Ventilation*

The natural ventilation of the mine is good, due to the many shaft connections as shown on the longitudinal projection (Fig. 10)—and to communications with eastern adjoining properties, Barron and La Blanca. A motor-driven Sirocco fan is occasionally used temporarily to ventilate some section pending the completion of connections for natural air currents.

### *Sanitation*

Particular attention is paid to keeping all parts of the mine free from refuse or filth. Special sanitary buckets are provided in suitable places

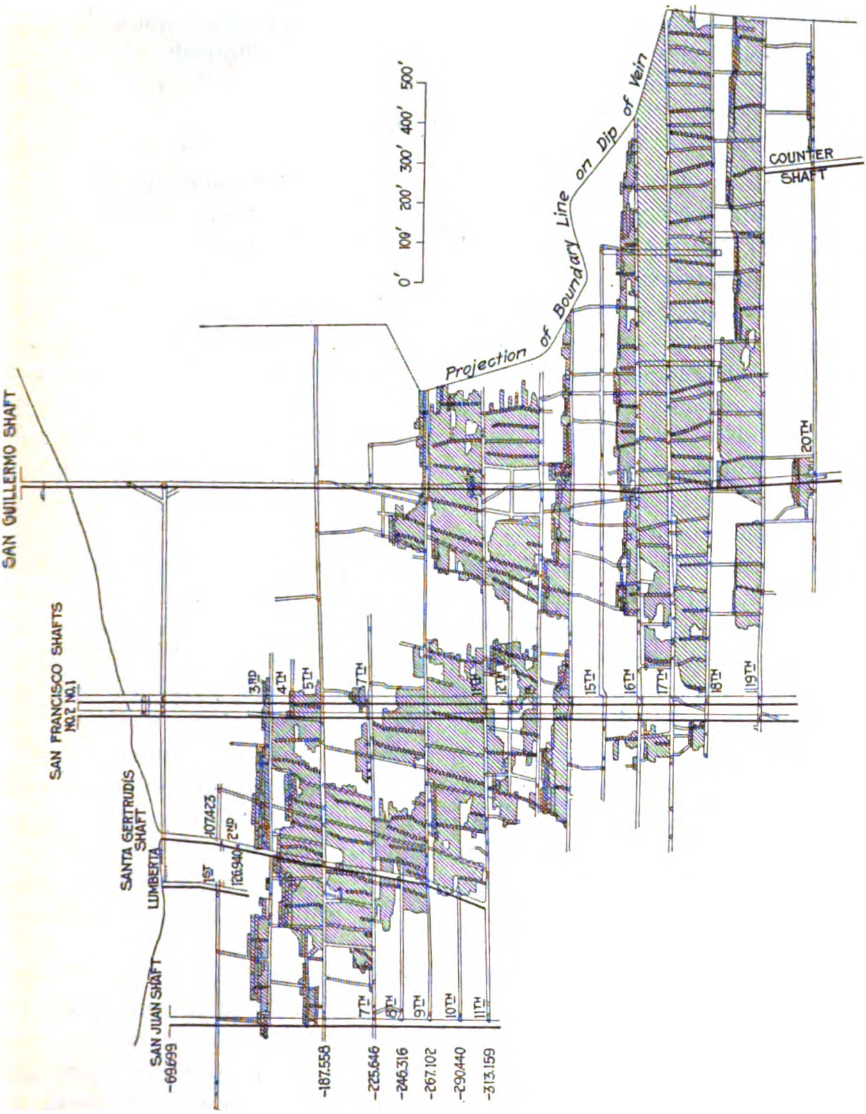



FIG. 10.—VERTICAL PROJECTION. NORTH VEIN.

on each level which the men are required to use and which are systematically taken to the surface for cleansing, lime being applied liberally underground to limit odors.

The main levels, where there is much traffic, are sprinkled with water daily, using a small truck fitted with a barrel having an ordinary sprinkler attachment.

Good drinking water is piped to each shaft station.

A well-ventilated and heated change room is provided, equipped with shower baths of hot and cold water. This facility is not ordinarily found at a mine in Mexico, but its use is to be encouraged.

### *Safety-First Measures*

Within the last two years, an active safety-first campaign has been in hand with very gratifying results. As is well known to all operators in Mexico, the native labor is unusually careless of danger or ordinary precautions against accident. Hence the problem presents exceptional difficulties and requires all the more careful attention to preventive details. Because of the reckless irresponsibility of the individual, coupled with a callousness to accidents which in fairness must be charged to the operators as well as to the men, the accident rate in Mexican mines is abnormally high. And for the reasons stated, this rate will always compare unfavorably with those of well-regulated mines in the United States or Europe. Nevertheless, a great deal can be done to improve this condition.

After securing information from some of the leading American mining companies, experienced in safety-first measures, similar steps were undertaken here. A book of instruction covering accident prevention and first-aid measures was issued in Spanish and English. The shift bosses or "capitanes" of each mine section were made directly responsible for accidents, careful statistics being kept of the work of each boss, with a monthly premium of money to the one having the best record for the month, and a gold watch with suitable engraved inscription for best record for 6 months. An annual contest is held in which various crews compete in rescue and first-aid measures. Bosses and men are drilled in the use of oxygen helmets. A monthly bulletin is printed in Spanish and distributed among the men, in line with similar bulletins issued by American companies.

As of interest to other operators in Mexico, some of the mechanical measures undertaken will be described briefly. All shaft entrances are provided with self-closing gates. After trying out several types, the one selected as most simple and efficient is as illustrated by Fig. 11. The gate is merely swung off center as shown and closes by gravity without undue jar. Fig. 11 also shows the upper gate similarly swung, to protect the individual from falling rocks while standing at the shaft.

Many accidents in Mexican mines are from men carelessly stepping into open chutes or manways in stopes or levels. All chuteways through-

out the mine are now covered with rail grizzlies; manways either have a trapdoor of light corrugated iron or, where this interferes with ventilation, the timber is carried up sufficiently so that the top can be covered, leaving an opening at one end or on the footside for passageway. This opening is kept small enough so that a man cannot slip into it inadvertently. All manway openings on levels have, in addition, a warning red electric light.

Explosives to be used currently are stored underground in suitably located crosscuts, fitted with steel doors set in masonry. Fuse and caps are similarly stored separately. It having been found impossible to

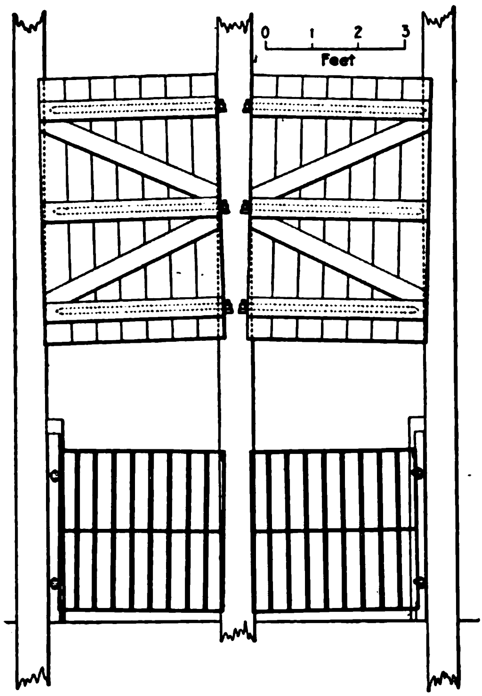



FIG. 11.—SELF-CLOSING GATE USED AT SHAFT ENTRANCES.

prevent the practice of crimping the caps with the teeth, fuse is now issued from these underground stores, cut to desired length and with caps properly crimped on by the storekeeper. The main storage of explosives is located some distance from the property.

Rock falls are a principal cause of many accidents. As elsewhere, the main preventive is constant vigilance on the part of the bosses combined with such coöperation as it is possible to secure from the men. This spirit of coöperation is obtainable only in a limited amount in Mexico, but continued application will bring it forth to a helpful extent.

Following are accident statistics per 10,000 shifts, before and after safety-first measures were undertaken:

*Accidents*

| | Deaths | Requiring Operation | Trivial | Total |
|---|---|---|---|---|
| Before | 0.574 | 1.436 | 12.067 | 14.077 |
| After | 0.334 | 0.669 | 8.358 | 9.361 |

NOTE.—Trivial accidents include numerous bumps, scratches, slight cuts, etc., which do not incapacitate men from working.

## MINING COSTS

For convenience of comparison and of preserving complete records in compact form, the monthly costs are entered upon a tracing form from which blueprint copies are taken. The system requires perhaps less work than where similarly detailed costs are typewritten, whereas the advantage is obvious over any method which does not permit continuous column comparison of items.

Average costs are given below. These do not include recent results which, because of special influences, have been too low to present fairly what should be obtainable under normal conditions:

| | Per Dry Short Ton |
|---|---|
| Development | $0.52 |
| Ore breaking | 1.47 |
| Mine to mine bins | 0.24 |
| Drainage | 0.24 |
| Surface expense | 0.04 |
| General expense | 0.27 |
| Total mining cost | $2.78 |

These results are on the basis of a daily production of 1,000 short tons of ore and may be taken as fairly representative of the Pachuca district. In the adjoining Real del Monte district, lower costs are obtainable because the harder walls permit the adoption of the shrinkage system of stoping with consequent reduction, if not elimination, of expenditure for square-set timbering and filling. At Santa Gertrudis, for each ton of ore extracted, two-thirds of a ton of waste must be supplied and handled for filling, and 17 ft. board measure of timber used. The shrinkage system affords another important advantage in Mexico by providing a reserve of broken ore which may be drawn upon more heavily at times to make up for current deficiencies due to numerous feast days and a relatively low constancy of labor. Another important

factor affecting local mining costs is the light weight of ore in place and filling ore (old fills). The former averages 13.6 cu. ft. per ton; the latter averages 17.2 cu. ft. per ton. At Santa Gertrudis, the moisture in the ore averages 5.0 per cent., and filling ores represent 65 per cent. of the total tonnage mined.

A fair grade of timber is obtainable at $20 per 1,000 ft. board measure. Other supplies, if carefully purchased in quantities, cost about the same as in a mining district in Colorado or Nevada.

Labor efficiency is low, the complicated mining system likewise adversely affecting the duty per man. Including all labor in stopes for drilling, timbering, filling, etc., 1.27 tons of ore per man-shift are broken and placed in the chutes; based upon all underground labor, including bosses and foreman, 0.687 ton of ore per man-shift is delivered to the surface.

Power is purchased at from $45 to $50 per horsepower-year from the Mexican Light & Power Co. whose hydro-electric plant is about 75 miles away.

## Surface Transportation

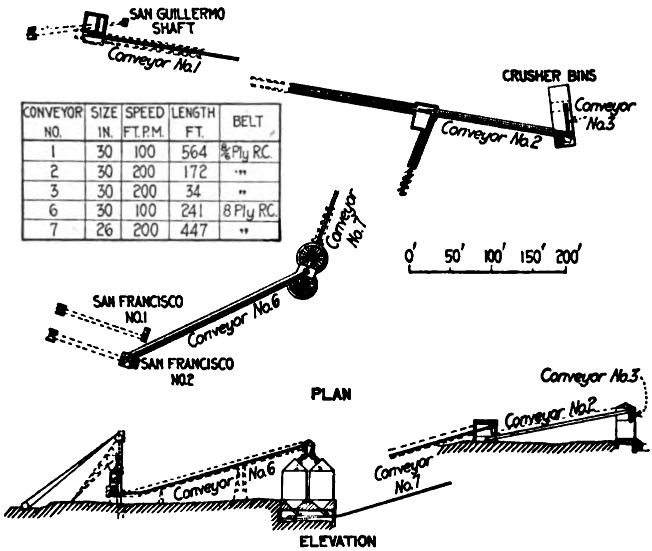
All ore is delivered from the mine by self-dumping skips, discharging into surface ore bins. Thence it is transported to the crushing-plant ore bins by means of belt conveyors. Figs. 12 and 13 show in plan the conveying system from the San Francisco No. 2 and San Guillermo shafts to the crusher bins, and also a cross-section of the line from San Francisco shaft No. 2.

Because of the previously mentioned free-running quality of the ore, plus the unusual percentage of fines which reach the belts first and form a cushion or bed for the larger rocks, the system operates with minimum wear or trouble. Hand-operated sector gates are used throughout, the chute lips being hinged and counterweighted to swing up from the belt when not in use.

On account of re-mining old fills, a large amount of decayed wood comes up with the ore and is picked off at the belts. Careful sorting tests have proved that waste picking cannot be done at a profit. Under close supervision, 3.5 per cent. by weight was picked off, averaging $0.70 per ton at a cost of $0.30 per ton of waste; increasing the waste sorted out to 5 per cent. resulted in an increase of value to $1 at a cost of $0.40 per ton of waste. The actual cost of passing these grades of material through the mill, less value recovered, is calculated to be lower than the sorting cost plus recoverable value lost. Therefore, waste sorting has not been adopted. Sorting high-grade ore for shipment to smelter, because of small quantity available as against high milling efficiency, is not profitable.

This "Mine Bins to Mill Bins" cost has averaged $0.02 per dry short

ton. Belt wear is exceedingly low. Belt No. 2 (Fig. 12) is the only one which has been replaced once against a total of 1,131,708 tons handled to date. Belt No. 1, originally installed and put into operation in June, 1911, has handled 691,999 tons of ore and from all appearances is good for a repetition of this duty. Belts 6 and 7, a later installation, have been in operation since May, 1913, handling a total of 439,709 tons and showing little wear.




| CONVEYOR NO. | SIZE IN. | SPEED FT. P.M. | LENGTH FT. | BELT |
|---|---|---|---|---|
| 1 | 30 | 100 | 564 | 8/6 Ply R.C. |
| 2 | 30 | 200 | 172 | " |
| 3 | 30 | 200 | 34 | " |
| 6 | 30 | 100 | 241 | 8 Ply R.C. |
| 7 | 26 | 200 | 447 | " |

Fig. 12.—Plan Conveying System.
Fig. 13.—Section View.

## Milling

The milling business is conducted by a separate organization on a custom basis. The capacity of the plant is 1,100 tons per day.

Before taking up details for construction and operation, a condensed description of the mill flow scheme (Fig. 14) will be given.

The crushing-plant ore bins of 600 tons capacity feed two No. 6K gyratory crushers, discharging to a double screen 14 ft. long. The oversize from the 4-in. round-hole screen passes by conveyor to the tube-mill rock-storage bin, while the undersize joins the oversize of the 2-in. round-hole screen for crushing in two No. 4K gyratory crushers. The undersize from the 2-in. screen, together with the discharge from the secondary crushers, is delivered to a 22-in. troughed-belt conveyor, equipped with a Merrick weightometer.

The 22-in. conveyor discharges over the first one of three Vezin samplers, a 5 per cent. cut being taken and stored in a sample bin of 15

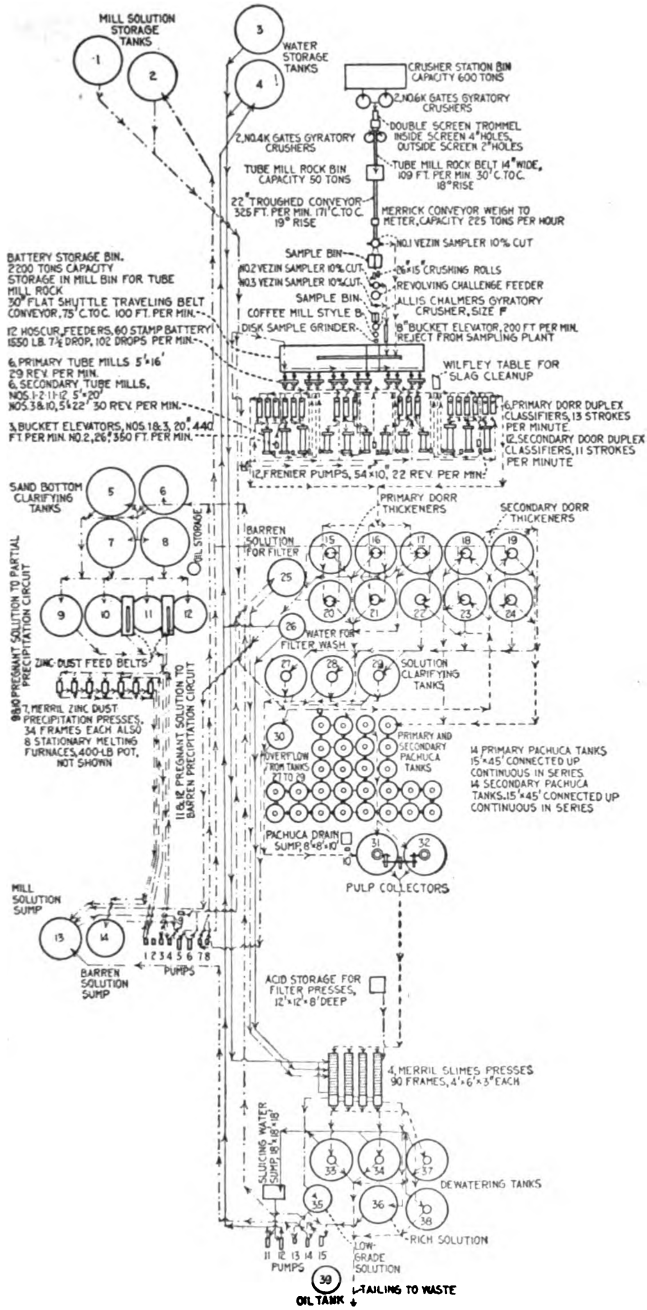



Fig. 14.—Mill Flow Sheet.

tons capacity. The reject flows to a 30-in. shuttle-type flat-belt conveyor for distribution into battery bins of 2,000 tons capacity. The first sample cut is fed from the 15-ton bin through a set of 26 by 15-in. rolls reducing to ¾ in. and finer, and discharging to a second Vezin sampler cutting out 10 per cent. which is delivered by a revolving Challenge feeder to the third Vezin taking a 20 per cent. cut. The sample thus obtained is 1 ton for each 1,000 tons milled and is crushed to ¼ in. in a size F gyratory crusher, thereafter being cut down by Jones riffles and reduced in the usual manner. Rejects from the second and third Vezin samplers and from the quartering floor are returned to the battery bins by an 8-in. elevator.

The ore passes from the battery bins through Hoscur feeders to 60 1,550-lb. stamps arranged in units of 10 stamps each, 20 stamps being driven by a 65-hp. motor, belted to a jack shaft; the stamps make 102 7½-in. drops per minute; 3-mesh and 4-mesh screens are used, the pulp from the batteries flowing through split distributing launders to six primary duplex Dorr classifiers, the sand passing to six 5 by 16-ft. tube mills, the discharges from which are delivered to the launder leading to the secondary classifiers by elevator or by reserve 10 by 54-in. Frenier pumps, one pump to each mill. The slime overflow from the primary classifiers is laundered to eight secondary duplex Dorr classifiers, which feed the sand to four 5 by 20-ft. and two 5 by 22-ft. tube mills, the discharges from which are returned by elevator or Frenier pumps to the classifiers.

The slime overflow from the secondary classifiers passes to six primary Dorr thickening tanks, 35 ft. diameter by 15 ft. deep, wherein the pulp is thickened from 10 to 1 to 1.5 to 1.

The thickened pulp is delivered to a set of 14 primary Pachuca tanks, 15 by 45 ft., operating in series; the discharge from the last tank in the series is by means of an air lift, submerged in the tank itself, and delivering to a launder where a wash of four parts of mill solution is applied, the pulp flowing to four secondary Dorr thickening tanks, 35 by 15 ft., wherein it is again thickened to about 1.75 to 1 and delivered to a set of 14 secondary Pachuca tanks, 15 by 45 ft., also operating in series, the discharge from the last tank of the series flowing to two 35 by 12-ft. storage tanks equipped with mechanical agitators.

These storage tanks feed by gravity at 35 lb. pressure four 90-frame Merrill filter presses, size of leaf 4 ft. by 6 ft., width of frame 3 in. The tailing sluiced from these presses flows to four 35 by 15-ft. Dorr thickening tanks for recovery of water before passing to the tailing storage dams.

The pregnant solution is clarified by passing through four sand filter tanks, 40 ft. diameter by 10 ft. deep.

Precipitation is effected by the Merrill zinc-dust process in two

circuits, partial and barren, in order to economize in zinc dust. Two presses are in use on the barren circuit and three on the partial.

The precipitate is melted in a battery of eight oil-fired No. 400 crucible furnaces arranged on the arc of a circle and served by a radial jib crane, fitted with an air cylinder for raising and lowering the pots.

All pumping of solution and water is concentrated in two pump houses, one located below the filter plant and the other below the precipitation plant. The pumps are of vertical triplex plunger types, gear-driven by motors, and are installed so that one pump is in reserve for two circuits.

An electrically operated inclined tramway runs from the top to the bottom of the mill delivering material to any floor.

## Details of Construction and Operation

Compactness of design was sacrificed in order to secure a gravity flow. Such design was permissible owing to an unusually ample mill site of 17° slope, coupled with a mild climate requiring no housing of tanks. Supervision is made easy because the size of the plants warrants the division into two departments, mill and cyanide, and the tramway is used by the bosses in getting round.

The mill is electrically driven throughout, 50-cycle alternating current being distributed at 440 volts.

### *Crushing, Weighing and Sampling*

The crushing-plant design exhibits no points of special interest. The elimination of elevator returns and the favorable character of the ore make the operation unusually easy and simple. Consumption of steel liners, etc., is almost negligible. The 6K gyratory crushers, driven by 30-hp. motors, take only 15 hp. each. This is again due to the character of the ore and the large percentage of fines which, if the ore had greater abrasive quality, it might pay to screen out beforehand.

An ample supply of mine rock, over 4 in. in size, for use in the tube mills in place of pebbles, is obtained cheaply from the revolving-screen oversize as described above.

The Merrick weightometer is checked weekly against a weighed quantity of ore, the average error being well under 1 per cent. Current weighings are corrected by the weekly factor thus obtained.

Particular attention is paid to securing an accurate sample of the ore delivered to the mill. Ordinarily two classes of ore are sampled separately each day. The rejects from these two are thoroughly mixed and quartered down, to make the mixture sample for the day. The calculated average assay of the two class samples checks very closely the

assay results of the mixture. The usual samples are taken for moisture, which averages about 5.0 per cent.

### *Stamping and Tube Milling*

The design of the stamp battery follows standard lines except in a few minor details. The mortar-box foundation bolts are crossed as shown in

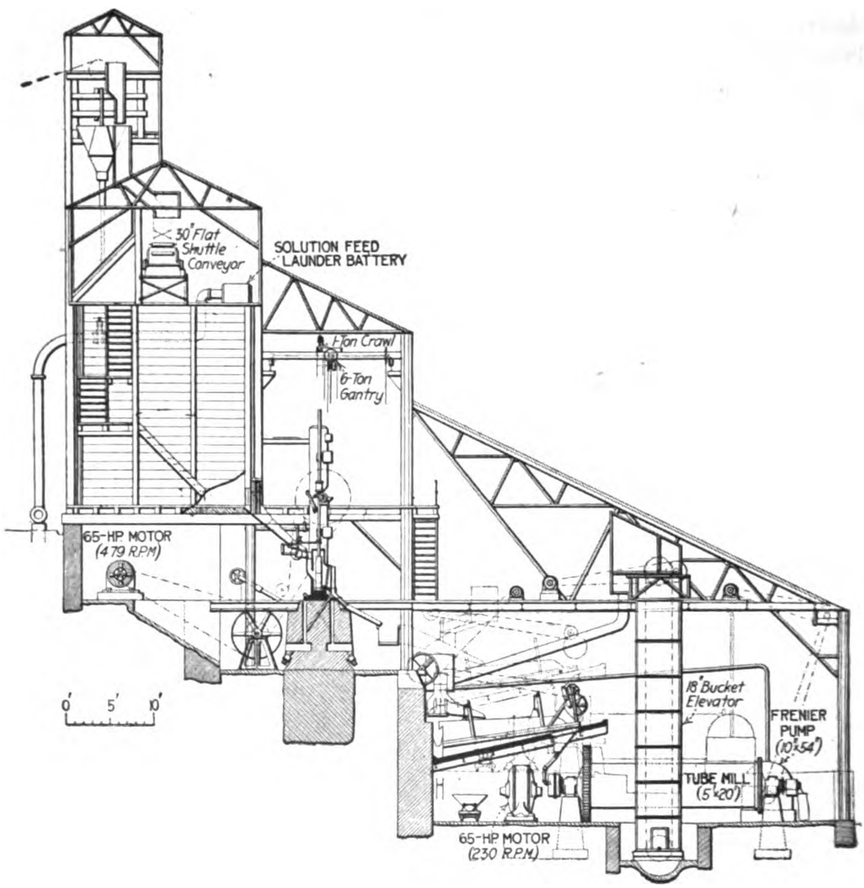



FIG. 15.—STAMP MILL. SECTIONAL VIEW.

Fig. 15, thereby permitting a broken bolt to be removed easily; none has broken thus far. A traveling crane, as well as a crawl, installed with the building, was found very useful in erecting the battery and in making subsequent current repairs.

The stamp duty averages 21.1 tons through 3-mesh and 4-mesh screens. Crushing is in mill solution, 10 parts of solution to 1 of ore. Screen wear is of no importance. Steel wear per ton is: Shoes 0.16

lb., dies 0.08 lb., liners 0.60 lb. Life of shoes, 9¼ in. diameter by 14 in. long, averages 90 days; of dies 97 days; of liners 43 days. Shoes and dies are forged steel. Liners are cast iron made locally. Manganese-steel liners have been tried, but the cost per ton crushed was considerably higher. The 65-hp. motors, each driving 20 stamps, are overloaded about 8 per cent.

A trial was made of introducing the battery solution through nozzles in the back of the mortar box above the dies, but without success. Stationary screens placed between the feeder and the mortar box to take out fines were also tried and abandoned, the benefit being doubtful while requiring more supervision.

Average screen analyses of feed and discharge of the stamp battery, equipped with 3-mesh, No. 32 wire screens, are:

| | Feed, Per Cent. | Discharge, Per Cent. |
|---|---|---|
| Inches | | |
| + 2 | 0.5 | |
| + 1 | 18.4 | |
| + ¾ | 13.8 | |
| + ½ | 8.0 | |
| Mesh | | |
| + 4 | 18.7 | 4.0 |
| + 8 | 11.5 | 16.3 |
| + 10 | 1.8 | 7.5 |
| + 20 | 8.5 | 14.8 |
| + 30 | 3.9 | 8.1 |
| + 40 | 1.4 | 4.4 |
| + 60 | 2.2 | 6.7 |
| + 80 | 2.0 | 6.2 |
| +100 | 0.5 | 2.7 |
| +120 | 0.8 | 3.2 |
| +150 | 0.4 | 1.7 |
| +200 | 0.2 | 2.7 |
| −200 | 6.9 | 21.2 |
| | 99.5 | 99.5 |

Distribution of the pulp from the battery to the primary classifiers by means of split launders could be improved by a mechanical distributor of one of the successful revolving types.

The classifier platforms were built on an incline to save headroom over the tube-mill gears as well as to follow the slant of the classifiers. Returns to the classifiers from the tube mills are by elevators with individual Frenier pumps as reserves. A better design following later practice would be to effect such returns by means of the classifiers themselves.

Tube-mill grinding is done in two stages, using 5-ft.-diameter mills

throughout, the primary series being 16 ft. long and the secondary series 20 ft. and 22 ft. Comparison of this system with single-stage grinding has failed to show conclusive results in its favor, although a slight benefit is apparent. This benefit is, however, insufficient to warrant a repetition of this refinement of design unless in conjunction with water concentration not required with this ore.

Danish flint pebbles were used for a considerable period, but their increasing cost led to the adoption of mine rock entirely to replace the pebbles. The mine rock supply is obtained mechanically in the crushing plant as described and is sent separately over the regular conveyors to a compartment in the battery bin from which it is transferred by chute to the primary tube-mill floor where it is distributed by car. Part of the rock is introduced into the mill through the feeder. As the trunnion opening of the mills is not as large as it should be, rocks over 5 in. in size as well as occasional large boulders, 12 to 15 in., required in the primary mills for efficient grinding, are loaded into the mills through the manholes once a day; 130 lb. of mine rock are required for each ton of ore milled and are credited to the total tonnage treated.

Tests are now in hand using cast-iron balls in place of mine rock. Results thus far obtained indicate a capacity increase of 33 per cent. with finer grinding. Power load shows an increase of 33 per cent., from 65 to 90 hp. per mill. Forged-steel balls ordinarily used for such grinding were not obtainable but it is quite probable that chilled cast-iron or semi-steel balls and liners will prove more economical, taking into account the low cost of locally made castings, 2.5 c. per pound, as against high first cost plus importation expenses of steel balls. Ball wear is 1.7 lb. per ton milled. Tube-mill liners are modified El Oro type, of hard cast iron, the average life being 6 months; cost 2.2 c. per ton of ore milled.

The motor driving each mill through a flexible coupling and one reduction of spur gearing is of high torque induction type, 65-hp., 230 r.p.m. Allis Chalmers make. Its ample design easily carries the overload. But the power factor of this motor is low, 75 per cent., and an improvement would be to use a higher-speed motor with a single reduction of herringbone gearing.

The economical grinding point is taken at 75 per cent. through 200-mesh. Table 2 gives screen tests of feed and discharge of primary and secondary tube mills, using mine rock and at a plant capacity of 1,000 tons per day.

### *Agitating*

An extraction of 55.5 per cent. of the gold and 18.6 per cent. of the silver takes place in the mill before the pulp reaches the Pachuca tank system.

TABLE 2.—*Screen Tests of Tube-Mill Feed and Discharge*

| Mesh | Primary Feed, Per Cent.* | Tube Mill Discharge, Per Cent. | Secondary Feed, Per Cent.† | Tube Mill Discharge, Per Cent. |
|---|---|---|---|---|
| + 4 | 13.9 | 0.2 | 0.8 | 1.6 |
| + 8 | 22.8 | 4.5 | 0.5 | 0.1 |
| + 10 | 8.6 | 2.2 | 0.4 | 0.2 |
| + 20 | 20.6 | 8.5 | 1.6 | 0.2 |
| + 30 | 11.8 | 10.9 | 3.4 | 0.7 |
| + 40 | 4.7 | 4.8 | 3.3 | 1.0 |
| + 60 | 8.0 | 12.1 | 17.5 | 10.8 |
| + 80 | 1.9 | 10.2 | 9.6 | 5.6 |
| +100 | 1.6 | 5.2 | 11.4 | 10.3 |
| +120 | 1.5 | 6.5 | 17.6 | 15.5 |
| +150 | 0.7 | 4.0 | 9.5 | 11.8 |
| +200 | 0.8 | 5.7 | 8.4 | 11.2 |
| −200 | 2.5 | 25.0 | 15.6 | 30.7 |
| | 99.4 | 99.8 | 99.6 | 99.7 |

* Moisture, 35 to 40 per cent. Operating without return, 175 tons of ore pass through tube mill per 24 hr.
† Moisture, 35 to 40 per cent. Operating with return, in closed circuit, 200 tons of ore pass through mill per 24 hr.

*Secondary Classifier Overflow*

(Finished Product of Mill)

| Mesh | Per Cent. |
|---|---|
| +100 | 3.8 |
| +120 | 5.7 |
| +150 | 5.5 |
| +200 | 9.5 |
| −200 | 75.1 |
| | 99.6 |

Cyanide of sodium, either 128 or 120 per cent. as obtainable, in lump or in brick form, is added at the first tank of the primary series, at the rate of 4 g. sodium cyanide for each gram of silver in the ore delivered to the mill. The sodium cyanide consumed is 3.15 lb. per ton, including mechanical loss. The strength of solution at the beginning of agitation is 0.55 per cent. KCN; of the mill solution, 0.4 per cent.

Protective alkalinity is maintained at about 0.75 per cent.; the lime, fed dry into the ore at the crushing plant, is low-grade, averaging about 65 per cent. available, the consumption being 20 lb. per ton. Arrangements are in hand to improve the method of feeding by emulsifying the lime, adding it either to the primary tube mills or to the Pachuca tanks.

Crude litharge, between 85 and 90 per cent. PbO, adopted in place of lead acetate as both cheaper and more efficient, is ground in a small

tube mill, 24 in. diameter by 37 in. long, discharging into the first Pachuca tank at the rate of 0.6 lb. per ton of ore.

The best results are obtained with 72 hr. agitation although 60 hr. give within 2 per cent. as high extraction. The air pressure is 27 lb., 75 cu. ft. per minute being required for each tank.

No difficulty has been experienced with the series operation of the Pachuca tanks. Connections between tanks are made by 10-in. horizontal pipes, located 3 ft. from the top, the joint between two abutting pipes being made leak proof by a wrapping of tarred canvas. This joint takes up the tank vibration and effects an easy connection.

To avoid undue accumulation of slime on the inside of the tanks, they are emptied and sluiced down once a month. This operation requires about 3 hr. per tank, using a 3-in. Traylor slime pump for the return.

Screen tests of the inflow and outflow of the system are practically identical, showing that there is no segregation or short-circuiting of sand or slime.

### *Filtering*

This step in the process is fairly difficult, due to fine grinding of ore containing a considerable amount of colloidal matter. Also, the dissolved values in the pulp to the filters are high, 3.5 oz. silver per ton of solution. After extended working-scale tests of several types of vacuum and pressure filters, Merrill presses were adopted, using the center system of filling.

A press cycle occupies 75 min. made up of:

| | Min. |
|---|---|
| Charging | 26 |
| Barren solution wash | 13 |
| Water wash | 13 |
| Sluicing | 23 |
| | 75 |

Bristol recording pressure gages are attached to the filling pipe of each press, the cycle curves from the chart giving an excellent check upon the care taken by the attendants in operating the presses. The sluice valve bar and the filling valves are electrically connected so that both cannot be coincidently opened without ringing an alarm bell. The cakes average 1¼ in. in thickness, thereby leaving ½ in. space in the center of the 3-in. frame for entrance of washes. Dry pulp handled per cycle, 14.5 tons.

Sluicing water is at 90 lb. pressure and five parts are required to clean out a press. The bulk of this water is recovered in dewatering tanks as explained heretofore.

No. 6 cotton duck, 72 in. wide, is used regularly, after extensive trial of several other weights. A set of cloths lasts about 2,500 cycles. Acid

washing to remove lime is done in the press every 10 days, using a 0.75 per cent. sulphuric acid solution.

Much care is taken with the nozzles of the sluice bar. The bar of each press is taken out and tested every 48 hr., any defective nozzles being replaced. Nozzles are ordinary ½-in. cast-iron plugs, drilled 3/16-in. hole; these last practically as long as special steel nozzles and are very much cheaper.

The unwashed values in the press discharge average a trace of gold and 0.08 oz. silver, showing an efficiency of 97.7 per cent.

### *Clarifying Solutions*

All solutions to be precipitated are first passed through ordinary sand filter tanks. The addition of about 40 per cent. by volume of saw-dust to the sand considerably improves the clarifying efficiency of the filtering medium and at the same time reduces the frequency of slime skimming and sand cleaning. The latter operation is effected by shoveling the sand from the tank to a launder leading to a small trommel where the sand is washed free of slime, with small loss, and returned for re-use.

In this way, 4,500 tons of solution are filtered daily at a cost of 0.28 c. per ton of solution. Possibly Merrill clarifying presses would be an improvement, although no comparative figures are available. At any rate, the existing equipment does the work efficiently and cheaply, with little supervision required.

### *Precipitating*

The Merrill zinc-dust process of precipitation has in every way justified its adoption. Its efficiency, safety, cleanliness, etc., compared to the zinc-shaving method are too widely known to need further elucidation here. The ease and rapidity with which a cleanup can be made gives it a further special advantage in a good-sized mill treating silver ores with the consequent considerable production of precipitate. Formerly the process enjoyed the further advantage of a lower price for zinc dust than for zinc shavings; the difference now is practically negligible.

The solution from the sand filters averages 0.015 oz. gold and 3.0 oz. silver. Two circuits are maintained, one precipitating to barren solution, sufficient for the filter press washing, and the other circuit to an effluent containing from 0.9 to 1.2 oz. silver per ton. To accomplish this a slight excess of zinc dust is fed to the barren circuit in the proportion of 1.1. oz. zinc dust to 1.0 oz. fine bullion; to the partial circuit 0.8 to 1.0. The average zinc consumption of the plant is 1.0 oz to 1.0 of fine bullion.

Cleanups made at the middle and end of month and sometimes oftener, provide an accurate check on current work.

### *Melting*

The only drying given the precipitates is by blowing air through the presses at 25 lb. pressure for 1 hr., reducing the contained moisture to about 30 per cent. The precipitate is cleaned from the presses into rectangular, steel cars, 5 ft. 10 in. by 6 ft. 4 in. by 1 ft. deep, in which it is weighed. Based upon the calculated dry weight of the precipitate, flux consisting of 1½ per cent. each of borax and bottle glass, is added on top without mixing, and the charge is then shoveled from the car into No. 400 crucibles. To permit the introduction of a high column of precipitate, a discarded pot with the bottom out is temporarily fitted into the crucible, being removed after the charge has melted down. This procedure reduces dust losses and accelerates charging and melting.

The eight furnaces (Fig. 16) are oil-fired, built flush with the floor of the melting room, on the arc of a circle with fire pit at rear. This arrangement has proved very satisfactory and is to be recommended as convenient and labor saving.

The crucible, upon being lifted from the furnace by tongs attached to ½-in. wire cable raised or lowered by air cylinder on the post of the jib crane, is swung around to one side to the pouring carriage containing the bullion molds. The slag is first skimmed off, using sand to assist in this operation. As the bars are poured, sticks of wood are laid on top and, igniting, prevent too rapid cooling of the center of the bar with the consequent subsidence and holing.

The precipitate averages 85 per cent. fine bullion; the Doré bars weigh 1,000 oz. and assay 5.0 gold and 940 silver fine. The No. 400 crucibles contain an average of seven bars per melt with a maximum of 10 bars. The average life of a crucible is 10 melts or 70 bars.

With bullion of this fineness, no difficulty with matte formation is experienced. The bars are carefully cleaned and all chips and corners removed before sampling and weighing. Drill samples are taken, one on the top of the bar, one-third the distance along the diagonal toward the center, and one on the bottom of the bar at one corner; each hole is drilled halfway through the bar.

The bullion was formerly refined by a French concern in Mexico City, the fine gold being sold to the Mexican Government and the fine silver shipped for sale in New York or London. The refinery has been closed down since the outbreak of the war in Europe, so that the Doré bars are now refined and sold in New York or London.

Melting the precipitate in an electric furnace, built after the Alaska-Treadwell and Lluvia de Oro design, is under trial. Alternating current at 120 volts is used, the input being about 600 amp. working at full heat. Trouble has been experienced with dusting and with burning a hole through the bottom of the furnace; a renewable iron plug (Fig. 17) may

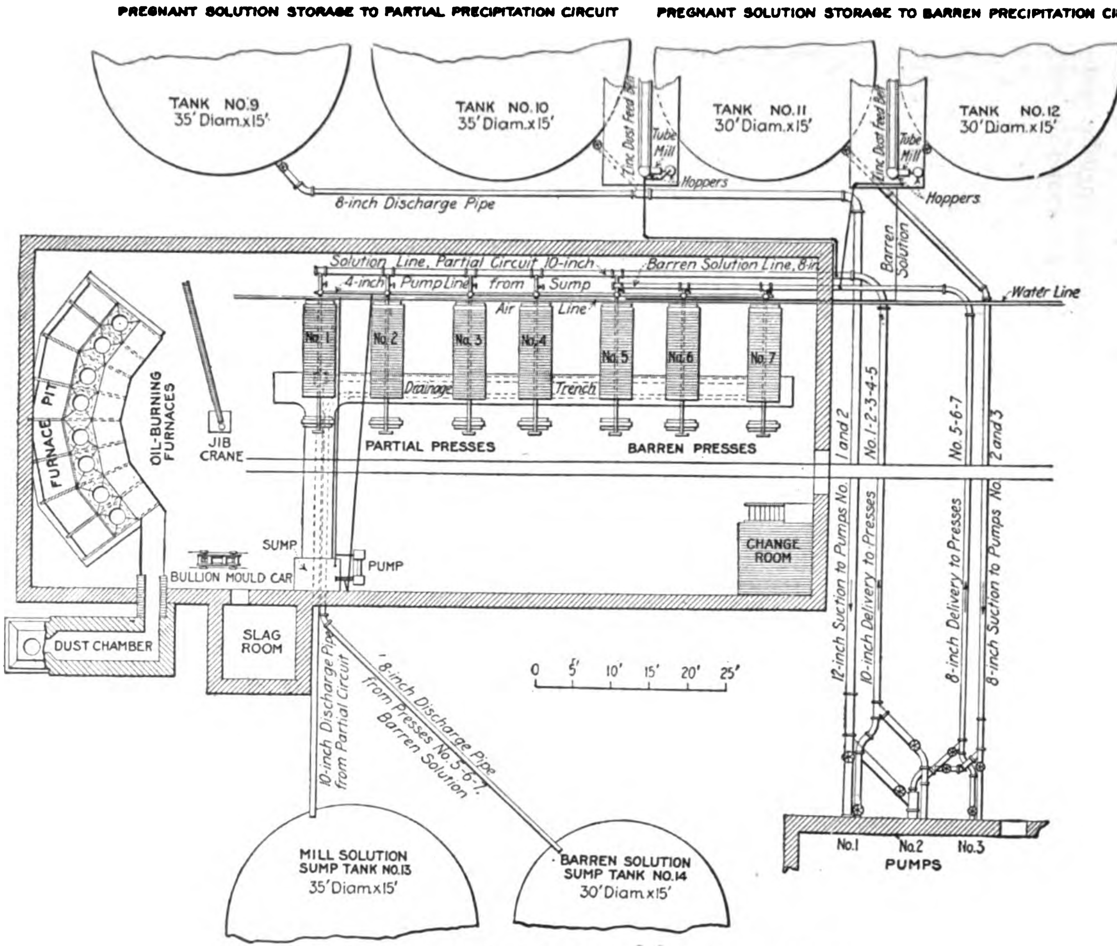



FIG. 16.—PLAN OF PRECIPITATING AND MELTING ROOM.

provide a convenient remedy for the latter. The dusting is probably due to the precipitate being wet, the intense heat at the electrode setting up a center draft of steam and air. Either the precipitate should be dried or some dust-collecting apparatus attached to the furnace top. The advantages of this system over the oil-fired crucible method of melting are rapidity of melting and discontinuance of the use of crucibles and oil, both of which require a long distance haul; no data are available yet as to the comparative economy.

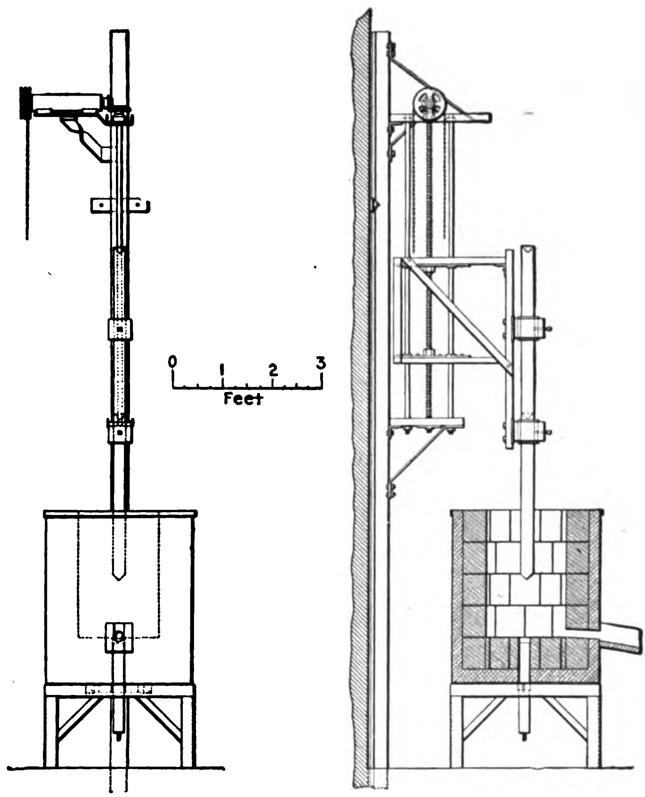



FIG. 17.—ELECTRIC MELTING FURNACE.

*Tailing Disposal*

As previously described, the tailing sluiced from the Merrill filter presses passes through four Dorr thickening tanks for recovery of water and is then automatically sampled before flowing to the storage basins. These basins or ponds, four in number, are located on the nearest flat land 2 miles away. The ditch thereto was dug U-shaped, 2½ ft. deep with a 3 per cent. grade; 1½ per cent. grade would suffice.

The walls of the storage basins are built up as required of tailing mud,

one basin receiving the entire inflow while the others are being prepared. Evaporation and particularly absorption of the solution are high, the decanted effluent representing only 30 per cent. of the solution inflow.

This solution contains an average of a trace of gold and 0.12 oz. silver, representing filter losses and further dissolved values. A small precipitation plant is being arranged to recover this escaping fraction.

### *Construction Costs*

On the basis of 1,100 tons daily capacity the total cost of the plant was $792 per ton capacity. This covered first-class construction methods throughout, designed for long life with minimum maintenance and renewals. A comparatively excessive amount of equipment was required due to fine grinding and especially to length of treatment necessary for a silver ore, resulting in unusually large tankage. On the other hand, the all-sliming design without concentration simplified operations and eliminated the installation of concentration equipment.

## Experimental Work

### *Aluminum Dust Precipitation*[3]

Upon the publication of results obtained at Nipissing with desulphurization and aluminum-dust precipitation, similar experiments were undertaken. Desulphurization, as was to be expected, gave no beneficial results on Santa Gertrudis ore, which contains no tellurides, arsenides, etc. On the other hand, aluminum-dust precipitation showed a considerable improvement over the zinc-dust method and its adoption has only been delayed by war conditions practically having put aluminum dust off the market. Comparison of the two methods was based upon normal prices of aluminum and zinc dust of 30 c. and 7 c. per pound, respectively.

As practically all the existing equipment can be used very conveniently in the aluminum-dust process, the problem as worked out depends for commercial success upon an economical method of converting the lime in solution, added during milling for neutralizing and settling, into calcium carbonate with the formation of the necessary caustic soda as a product of this reaction, all prior to precipitation by aluminum-dust. Any lime present in solution during precipitation will be converted to calcium aluminate, lowering the grade of the precipitate and causing difficulty in melting. The conversion and elimination of lime can be

[3] Tests conducted at mill by A. W. Hahn.

accomplished by the addition of carbonate or bicarbonate of soda, as follows:[4]

$$Ca(OH)_2 + Na_2CO_3 = CaCO_3 + 2NaOH$$
$$Ca(OH)_2 + NaHCO_3 = CaCO_3 + NaOH + H_2O$$

In place of commercial carbonate or bicarbonate of soda, laboratory tests showed the probable successful use of an impure local product known as "Tequesquite," obtained from the dried-up bed of Lake Texcoco and containing approximately 56 per cent. carbonate and bicarbonate of soda in about equal quantities. In practice, the "Tequesquite" solution would be added to the pulp flowing to the collecting tanks above the Merrill filter presses.

In experimental work, the aluminum-dust precipitate yields a bullion 975 fine. The consumption of aluminum dust was about one-third that of zinc dust, or 0.34 oz. to 1.0 oz. fine bullion.

The slightly increased cost per ton of ore of aluminum dust is more than offset by decreased chemical consumption of cyanide, the net saving being calculated at $0.15 per ton of ore.

### *Electrolytic Regeneration of Cyanide*[5]

Prior to the favorable conclusions reached in the aluminum-dust tests, a lengthy investigation was conducted in the regeneration of cyanide in the mill solution by electrolysis.

Anything like a comprehensive description of this interesting work cannot be taken up here and only the important features will be outlined.

The principal difficulty presented was in finding an anode of high conductivity that would not dissolve or disintegrate. Graphite, lead, plumbago, carbon, charcoal, magnetite, peroxidized lead, were tried without much success. An alloy of lead with 6 to 9 per cent. antimony was finally selected as fairly satisfactorily filling the requirements. The following conclusions were then worked out:

1. The solution to be treated by electrolysis must be barren of gold and silver to avoid their precipitation. The question of enlarging the

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[4] For chemistry and details of aluminum-dust process refer to:

E. M. Hamilton: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal*, 95, No. 19, p. 935 (May 10, 1913).

E. M. Hamilton: *Engineering and Mining Journal*, vol. 99, No. 13, p. 568 (March 27, 1915).

S. F. Kirkpatrick: Aluminum Precipitation at Deloro, Ontario, *Engineering and Mining Journal*, vol. 95, No. 26, p. 1277 (June 28, 1913).

James Johnston: The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, *Trans*, vol. 48, p. 3 (1914).

G. H. Clevenger: The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, *Trans.*, vol. 49, p. 156 (1914).

[5] Tests conducted at mill by A. W. Hahn.

scope of the process to include electrolytic precipitation of gold and silver was also investigated separately, as hereinafter discussed.

2. The regeneration of cyanide takes place at the cathode, with a destruction of cyanide at the anode which may be principally overcome by maintaining a high protective alkalinity at the anode.

3. A porous diaphragm such as canvas separating the anode and cathode assists in maintaining an alkaline anolyte, circulation of anolyte and catholyte being accomplished individually.

4. Power consumption is affected by resistance of electrodes, electrolyte and diaphragm, by temperature and circulation rate of electrolyte, and by spacing of electrodes.

5. The voltage drop per cell, at a current density of 15 amp. per square foot of anode surface, treating barren solution containing 0.5 per cent. total KCN and 0.115 per cent. CaO, with canvas diaphragm and electrodes spaced 1½ in. apart, is 6 volts at 70° F. Under these conditions the solution flow should be at least 0.25 gal. per square foot of anode surface.

6. The deposit on the cathode analyzed 71 per cent. zinc, with impurities of copper, iron, arsenic and organic matter. The recovery of this zinc might be turned to some commercial advantage.

7. The reaction representing the regeneration of cyanide is presumably as follows:

$$Na_2Zn(CN)_4 + Ca(OH)_2 + \text{Current} = 2NaCN + Ca(CN)_2 + Zn + H_2O + O.$$

8. The net reduction of cost was estimated at $0.12 per ton of ore. The equipment outlay, however, proved to be several times greater than that required to convert the precipitation plant to the use of aluminum dust and as the latter process at the same time indicated a somewhat larger cost reduction without further complexities of plant, its adoption in place of regeneration was decided upon.

### *Electrolytic Precipitation and Regeneration*

Following the experience gained in the experiments with regeneration above described, the next step was to endeavor to combine with it the electrolytic precipitation of gold and silver from solution.

Difficulty was encountered in obtaining a coherent cathode deposit. It was found uneconomical to attempt a complete precipitation, the limit of commercial work being about 1.3 oz. silver remaining in solution. Plant cost would be high. Altogether the proposition did not present sufficiently attractive possibilities to warrant following it up very thoroughly. With the perfection of the diaphragm cell it is probable that both precipitation and regeneration electrolytically could be worked out to a fair success.

### *Electrolytic Refining of Precipitate*

A preliminary investigation has been made of electrolytic refining of precipitate from the present zinc-dust installation. The chief difficulties to be overcome are resistance of the precipitate to the current, purification of the electrolyte, high acid consumption due to impurities, and refining and melting of the gold slime.

### *Flotation Concentration*

As previously stated, the ore is clean with only small quantities of base metals. A representative analysis is: Au, 0.067 oz.; Ag, 12.0 oz.; Pb, 0.10 per cent.; Cu, 0.20; Zn, 0.15; Fe, 4.40; $SiO_2$, 86.67; S, 1.0 per cent.

Most of the silver is present as argentite, $Ag_2S$ – 0.05 per cent. It is evident, therefore, that little benefit in recovery is to be expected by gravity concentration. At the time of building the plant complete experiments were conducted on a working scale, the recovery by concentration representing from 8 to 10 per cent. of the value of the ore. This did not justify the installation of gravity concentrating equipment plus added cost of operation as against direct cyaniding.

The growing success of flotation concentration in the United States led naturally to local investigation of the problem, which though still in a laboratory stage, already shows encouraging results. If the process is worked out to a commercial success, a detailed description of results should be of interest. Meanwhile a summary of preliminary conclusions is given:

1. A large variety of vegetal and mineral oils have been tested, the best results thus far having been obtained with one-third pine oil or wood creosote and two-thirds Mexican gas oil, such as is used locally in the oil-fired melting furnaces. The pine oils and wood creosote tried are imported, but doubtless satisfactory substitutes can be made near at hand. Several coal tars and a number of Mexican oil field products remain to be tested.

2. The addition of acid shows no particular benefit. A slight alkalinity not exceeding 0.03 lb. lime per ton solution is satisfactory.

3. Heating the solution does not materially improve results.

4. The flotation tailing is cyanided readily, the presence of the small quantity of oil having no deleterious effect.

5. The raw flotation concentrate cyanides with no more difficulty than a gravity concentrate; the cyanide consumption maintains more or less the same ratio as in the crude ore, viz.: 4 g. sodium cyanide per gram of silver; a 98 per cent. extraction of gold and silver in the concentrate is obtainable.

6. In the small laboratory machines without cleaning cells, a recovery

by flotation of 65 to 70 per cent. can be made at a ratio of concentration of between 80 and 90 to 1.

7. In the 100-ton trial plant now under construction, equipped with middling returns and cleaning cell, it is expected that these results can be improved.

8. It is possible, though not certain, that flotation, even with this clean ore, may replace cyanidation to a large extent.

## Power

The Pachuca district is supplied with electric power generated by several hydro-electric plants owned by three power-distributing companies, practically under one control. The main plant, owned by the Mexican Light & Power Co. and located at Necaxa, Puebla, about 75 miles by transmission line to the east, also furnishes energy for Mexico City and El Oro. The 50-cycle alternating current is transmitted at 85,000 volts to the Pachuca substation where it is stepped down to 20,000 and 6,000 volts for local distribution. The power is sold at the substation of the purchaser on the basis of integrating wattmeter readings, no peak or installation loads being considered. A power factor of at least 80 to 85 per cent. is required of the purchaser and prices vary from $42.50 to $50 per horsepower-year, depending on amount consumed.

Although most of the mines are wet, there are no reserve steam plants in the district, with the exception of a relatively few old steam hoists that could be used for baling in an emergency. Nevertheless, the several widely separated hydro-electric plants with individual transmission lines provide a very reliable power supply.

## General Remarks

As can be imagined, operating conditions in Mexico during the past several years have been subjected to many unusual difficulties, although the Pachuca district perhaps has been as fortunate as any in maintaining a fair rate of work. This has been due principally to its advantageous geographical location, numerous railroad connections and steady power service. Added to these favorable factors, there has been coöperation among the mining companies and a considerable resourcefulness in looking after the labor and in bringing in supplies.

Like other countries at present under strain, Mexico has found it necessary to increase all taxes very materially, including those on mining property and on metals produced and exported. Very luckily, the prevailing high metal prices still enable the silver and base-metal producers to make a profit, but the gold-mine owner is hard hit with no rise in the

market price of his product to offset steadily increasing costs as affected by the taxes, higher prices for supplies, etc.

The labor outlook is not encouraging. As a natural outcome of several years of revolutions, labor is also passing through a similar period of unrest, and much tact combined with fair and firm treatment will be necessary to meet this situation in any way successfully. The conditions are likely to be accentuated, due to a shortage of labor supply when mining activity throughout Mexico is again possible.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# A Comparative Test of The Marathon, Chilean and Hardinge Mills

BY F. C. BLICKENSDERFER,* MORENCI, ARIZ.

(Arizona Meeting, September 1916)

## Introduction

During 1914 and 1915 extensive experiments were conducted at the concentrator of The Detroit Copper Mining Co. of Arizona, at Morenci, Ariz., in order to test the relative grinding efficiencies of the Marathon, the Chilean and the Hardinge Mills.

The Marathon mill used in the experiments was the first one ever built by the manufacturers; and since this was the first time that it was given a thorough tryout, with such remarkable results, the data have been assembled in the following paper and submitted in the interests of the milling profession.

## Description of the Flow Sheet

The concentrator flow sheet, as it existed at the time of these tests, is shown in Fig. 1. The total mill feed, as delivered from the crushing plant, had as its maximum size 1-in. material. This material was screened dry on Cole Zig-Zag screens fitted with 4-mesh openings; the oversize going to one pair of coarse rolls, the undersize to five Wilfley tables fitted with Butchart National riffles. The coarse-roll product was elevated and screened wet on Zig-Zag screens fitted with 4-mesh openings, the undersize going to the five tables mentioned above and the oversize to one pair of fine rolls.

The fine-roll product was elevated and screened together with the coarse-roll product; so that, before any concentrating operations took place, the total mill feed was reduced to pass a 4-mesh screen.

After concentration on Butchart-riffled tables, the tailings from these tables passed to de-sliming hoppers, where two products were formed, *viz.*, an overflow and a discharge. The discharge constituted the feed for the Chilean mills. It contained 16 per cent. slime (−200 mesh) and 3 parts of water to 1 of solids. The concentrator was equipped regularly with four Chilean mills (Monadnock type) to the unit; but, in order to

* Concentrator Metallurgist, Detroit Copper Mining Co.

provide similar conditions for a test, two of these mills were replaced by a Hardinge mill and a Marathon mill. The Hardinge mill was fed with the discharge from the de-sliming hopper; but a greater proportion of the slime was removed in order to get a drier feed, containing but 8 per cent. slime (−200 mesh) and with a ratio of 1.3 parts water to 1 of solids. The Marathon mill was also fed with the discharge from the de-sliming

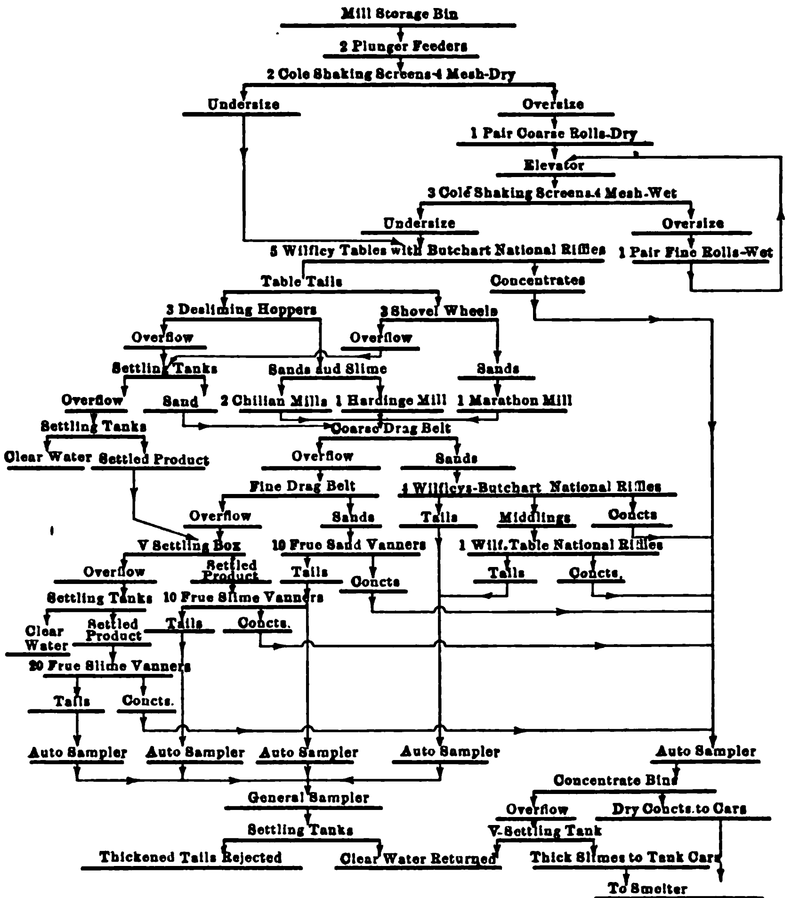


FIG. 1.—FLOW SHEET OF 750-TON CONCENTRATOR UNIT OF DETROIT COPPER MINING CO., MORENCI, ARIZ.

hopper, containing 12 per cent. slime and having a water to solids ratio of 1.8 : 1. Three shovel-wheels were temporarily installed to remove all of the slime from the table tails, for the purpose of giving a much larger tonnage to the Marathon mill, it being found that the total normal unit tonnage could be handled by the three shovel-wheels.

All of the mill products passed to a coarse drag belt which took out the coarse sand for treatment on the Butchart-riffled tables; the coarse

drag-belt overflow flowed to a fine drag belt, which removed fine sand for treatment on Frue sand-vanners. The overflow from the fine drag belt passed to a V-box which removed a thick product for treatment on Frue slime-vanners; the V-box overflow was thickened for final treatment on Frue slime-vanners. All rejected tailings were sampled separately as they came from each group of machines, then a general sample of the tails was taken after all of the tailings came together at a common point. The clear water at each point, if at a sufficiently high level, was used for wash water on machines below; but, if not, it was collected at the bottom of the concentrator and pumped to tanks at the top to be used over again.

The crude milling ore is a quartz-monzonite porphyry with chalcocite as the copper mineral. An average assay for one year is: Cu, 2.80; $SiO_2$, 59.5; $Al_2O_3$, 16.8; Fe, 5.0; S, 5.0; CaO, 0.7; and MgO, 0.9 per cent. The ore contains a small percentage of copper sulphate, which is soluble in water. This "acid water" is very destructive to all iron parts, so that iron consumed in all these tests is excessive to a small degree for quantities of feed crushed. Unslacked lime is added continuously at the rate of 2,000 lb. daily to circulating mill water to neutralize the copper sulphate.

## General Conditions Regarding the Tests

These experiments were carried on for several months, so that results obtained are not spasmodic, but represent the average results of routine work of each mill under all the variable conditions encountered in actual milling practice.

The Chilean and Hardinge mills ran together for a continuous period of 63 days, or for the length of time required to wear out the tires and die of the Chilean mill. The Hardinge mill ran the first 30 days with an old lining. The old lining was then removed, a new one put in and the test completed.

The average results of the Hardinge and Chilean mills for the 63-day period will be used in comparison with the Marathon mill. Test No. 1 of the Marathon mill was run under the same conditions regarding feed as the Hardinge and Chilean, so that an excellent comparison is possible.

Some time elapsed between tests Nos. 1 and 2 of the Marathon mill, during which some preliminary tests were made to determine the best conditions for operating with a heavy tonnage. During this time the three shovel-wheels were installed to remove all slimes and the greater amount of water. Also about this time the tonnage per unit was reduced from 750 tons to 500 tons per 24 hr., and this total unit feed was sent through the Marathon mill. Subtracting the weight of concentrates produced in primary table concentration, and the weight of slimes removed by the shovel-wheels preceding the Marathon mill, a net dry

weight of 430 tons per 24 hr. was obtained. The total recovery of the unit under these conditions was almost exactly the same as the second unit operating three Chilean mills in the same step. This fact paved the way for a heavy tonnage on the Marathon mill, known in this report as test No. 2.

## Description of the Three Types of Mills Tested

The Chilean mill used in the test was of the Monadnock type, 5 ft. in diameter, and fitted with 1½-mm. round-hole punched screens. The Chilean die (or ring), when new, weighed 1,722 lb., the three tires together weighed 2,526 lb. A 50-hp. motor furnished power to drive the Chilean mill, transmission taking place in two stages, viz., from motor pulley to line shaft, thence to Chilean mill belt wheel.

The cylindrical portion of the Hardinge mill was 8 ft. in diameter by 3 ft. long. It was lined with Danish pebbles of uniform size, set in concrete. The method of lining was as follows: The lower half was lined

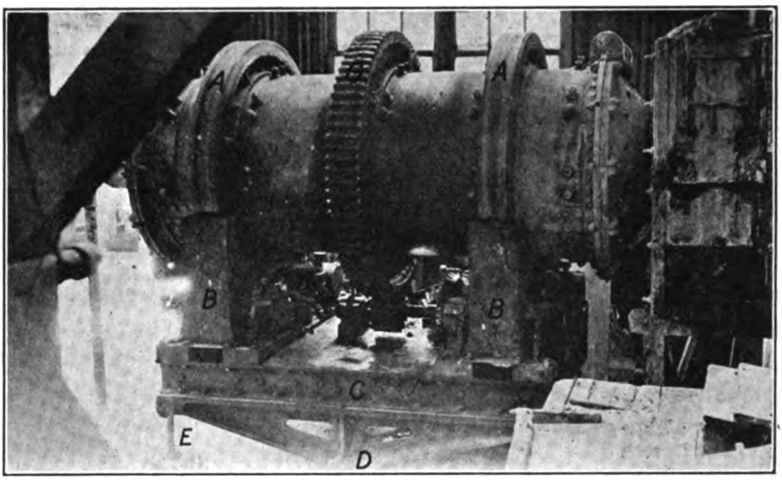


Fig. 2.—Old Marathon Mill.

first and allowed to "set," then the mill was turned over so that the unlined portion was below. This section was then lined and the total lining allowed to set 48 hr. The Hardinge mill was charged with 5 tons of Danish pebbles of fairly uniform size. Power was furnished by a 75-hp. motor direct-connected to a gear shaft which in turn engaged with the large gear wheel of the same diameter as the mill itself.

The Marathon mill consisted essentially of a tube, 3 ft. in diameter by 7 ft. long, surrounded by two tires, *A* (Fig. 2), near each end, these tires

being supported by and revolving upon two pairs of rollers, *B*. Two idler rollers, *H*, keep the tires in alignment with the rollers when the mill is in action. The four rollers *B* are set on frame *C* and are rigidly bolted to it. Frame *C* is pivoted at its center, *D*, thus providing a tilting arrangement, which permits of adjustment to any desired slope by bolts, *E*, at either end of the frame. The ends are bolted to the mill tube, and can be quickly and easily removed for the inspection or renewal of crushing parts. The feeding arrangement consists of an L-shaped tube, the horizontal leg of which discharges into the mill through a circular opening in the center of the head end. The product, in wet crushing, is discharged through a 6-in. pipe centrally located in the tail end. In dry crushing, the Marathon mill has a special end with slots in the periphery for the free discharge of the crushed product. A large hopper is set below the discharge end to collect the product.

The Marathon mill lining consists of 16 cast-iron plates bolted to the mill shell, the total weight of the plates being 4,480 lb. The crushing medium is composed of iron rods ranging in size from ½ in. to 2 in. in diameter; the initial charge of rods weighed 7,000 lb. The mill is driven by a gear wheel, *G*, connected by two sets of gears to a drive wheel, which is belted to a 25-hp. motor.

### *Modifications of Original Design of Marathon Mill*

During the progress of the experimental work with the Marathon mill, several construction defects developed, which had to be remedied before further operation could proceed. A preliminary run with no load of ore in the mill, showed that the understructure was not rigid enough to prevent excessive vibration. The top and bottom of the frame were entirely covered with ½-in. and 5/16-in. sheet-steel plates respectively, the plates being firmly riveted to the legs of channel irons composing the top of the tilting frame.

The feed spout as furnished with the mill gave endless trouble with a wet feed, because the pulp was dashed out around the feed spout where it enters the center of the mill. The feed spout was replaced by a scoop feeder which was bolted to the head end of the mill.

A circular cast-iron liner (1¼ in. thick at the edges by 2¼ in. at the center) of the same diameter as the inside of the mill, was bolted to the head end to protect it against excessive wear.

The roller bearings supporting the axles of rollers were very unsatisfactory. Being of a spiral type, they were forced to extend by the weight of the mill to such a degree that the bearing end was thrust off. This extension of spiral rollers caused a reduction in their diameter, which resulted in a loose wabbly bearing. In case of such a failure the roller-bearing parts were renewed.

### *Methods of Operating the Three Types of Mills*

The best method of operating the Chilean mill is perhaps too well known to require a detailed description. It is essential to have sufficient water in the feed for the Chilean mill to obtain best results from the mill screens, for if the pulp is too thick the screens clog and cause the mill to become overloaded. If too great a quantity of pulp is fed to the mill, overloading follows at once and the mill does practically no grinding. When the Chilean mill became overloaded, the power required to drive it was about one-third more than under normal conditions. An average ratio of 3 parts water to 1 part solids (25 per cent. solids) was the best pulp to feed to the Chilean mill for all-around good work. The speed was 40 r.p.m.

The Hardinge required a less dilute pulp (42.8 per cent. solids) for best results. It is practically impossible to overload the Hardinge mill, "so to speak," as long as the mill is able to discharge the product, but constant attention is required to prevent "oversize." When its grinding capacity is exceeded, a considerable amount of very coarse material traverses the length of the mill without receiving a single blow from the falling pebbles. Special attention was given each of the mills in order to keep them up to full capacity with the production of a minimum quantity of "oversize." The mill was operated at 29 r.p.m.

The Marathon mill in Test No. 1 was operated as a fine grinding machine so as to give a product of the same fineness as the Hardinge and the Chilean mills. The most favorable results were obtained with pulp that contained 36.7 per cent. solids. When the Marathon mill became overloaded it was surprising to find no coarse oversize in the product, *but a product that was uniformly coarser*. This is a fact that requires special notice. It is equivalent to saying that if the mill is once started up it is not necessary to pay any attention to the product, for if overloaded, the whole product will be slightly coarser, and if underloaded the product will be slightly finer.

In test No. 2 of the Marathon mill the amount of water and slime in the feed was reduced to a minimum by shovel-wheels to secure a less volume of pulp and a consequent slower velocity of pulp through the mill. The tonnage of test No. 2 was greatly increased over that of test No. 1, and the Hardinge and Chilean mill tonnages. The feed contained an average of 63.5 per cent. solids, and only 2.92 per cent. slime (−200 mesh). The speed in all the tests of the Marathon mill was 30 r.p.m. In a diameter of 3 ft. at 30 r.p.m. there is no dead zone in the periphery of the mill, while excessive slipping of rods is prevented by having the liner plates thicker at each edge than in the middle, resulting in a corrugated interior surface. The lining retains this corrugated surface until it is worn out. The rods wear evenly from end

to end, do not get crosswise in the mill and are reduced to a diameter of ⅜ in. before they begin to crush, flatten out or break into pieces. When worn to this thinness, some rods roll up and are discharged; the short half-length pieces retain their positions in the charge of rods without any serious results, but may reduce crushing efficiency to some extent. It is good practice to take out the entire charge of rods every 10 days, sort out the small and "disabled" rods and replace the loss in weight with 1¼-in. rods. As soon as the weight of rods consumed per day was determined, their loss in weight was replaced daily with fresh rods. The mill must be closed down to add the daily charge of rods. During these tests the Marathon mill was tilted ¼ in. per foot, but later tests proved that it would produce equally as good results when run horizontally.

Some experiments were made on coarse (+ ½-in.) material both wet and dry, using the peripheral discharge end and feed spout. The charge of rods in the mill was increased by the addition of steel shafting 4 in. in diameter. These tests were not extended over any length of time, nor were sufficiently conclusive results obtained to warrant their mention. The primary fact established was this: The mill was able to handle coarse feed up to 2 in. in a way to encourage further experiment.

### *Explanation of the Action of the Marathon Mill*

It might be well at this point to offer an explanation of the action that takes place within the Marathon mill.

For the sake of comparison, if we select any type of mill with spherical-shaped grinding media, the Hardinge mill for example, and consider a single crushing element, we have a row of pebbles in contact with another row of pebbles as indicated in Fig. 3.

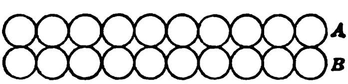


FIG. 3.

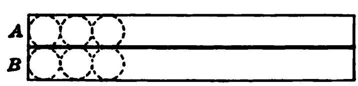


FIG. 4.

Now when falling, row *A* strikes row *B*, or if rolling together, row *A* rolls on row *B*. It is evident that crushing takes place where one pebble strikes or rolls against another, or at their point of contact. The number of points of contact in an element is determined by the size of pebbles; the smaller the size of pebbles, the greater the number of points of contact and *vice versa*. But it has been demonstrated in actual practice that the pebbles must be above a certain size to develop a crushing blow when falling of sufficient force to be of any value. Hence crushing ability depends not only on points of contact, but also on the size of pebbles and on their mass (by *mass* is meant specific gravity).

Since the number of points of contact must vary inversely as the size of pebbles, the two factors are compromised by using various sizes of pebbles. In order to take advantage of the mass factor, iron balls have been recently substituted for pebbles as a crushing medium.

Now consider a solid bar for the grinding medium, in the case of the Marathon mill, as shown in Fig. 4. When falling, rod *A* strikes rod *B*, or if rolling together, rod *A* rolls on rod *B*. There is a continuous row of points of contact, or geometrically speaking, a line. It has been shown in experiments that the rods remain parallel when the mill is in operation. Hence we have the maximum number of contact points that

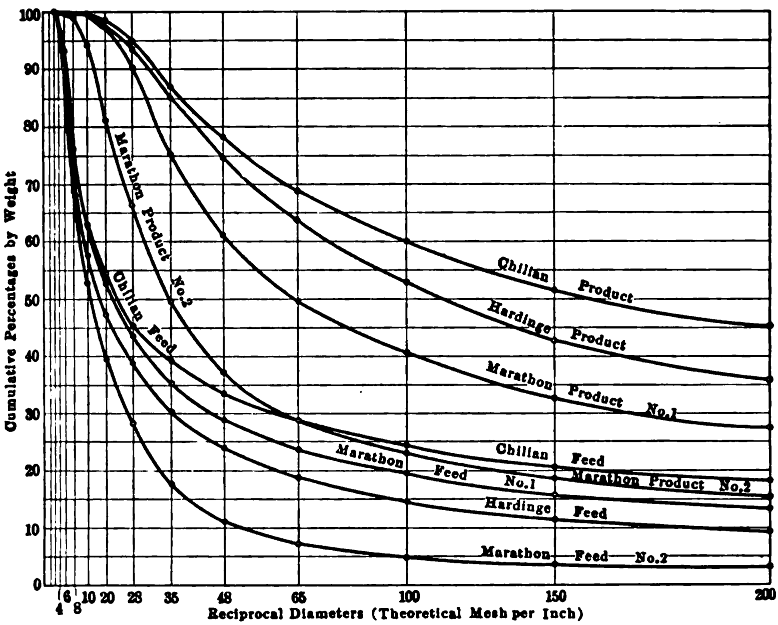


Fig. 5.—Diagram of Crushing Efficiency—Gates Method.

can exist between the surfaces of two rods (cylindrically shaped). Again, we may consider the two rods *A* and *B* of Fig. 4 as made up of solid iron balls packed closely and bound together throughout their length by additional iron which is 0.4764 of their own weight. Therefore, the crushing ability of iron rods—bearing in mind that crushing ability depends upon points of contact and upon the mass of crushing medium—must be greatly superior to those mills which use spherical-shaped crushing media. The relatively small diameter of rods, 2 in. to ⅜ in., causes them to fit compactly, thus requiring a minimum falling distance to deliver a crushing blow. The center of mass of a charge of rods is much nearer the axis of the mill than in a mill of greater diameter whose crushing charge consists of steel balls or pebbles.

It is necessary in ball and pebble mills to have a sufficient weight of charge to deliver a crushing blow. This condition is effected either by making the mill of large diameter or by increasing its length. When the length is increased the power factor increases, when the diameter is increased the power factor increases very rapidly and at the same time there is danger of the peripheral speed becoming too great, tending to form a dead zone.

## Tables of Results Showing Items and Conclusions

Table 1.—*Cost Comparisons*

| | Marathon Test No. 1, Per Cent. | Marathon Test No. 2, Per Cent. | Hardinge Mill, Per Cent. | Chilean Mill, Per Cent. |
|---|---|---|---|---|
| Operating Labor | 13.38 | 11.21 | 7.75 | 11.63 |
| Labor on Repairs | 12.20 | 12.27 | 3.85 | 7.43 |
| Material on Repairs | 40.38 | 42.29 | 28.40 | 22.52 |
| Power | 34.04 | 34.23 | 60.00 | 58.42 |
| | 100.00 | 100.00 | 100.00 | 100.00 |

| Marathon No. 1 is Exceeded in Operating Costs by | Excess Cost, Per Cent. |
|---|---|
| Marathon No. 2 | 19.34 |
| Hardinge | 72.56 |
| Chilean | 15.00 |

| Chilean is Exceeded in Operating Costs by | Excess Cost, Per Cent. |
|---|---|
| Marathon No 2 | 3.77 |
| Hardinge | 50.04 |

| Marathon No. 2 is Exceeded in Operating Cost by | Excess Cost, Per Cent. |
|---|---|
| Hardinge | 44.59 |

Costs are generally governed by local conditions. In each of the divisions, Operating Labor, Labor on Repairs, Material on Repairs, and Power, the percentage of the total cost of the mill is represented, no correction being made for the tonnage. From the second part of this table it is to be noted that the Marathon mill costs as operated in test No. 1 are the least; the Chilean mill costs are the next lowest; then the Marathon mill costs as operated in test No. 2; and the Hardinge mill costs are the highest. In comparing the costs of one mill with those of any other, for instance the Hardinge costs and the Chilean costs, the mill with the lowest costs is taken as 100 per cent. The Hardinge costs on this basis are 150.04 per cent., or 50.04 per cent. in excess of the Chilean costs.

Material on Repairs includes rods and liners for the Marathon mill, die and tires for the Chilean mill, and pebble lining and pebbles for the Hardinge mill.

In each case the power factor is strongly in favor of the Marathon mill. The tonnage factor is omitted at this point because it properly belongs to a comparison of grinding efficiencies.

TABLE 2.—*Record of the Average for Each Test*

| Type of Mill | Running Time in Per Cent. of Total Time | Time Lost, Per Cent. | Total Actual Running Time in Hours | Tons Dry Feed in 24 Hr. | Tons Dry Feed per Hour | Tons Pulp in 24 Hr. | Tons Pulp per Hour | Per Cent. Solids in Feed |
|---|---|---|---|---|---|---|---|---|
| Marathon No. 1 | 94.00 | 6.00 | 383.0 | 236.5 | 9.854 | 662.4 | 27.60 | 35.7 |
| Marathon No. 2 | 94.25 | 5.75 | 200.0 | 440.0 | 18.333 | 693.0 | 28.87 | 63.5 |
| Hardinge | 96.25 | 3.75 | 1,455.3 | 244.0 | 10.166 | 570.1 | 23.75 | 42.8 |
| Chilean | 93.75 | 6.25 | 1,401.7 | 237.0 | 9.875 | 951.4 | 39.64 | 24.9 |

| Type of Mill | Ratio of Solids to Water | Water with Each Ton of Feed, Gallons | Water with 24-Hr. Tonnage, Gallons | Water in Pulp, Gallons per Minute | Kilowatts Consumed per Hour | Horsepower Consumed per Hour | Tons Dry Feed per Hp.-hr. | Tons Dry Feed per Hp.-day | Speed of Mills, R.p.m. |
|---|---|---|---|---|---|---|---|---|---|
| Mar. No. 1 | 1 to 1.8 | 432.2 | 102,215 | 70.98 | 13.880 | 18.60 | 0.5316 | 12.758 | 30.27 |
| Mar. No. 2 | 1 to 0.57 | 138.0 | 60,720 | 42.17 | 16.791 | 22.50 | 0.8148 | 19.555 | 30.10 |
| Hardinge | 1 to 1.3 | 320.6 | 78,226 | 54.32 | 42.208 | 56.56 | 0.1797 | 4.313 | 29.03 |
| Chilean | 1 to 3.0 | 723.8 | 171,540 | 119.13 | 27.388 | 36.70 | 0.2691 | 6.485 | 39.88 |

The term pulp is here used to signify a mixture of water and dry feed, and the quantity of water in the pulp is the amount in combination with the dry feed.

The delays of the Hardinge mill are due to relining and allowing time for lining to "set." The delays of Chilean mill are due to renewing of crushing parts, replacing of screens and oiling of mill. The delays of the Marathon mill are due to changing of rod charge, addition of new rods, relining, and extraordinary repairs on defective parts.

The power consumed by the Marathon mill is practically constant, the rise in power consumed in test No. 2 being due to an increased charge of rods. The Hardinge power is also constant. The Chilean mill consumes less power when the tires and die are new than when they have been in operation for some time. If mills are slightly overloaded, the power increases very rapidly.

TABLE 3.—*Average of Test Conditions*

| Type of Mill | Stationary Grinding Parts: Kind of Stationary Grinding Surface | Wt. of Stationary Grinding Surface, Pounds. | Life of Liners or Die, Days | Liners or Die Consumed in 24 Hr., Lb. | Liners or Die per Ton, Pounds | Movable Grinding Parts: Charge Used in Mill | Wt. of Initial Charge, Pounds. | Total Pounds Added During Test | Pounds Consumed in 24 Hr. | Consumed per Ton of Feed, Pounds. | Total Grinding Parts: Consumed Per Ton of Feed, Pounds. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Mar. No. 1... | Iron | 4,480 | 82 | 54.707 | 0.2313 | Iron | 6,395 | 2,404 | 150.64 | 0.63697 | 0.86827 |
| Mar. No. 2... | plates | 4,480 | 72 | 62.256 | 0.14149 | rods | 7,124 | 1,441 | 172.92 | 0.39295 | 0.53544 |
| Hardinge..... | Pebbles | 6,740 | 159 | 41.761 | 0.17115 | Pebbles | 10,175 | 31,956 | 527.00 | 2.16 | 2.33115 |
| Chilean...... | Die | 1,722 | 63 | 27.333 | 0.11533 | 3 Tires | 2,526 | None | 41.66 | 0.1762 | 0.29153 |

This table expresses the life of linings, the rate of wear, and pounds of each grinding medium sacrificed per ton of ore ground. The Chilean mill die corresponds to liners and lining of the other two mills. It is to be noted that the Chilean mill required the least amount of grinding medium per ton. It is the auxiliary parts, such as screens, screen frames, tire wheels or mullers, feed spouts, ploughs, mortars, and the power, that completely reverse this good quality.

TABLE 4.—*Comparison of Grinding Efficiencies*

| Marathon No. 1: Mesh | Reciprocal of Average Diameter | Feed, Per Cent. by Weight | Relative Surface in Feed | Product, Per Cent. by Weight | Relative Surface in Product | Marathon No. 2: Mesh | Reciprocal of Average Diameter | Feed, Per Cent. by Weight | Relative Surface in Feed | Product, Per Cent. by Weight | Relative Surface in Product |
|---|---|---|---|---|---|---|---|---|---|---|---|
| +4 | 4.46 | 0.13 | 0.58 | ...... | ......... | +4 | 4.46 | 0.09 | 0.40 | ...... | ......... |
| +6 | 6.33 | 6.49 | 41.08 | ...... | ......... | +6 | 6.33 | 9.37 | 59.31 | 0.04 | 0.25 |
| +8 | 8.93 | 17.12 | 152.88 | 0.01 | 0.09 | +8 | 8.93 | 21.58 | 192.71 | 0.81 | 7.23 |
| +10 | 12.66 | 13.40 | 169.64 | 0.28 | 3.54 | +10 | 12.66 | 15.90 | 201.29 | 4.90 | 62.03 |
| +14 | 17.86 | 10.44 | 186.46 | 2.12 | 37.86 | +14 | 17.86 | 13.55 | 242.00 | 12.05 | 215.21 |
| +20 | 25.38 | 8.51 | 215.98 | 7.06 | 179.18 | +20 | 25.38 | 11.14 | 282.73 | 15.55 | 394.66 |
| +28 | 35.71 | 8.35 | 298.18 | 14.77 | 527.44 | +28 | 35.71 | 10.42 | 372.10 | 16.82 | 600.64 |
| +35 | 50.51 | 6.61 | 333.87 | 14.43 | 728.86 | +35 | 50.51 | 6.96 | 351.55 | 12.50 | 631.37 |
| +48 | 71.43 | 5.30 | 378.58 | 11.68 | 834.30 | +48 | 71.43 | 3.92 | 280.01 | 8.53 | 609.30 |
| +65 | 101.01 | 3.91 | 394.95 | 8.72 | 880.81 | +65 | 101.01 | 1.97 | 198.99 | 5.43 | 548.48 |
| +100 | 142.86 | 3.78 | 540.01 | 8.05 | 1,150.02 | +100 | 142.86 | 1.15 | 164.29 | 4.67 | 667.16 |
| +150 | 200.00 | 2.46 | 492.00 | 5.15 | 1,030.00 | +150 | 200.00 | 0.66 | 132.00 | 2.82 | 564.00 |
| +200 | 285.71 | 1.29 | 368.57 | 2.67 | 762.85 | +200 | 285.71 | 0.37 | 105.71 | 1.70 | 485.71 |
| −200 | 454.55 | 12.21 | 5,550.05 | 25.06 | 11,391.02 | −200 | 454.55 | 2.92 | 1,327.29 | 14.18 | 6,445.52 |
| Total...... | | 100.00 | 9,122.83 | 100.00 | 17,525.97 | Total...... | | 100.00 | 3,910.38 | 100.00 | 11,231.56 |

TABLE. 4.—(*Continued*)

| Hardinge Mill | | | | | | Chilean Mill | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Mesh | Reciprocal of Average Diameter | Feed, Per Cent. by Weight | Relative Surface in Feed | Product, Per Cent. by Weight | Relative Surface in Product | Mesh | Reciprocal of Average Diameter | Feed, Per Cent. by Weight | Relative Surface in Feed | Product, Per Cent. by Weight | Relative Surface in Product |
| +4 | 4.46 | 0.97 | 4.33 | ...... | ......... | +4 | 4.46 | 0.94 | 4.19 | ...... | ......... |
| +6 | 6.33 | 8.43 | 53.36 | ...... | ......... | +6 | 6.33 | 7.71 | 48.80 | ...... | ......... |
| +8 | 8.93 | 18.79 | 167.79 | 0.07 | 0.63 | +8 | 8.93 | 16.49 | 147.26 | 0.05 | 0.45 |
| +10 | 12.66 | 13.86 | 175.47 | 0.58 | 7.34 | +10 | 12.66 | 11.84 | 149.89 | 0.10 | 1.26 |
| +14 | 17.86 | 10.45 | 186.64 | 1.95 | 34.83 | +14 | 17.86 | 8.98 | 160.38 | 1.07 | 19.11 |
| +20 | 25.38 | 8.49 | 215.48 | 3.80 | 96.44 | +20 | 25.38 | 7.18 | 182.23 | 3.63 | 92.13 |
| +28 | 35.71 | 8.40 | 299.96 | 8.17 | 291.75 | +28 | 35.71 | 7.43 | 265.33 | 7.75 | 276.75 |
| +35 | 50.51 | 6.53 | 329.83 | 10.45 | 527.83 | +35 | 50.51 | 6.04 | 305.08 | 9.08 | 458.63 |
| +48 | 71.43 | 5.29 | 377.86 | 11.60 | 828.59 | +48 | 71.43 | 4.80 | 342.86 | 9.38 | 670.01 |
| +65 | 101.01 | 4.10 | 414.14 | 10.25 | 1,035.35 | +65 | 101.01 | 3.89 | 392.93 | 8.53 | 861.62 |
| +100 | 142.86 | 3.47 | 495.72 | 10.40 | 1,485.74 | +100 | 142.86 | 3.76 | 537.15 | 8.95 | 1,278.60 |
| +150 | 200.00 | 1.81 | 362.00 | 6.75 | 1,350.00 | +150 | 200.00 | 2.58 | 516.00 | 6.23 | 1,246.00 |
| +200 | 285.71 | 1.04 | 297.14 | 4.13 | 1,179.98 | +200 | 285.71 | 1.82 | 519.99 | 4.21 | 1,202.84 |
| −200 | 454.55 | 8.37 | 3,804.58 | 31.85 | 14,477.42 | −200 | 454.55 | 16.54 | 7,518.26 | 41.02 | 18,645.64 |
| Total.... | ...... | 100.00 | 7,184.30 | 100.00 | 21,315.90 | Total | | 100.00 | 11,090.35 | 100.00 | 24,753.04 |

| | Marathon No. 1 | Marathon No. 2 | Hardinge | Chilean |
|---|---|---|---|---|
| Work units in product......... | 17,525.97 | 11,231.56 | 21,315,90 | 24,753.04 |
| Work units in feed ........... | 9,122.83 | 3,910.38 | 7,184.30 | 11,090.35 |
| Work units expended in crushing | 8,403.14 | 7,321.18 | 14,131.60 | 13,662.69 |
| Work units expended in crushing corrected for tonnage.......... | 107,207.3 | 143,165.70 | 60,949.60 | 88,602.54 |

| Excess Efficiency of Marathon No. 2 over | Per Cent. |
|---|---|
| Marathon No. 1 ........................................ | 33.54 |
| Hardinge........................................ | 134.80 |
| Chilean ........................................ | 61.58 |
| **Excess Efficiency of Marathon No. 1 over** | **Per Cent.** |
| Hardinge........................................ | 75.90 |
| Chilean ........................................ | 30.52 |
| **Excess Efficiency of Chilean over** | **Per Cent.** |
| Hardinge........................................ | 45.37 |

Tyler Standard screens were used in all of the screen analyses, and constants of this screen scale are employed in computations. The principle upon which the comparison of efficiencies is based may be summed up in a few words: "Work done in crushing is proportional to surface exposed in crushing, and therefore nearly proportional to reduction in diameter," or, "nearly proportional to the reciprocals of diameters crushed to." This is the well-known and substantial efficiency comparison of Del Mar. Per cent. weight is multiplied by reciprocals of *average diameter* in both feed and product, the difference in these two quantities measuring the work expended in crushing. When this quantity is multiplied by tons per horsepower-day, we involve the two factors that complete the efficiency comparison. This method is commendable with but one exception; the assuming of a value for the average diameter of −200-mesh material. This assumed value probably comes somewhere near to the approximate value, but no more. In logarithmic plotting of feeds and products, Fig. 6, the lines have been extended in a smooth curve to the edge of the paper, and in every case the assumed value is only approximate. It is of no advantage to assume this value, especially when there is large quantity of −200-mesh material. The curve of Chilean product lacks sufficient points to tell just where it does extend. The others have a fair degree of accuracy.

### *Crushing-Surface Diagram*

This is the method proposed by Arthur O. Gates for comparing efficiencies in grinding. Cumulative percentages are plotted vertically and reciprocals of screen-mesh diameters, horizontally (Fig. 5). This

TABLE 5.—*Gates Method of Comparison*

| Mesh | Reciprocal of Mesh Diameter | Mar. No. 1 Feed Cumulative, Per Cent. by Weight | Mar. No. 1 Product Cumulative, Per Cent. by Weight | Mar. No. 2 Feed Cumulative, Per Cent. by Weight | Mar. No. 2 Product Cumulative, Per Cent. by Weight | Hardinge Feed Cumulative, Per Cent. by Weight | Hardinge Product Cumulative, Per Cent. by Weight | Chilean Feed Cumulative, Per Cent. by Weight | Chilean Product Cumulative, Per Cent. by Weight |
|---|---|---|---|---|---|---|---|---|---|
| +4 | 5.40 | 100.00 | ........ | 100.00 | ........ | 100.00 | ........ | 100.00 | ........ |
| +6 | 7.63 | 99.87 | ........ | 99.91 | 100.00 | 99.03 | ........ | 99.06 | ........ |
| 8 | 10.75 | 93.38 | 100.00 | 90.54 | 99.96 | 90.60 | 100.00 | 91.35 | 100.00 |
| 10 | 15.38 | 76.26 | 99.99 | 68.96 | 99.15 | 71.81 | 99.93 | 74.86 | 99.95 |
| 14 | 21.74 | 62.86 | 99.71 | 53.06 | 94.25 | 57.95 | 99.35 | 63.02 | 99.85 |
| 20 | 30.48 | 52.42 | 97.59 | 39.51 | 82.20 | 47.50 | 97.40 | 54.04 | 98.78 |
| 28 | 43.10 | 43.91 | 90.53 | 28.37 | 66.65 | 39.01 | 93.60 | 46.86 | 95.15 |
| 35 | 60.98 | 35.56 | 75.76 | 17.95 | 49.83 | 30.61 | 85.43 | 39.43 | 87.40 |
| 48 | 86.20 | 28.95 | 61.33 | 10.99 | 37.33 | 24.08 | 74.98 | 33.39 | 78.32 |
| 65 | 121.95 | 23.65 | 49.65 | 7.07 | 28.80 | 18.79 | 63.38 | 28.59 | 68.94 |
| 100 | 172.41 | 19.74 | 49.93 | 5.10 | 23.37 | 14.69 | 53.13 | 24.70 | 60.41 |
| 150 | 243.90 | 15.96 | 32.88 | 3.95 | 18.70 | 11.22 | 42.73 | 20.90 | 51.46 |
| 200 | 344.82 | 13.50 | 27.73 | 3.29 | 15.88 | 9.41 | 35.98 | 18.36 | 45.23 |
| −200 | ........ | 12.21 | 25.06 | 2.92 | 14.18 | 8.37 | 31.85 | 16.54 | 41.02 |

TABLE 5.—(*Continued*)

From Crushing Diagram by Gates Method (Fig. 5)

| | Square Inches | Efficiency Corrected for Tonnage |
|---|---|---|
| Marathon No. 1 | 7.51 | 95.812 |
| Marathon No. 2 | 6.65 | 130.004 |
| Hardinge | 12.02 | 51.842 |
| Chilean | 11.12 | 72.113 |

| Excess Efficiency of Marathon No. 2 over | Per Cent. |
|---|---|
| Marathon No. 1 | 35.69 |
| Hardinge | 150.75 |
| Chilean | 80.29 |
| **Excess Efficiency of Marathon No. 1 over** | **Per Cent.** |
| Hardinge | 84.81 |
| Chilean | 32.86 |
| **Excess Efficiency of Chilean over** | **Per Cent.** |
| Hardinge | 39.10 |

diagram averages the diameters without calculation, and areas upon it are proportional to surface produced, and, in accordance with Rittinger's law, to energy spent in crushing alone. Measuring these areas up to 200 mesh and multiplying by tons per horsepower-day, we obtain efficiency units for comparison, as indicated in Table 5.

TABLE 6.—*Comparison of Del Mar and Gates Methods*

| Excess Units by | Del Mar, Per Cent. | Gates, Per Cent. | Numerical Average, Per Cent. |
|---|---|---|---|
| Marathon No. 2 over | | | |
| Marathon No 1 | 33.54 | 35.69 | 34.61 |
| Hardinge | 134.80 | 150.75 | 142.77 |
| Chilean | 61.58 | 80.29 | 70.93 |
| Marathon No. 1 over | | | |
| Hardinge | 75.90 | 84.81 | 80.35 |
| Chilean | 30.52 | 32.86 | 31.69 |
| Chilean over | | | |
| Hardinge | 45.37 | 39.10 | 42.23 |

In view of the fact that in Del Mar's system the $-200$ mesh material is taken into account and in the other it is not, we may consider a check existing between the two methods with but one exception, comparing

Marathon No. 2 with Chilean, the two extremes of feeds and products; the one producing the most slime and the other producing the least slime. To be impartial and conservative at the same time, we will adopt the numerical average of the two methods.

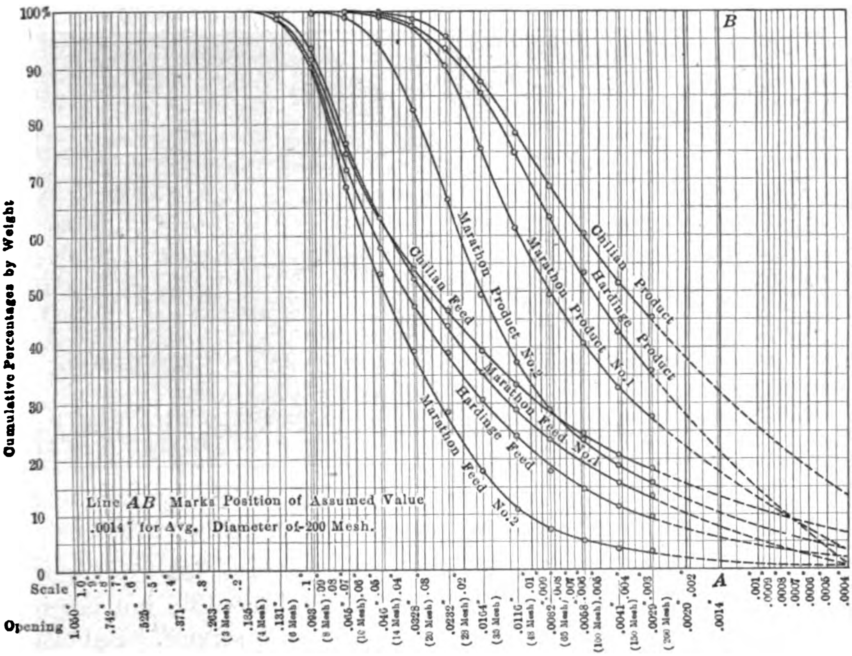


FIG. 6.—CUMULATIVE LOGARITHMIC PLOT OF SCREEN ANALYSES ON FEEDS AND PRODUCTS.

## SUMMARY

For the material treated and under the conditions of these tests, the Marathon mill is far superior to the Hardinge and Chilean mills in grinding efficiency. Of the Hardinge and Chilean mills, the Chilean is superior to the Hardinge. The Marathon mill will bear further investigation and experiment. Its costs will be much less in the actual installation of several mills than in a single mill requiring perfection of parts, as did this one.

On the basis of the results obtained in these tests, two mills, 4 ft. in diameter by 8 ft. long, for both coarse and fine grinding (Fig. 7) have been ordered, the principal features of which are as follows:

(*a*) Feed scoop attached to the head end of the mill. There are three channels in this scoop, entrance of each channel having a rectangular area of 108 sq. in. (9 by 12 in.).

(b) Sectional liners for head end and solid circular liner for tail end.

(c) Rigid base, with no tilting arrangement.

(d) Solid roller bearings (not spiral).

(e) Driving gear attached to tail end of shell, the pulley being supported rigidly by having its bearings set solidly on a separate concrete

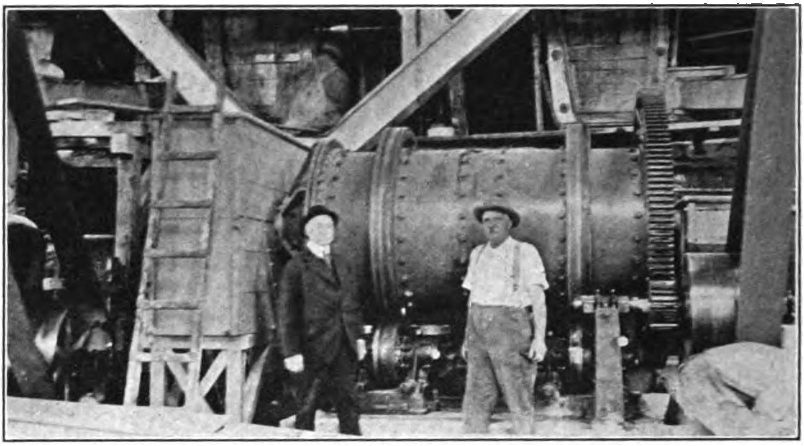

Fig. 7.—New Marathon Mill.

base. One of these mills has been in operation for a few days and is running smoothly after having considerable trouble with a hot bearing. These bearings are not able to stand up under such service. Babbitted bearings will undoubtedly remedy this trouble.

Thanks are due to Milton H. McLean, General Manager of the Detroit Copper Mining Co. of Arizona, for permission to assemble and publish the above data.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Illuminating Power of Safety Lamps

BY W. M. WEIGEL,* E. M., STATE COLLEGE, PA.

(Arizona Meeting, September, 1916)

WHILE electric lamps both of the cap and hand type are being introduced into many mines requiring the use of safety lamps, the oil-burning safety lamp is still used in the great majority of cases, and even where the former are used the latter must continue to be employed when testing for gas. It is not the intention to discuss here the relative merits of the two types, but simply to give a comparison of the illuminating effect or candlepower of several kinds of ordinary safety lamps used for general work and for testing. Few data have been published in recent years on this subject and because of this it is hoped that the following information may be of some interest and value.

The lamps tested were in all cases of standard size and in good condition, all practically new, and were selected from a collection of about 100 with a view to obtaining results on as many different common types as possible. With the exception of the gasoline-burning lamps, five different oils were used in each lamp to determine whether particular oils were suited to particular lamps and whether there was any great difference in the illuminating power of the different oils, or whether some oils would maintain their illuminating power better than others throughout a working period of 8 hr. No attempt was made to determine the amount or cost of oil burned by each lamp, but the condition of the lamp, wick, glass, and gauzes was noted after each test.

## *Description of Apparatus Used*

The candlepower was determined with a Weber photometer, Fig. 1, which consists of a tube *A* containing a movable milk-glass screen adjustable by the milled screw *B*. The distance of this screen from the standard light at *C* is read on the scale attached to *A*. The standard light consists of a benzine lamp so arranged that the height of the flame may be adjusted exactly by observing it against a scale in the lamp box through an opening in the front. The lamp to be tested is placed at *D* so as to illuminate a milk-glass screen in the holder at *E*. At the intersection

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of the axes of tube $A$ and sight-tube $F$ are placed two 45° prisms with their surfaces in contact so that the light from $D$ may pass through without deflection while that from $C$ is reflected at the contact in a direction parallel to that from $D$. The faces of the prisms are blackened around an elliptical field and on this field is an annular silvered ring which reflects the light from $C$. When viewed through the telescopic eyepiece, $G$, focused on the prisms, on outer annular ring and inner elliptical surface are seen illuminated by light $D$ and an inner ring illuminated by that from $C$. With $D$ in a fixed position, the milk-glass screen in $A$ is adjusted until an evenly illuminated field is observed through the eyepiece. Then the

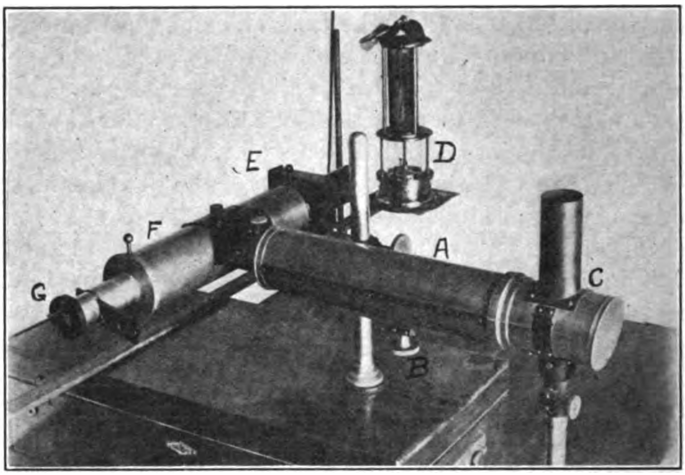


FIG. 1.—WEBER PHOTOMETER USED IN DETERMINING CANDLEPOWER OF LAMPS.

screens in $A$ and $E$ are illuminated to the same intensity and the intensities of the standard and unknown light are to each other as the squares of their distances from the respective screens, or expressed as a formula

$$i = \frac{R^2}{r^2} C$$

in which $i$ = candlepower or intensity of light being tested,
$R$ = distance of $D$ (light being tested) from screen $E$,
$r$ = distance of $C$ (standard light) from screen in tube $A$,
$C$ = constant involving the candlepower of the standard lamp, relative opacity of the two screens and characteristics of the apparatus.

*Determination of the constant* $C$ was accomplished by placing a standard candle at $D$. The value of $C$ is then given by writing the formula

$$C = \frac{r^2}{R^2}$$

as the intensity $i$ is then unity. About 50 readings were taken using standard spermaceti candles. Several candles were employed and their illuminating power checked by weighing before and after use and determining the consumption of spermaceti per hour. A standard candle is supposed to burn 120 grains per hour. The candles used consumed from 123 to 124 grains per hour equivalent to a candlepower of 1.03. The average value obtained of the constant $C$ was 0.36. The candle was placed at the same distance as the lamps tested and also at greater and less distances.

*The specific gravity of the oils* was determined with a Westphal balance, and the Baumé degrees calculated.

### *Description of the Lamps*

The lamps tested were selected to give a wide range in variety and were of standard make and size (Figs. 2, 3 and 4). Before beginning each test with the several oils they were all thoroughly cleaned, the oil vessels rinsed out with gasoline and dried and fresh wicks put in. The shape and size of wicks, number and dimensions of gauzes and oils used are given in Table 1.

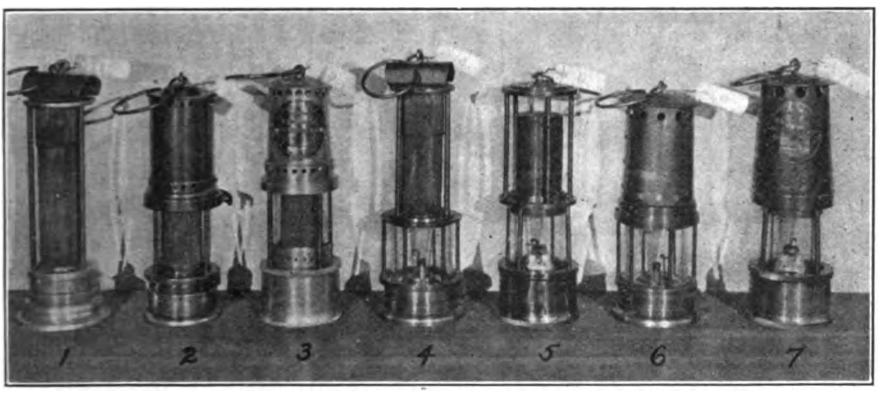


FIG. 2.—SAFETY LAMPS NOS. 1 TO 7 TESTED FOR ILLUMINATING POWER.

Lamp No. 1 is an ordinary Davy with aluminum frame.

Lamp No. 2 is a Davy lamp with bonnet and glass shield surrounding the lower portion of the gauze but which could be lifted when testing. It is also known as the Hughes-Evan Thomas lamp. Normally it takes its air through openings in the base ring covered on the inside by the lower end of the gauze.

Lamp No. 3 is of the same type but with aluminum oil vessel, frame, and bonnet. Also it has a Hailwood porcelain flat wick burner.

TABLE 1.—*Candlepower of Safety Lamps*

| Number | Type | Location of Air Inlet | No. of Gauzes | Size of Gauzes, Inches | Bonnet | Size of Wick Tube, Inches R = round F = flat | Kind of Oil | Candlepower, Freshly Filled and Clean | Candlepower at End of 4 Hr. | Candlepower at End of 8 Hr. | Average Candlepower | Condition of Lamp at End of 8 Hr. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Common-Davy... | Entire gauze unprotected. | 1 | 1½×5½ | No | ¼ R | Eureka.......... | 0.12 | 0.16 | 0.12 | 0.13 | Gauze fairly clean, wick in good shape. |
| | | | | | | | Mineral Seal .... | 0.10 | 0.09 | 0.11 | 0.10 | Top of gauze smoked. |
| | | | | | | | Sperm.......... | 0.15 | 0.11 | 0.14 | 0.13 | Required frequent attention and snuffing. |
| | | | | | | | Lard and kerosene | 0.14 | 0.13 | 0.10 | 0.12 | Good. |
| 2 | Hughes-Evan Thomas. | Below glass shield.... | 1 | 1½×5⅜ | Yes | ¼ R | Eureka.......... | 0.12 | 0.11 | 0.11 | 0.11 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.12 | 0.09 | 0.11 | 0.11 | Top of gauze and cap smoked. |
| | | | | | | | Sperm.......... | 0.10 | 0.11 | 0.13 | 0.11 | Required frequent attention and snuffing to keep burning. |
| | | | | | | | Lard and kerosene | 0.14 | 0.10 | 0.12 | 0.12 | Good. |
| 3 | Hughes-Evan Thomas. | Below glass shield.... | 1 | 1⅜ top 1½ bottom ×4½ | Yes | 3/32×½ F | Eureka.......... | 0.16 | 0.18 | 0.15 | 0.16 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.16 | 0.11 | 0.12 | 0.13 | Gauze clean. |
| | | | | | | | Sperm.......... | 0.15 | 0.15 | 0.18 | 0.16 | Same as No. 2, gauze clean. |
| | | | | | | | Lard and kerosene | 0.17 | 0.09 | 0.13 | 0.13 | Good. |
| 4 | Clanny.......... | Base of gauze....... | 1 | 1⅝×4¼ | No | 9/32 R | Eureka.......... | 0.34 | 0.35 | 0.43 | 0.37 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.32 | 0.30 | 0.33 | 0.32 | Gauze clean. |
| | | | | | | | Sperm.......... | 0.27 | 0.24 | 0.25 | 0.25 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.47 | 0.41 | 0.33 | 0.40 | Good. |
| 5 | Clanny, but with perforated sheet copper gauze. | Base of gauze....... | 1 | 1⅞×3 | No | 3/32×½ F | Eureka.......... | 0.34 | 0.42 | 0.33 | 0.36 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.42 | 0.35 | 0.36 | 0.38 | Gauze clean. |
| | | | | | | | Sperm.......... | 0.37 | 0.28 | 0.37 | 0.34 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.48 | 0.28 | 0.25 | 0.34 | Good. |
| 6 | Evan Thomas No. 7. | Base of bonnet...... | 1 | 1 9/16 top 1 15/16 bottom ×3¼ | Yes | 9/32 R | Eureka.......... | 0.38 | 0.38 | 0.31 | 0.36 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.38 | 0.36 | 0.40 | 0.38 | Top of gauze and cap slightly choked. |
| | | | | | | | Sperm.......... | 0.30 | 0.21 | 0.32 | 0.28 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.46 | 0.41 | 0.34 | 0.40 | Good. |
| 7 | Marsaut......... | Base of bonnet...... | 2 | 1 13/16 & 1 15/16×3½ 1 9/16 & 1 11/16×3 | Yes | 3/32×½ F | Eureka.......... | 0.35 | 0.39 | 0.28 | 0.34 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.40 | 0.31 | 0.35 | 0.35 | Gauzes clean. |
| | | | | | | | Sperm.......... | 0.45 | 0.37 | 0.38 | 0.40 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.43 | 0.37 | 0.32 | 0.37 | Good. |
| 8 | Clanny, bonneted. | Base of bonnet....... | 1 | 1⅝ top 1 15/16 bottom ×3 | Yes | 5/16 R | Eureka.......... | 0.35 | 0.35 | 0.36 | 0.35 | Same as No. 1. |
| | | | | | | | Mineral Seal .... | 0.35 | 0.31 | 0.32 | 0.33 | Top of gauze choked. |
| | | | | | | | Sperm.......... | 0.30 | 0.29 | 0.39 | 0.33 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.40 | 0.34 | 0.30 | 0.35 | Good. |

TABLE 1.—(*Continued*).

| | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 9 | Marsaut with Stokes shut-off. | Base of bonnet, or lower holes at top of bonnet. | 2 | 1 3/4 × 2 × 3 1/2<br>1 9/16 × 1 11/16 × 3 | Yes | 3/32 × 1/2 F | Eureka | 0.32 | 0.31 | 0.31 | 0.31 | Same as No. 1. |
| | | | | | | | Mineral Seal | 0.36 | 0.34 | 0.35 | 0.35 | Inner gauze slightly choked. |
| | | | | | | | Sperm | 0.30 | 0.30 | 0.36 | 0.32 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.43 | 0.24 | 0.32 | 0.33 | Good. |
| 10 | Beard-Deputy | Below flame | 2 | 1 1/2 & 1 11/16 × 3 1/3<br>1 1/4 & 1 3/4 × 3 3/4 | Yes | 5/16 R | Eureka | 0.26 | 0.32 | 0.33 | 0.30 | Same as No. 1. |
| | | | | | | | Mineral Seal | 0.33 | 0.31 | 0.35 | 0.33 | Gauzes clean. |
| | | | | | | | Sperm | 0.21 | 0.27 | 0.16 | 0.21 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.40 | 0.29 | 0.25 | 0.31 | Good. |
| 10a | | | | | No | | Lard and kerosene | 0.41 | 0.31 | 0.34 | 0.35 | Good. |
| 11 | Special | Below flame or through base of bonnet. | 1 | 1 9/16 top<br>1 11/16 bottom<br>× 2 1/2 | Yes | 3/32 × 1/2 F | Eureka | 0.38 | 0.36 | 0.35 | 0.36 | Same as No. 1. |
| | | | | | | | Mineral Seal | 0.47 | 0.46 | 0.49 | 0.47 | Good. |
| | | | | | | | Sperm | 0.32 | 0.23 | 0.33 | 0.29 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.49 | 0.47 | 0.35 | 0.44 | Good. |
| 12 | Ashworth-Hepplewhite-Gray | Below flame through top or bottom of air tubes. | 1 | 1 top<br>1 1/8 bottom<br>× 2 3/4 | Chimney | 1/4 R | Eureka | 0.40 | 0.32 | 0.34 | 0.35 | Same as No. 1. |
| | | | | | | | Mineral Seal | 0.34 | 0.10 | 0.12 | 0.19 | After 4 and 8 hr. glass very smoky, lamp had to be opened and cleaned. Top and one side of gauze choked. |
| | | | | | | | Sperm | 0.26 | 0.30 | 0.26 | 0.27 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.44 | 0.26 | 0.26 | 0.32 | Fair. |
| 13 | A-H-G Improved, 6 air tubes. | Below flame through top or bottom of air tubes. | 1 | 1 top<br>1 3/16 bottom<br>× 2 3/8 | Chimney | 1/8 × 1/2 F | Eureka | 0.57 | 0.47 | 0.50 | 0.51 | Same as No. 1. |
| | | | | | | | Mineral Seal | 0.49 | 0.50 | 0.55 | 0.51 | Gauze clean. |
| | | | | | | | Sperm | 0.51 | 0.56 | 0.54 | 0.54 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.66 | 0.44 | 0.49 | 0.53 | Good. |
| 14 | A-H-G Improved, 1 large air tube. | Below flame through top or bottom of air tubes; top of tube opens into bonnet. | 2 | 1 9/16 × 2 1/4<br>1 5/16 × 2 | Bonnet | 5/8 × 3/32 F | Eureka | 0.43 | 0.36 | 0.40 | 0.40 | Glass smoked. |
| | | | | | | | Mineral Seal | 0.52 | 0.44 | 0.47 | 0.48 | Glass smoked, gauze slightly choked. |
| | | | | | | | Sperm | 0.52 | 0.46 | 0.45 | 0.48 | Same as No. 3. |
| | | | | | | | Lard and kerosene | 0.53 | 0.48 | 0.47 | 0.49 | Good. |
| 15 | Koehler | Below flame | 2 | 2 & 2 3/16 × 3 11/16<br>1 3/8 & 1 11/16 × 3 13/16 | Bonnet | 9/32 R | Gasoline 72° Bé. | 0.64 | 0.65 | 0.60 | 0.63 | Gauzes clean. |
| 17 | Koehler | Below flame | 2 | 2 & 2 3/16 × 3 11/16<br>1 3/8 & 1 11/16 × 3 13/16 | Bonnet | 3/32 × 5/8 F | Gasoline 72° Bé. | 0.66 | 0.64 | 0.66 | 0.65 | Gauzes clean. |
| 18 | Cremer | Below flame | 2 | 1 3/4 & 2 × 3 1/2<br>1 1/4 & 1 9/16 × 3 1/2 | Bonnet | 1/16 × 5/8 F | Gasoline 72° Bé. | 0.72 | 0.70 | 0.67 | 0.70 | Gauzes clean. |
| 19 | Wolf | Below flame | 2 | 1 9/16 & 1 7/8 × 3 5/8<br>1 5/16 & 1 5/8 × 3 1/2 | Bonnet | 9/32 R | Gasoline 72° Bé. | 0.59 | 0.60 | 0.57 | 0.59 | Gauzes clean. |
| 20 | Wolf | Below flame | 2 | 1 1/2 & 1 3/4 × 3 5/8<br>1 1/4 & 1 5/8 × 3 7/16 | Bonnet | 3/32 × 5/8 F | Gasoline 72° Bé. | 0.64 | 0.59 | 0.61 | 0.61 | Gauzes clean. |

Lamp No. 4 is an ordinary improved Clanny unbonneted with brass oil vessel and frame.

Lamp No. 5 is constructed on the principle of a Clanny, unbonneted, but the gauze consists of perforated sheet copper instead of woven wire.

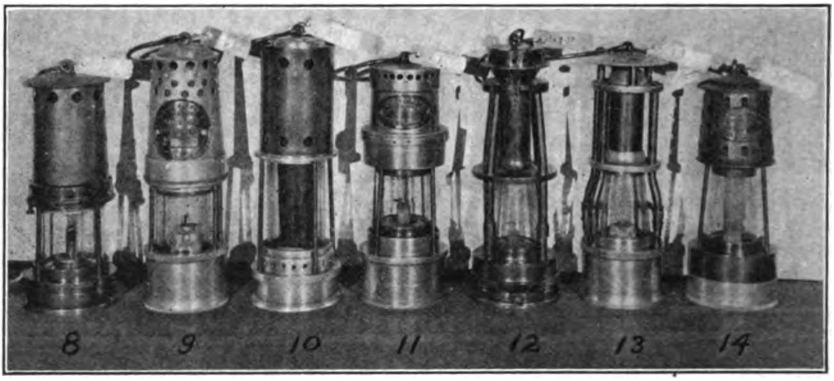


FIG. 3.—LAMPS NOS. 8 TO 14 TESTED FOR ILLUMINATING POWER.

It is also equipped with a Hailwood porcelain flat burner and brass oil vessel and frame.

Lamp No. 6 is a bonneted Clanny very similar to the lamp known as the Evan Thomas No. 7. The air is admitted through openings in the

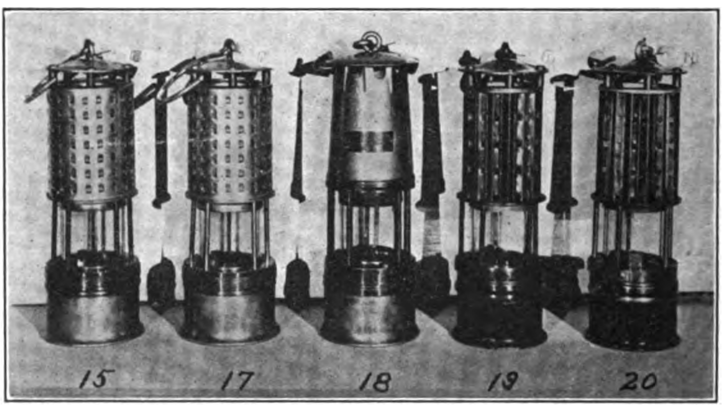


FIG. 4.—GASOLINE-BURNING LAMPS TESTED.

lower part of the bonnet, and a deflecting ring is placed between these openings and the gauze, requiring the air to pass upward and over this deflector. The oil vessel and frame is brass and bonnet is of sheet iron.

Lamp No. 7 is a standard Hailwood working lamp constructed on the Marsaut principle with two gauzes. Oil vessel and frame of brass, bonnet of sheet iron, and porcelain flat-wick burner.

Lamp No. 8 is a bonneted Clanny with smoke gauze.

Lamp No. 9 is a Marsaut type with Stokes shut-off ring at base of bonnet by which air is admitted at that point or, when closed, forced to enter the lower set of holes near the top of bonnet and pass down behind a deflecting ring extending from just above the inlet openings to about the middle of the bonnet. The lamp is fitted with Hailwood flat-wick porcelain burner.

Lamp No. 10 is a Beard Deputy with Beard-Mackie sight indicator. The circulation of air is on the Eloin principle, the air entering through gauze-covered openings around the base below the flame.

Lamp No. 11 is a testing lamp of English manufacture. By means of a sliding ring around the base the air may enter gauze-covered openings below the flame, or, when these are closed, it enters through openings in the lower part of the bonnet passing through the lower part of the gauze as in the ordinary Clanny lamp.

Lamp No. 12 is an ordinary Ashworth-Hepplewhite-Gray.

Lamp No. 13 is an improved Ashworth-Hepplewhite-Gray with Hailwood porcelain flat-wick burner and six hollow inlet tubes from top to bottom, each with an opening at the bottom, instead of four inlet tubes as in the ordinary form No. 12.

Lamp No. 14 is an improved Gray lamp with brass flat-wick burner. A single hollow standard of large size connects the gauze-covered air inlet in the base with the interior of the bonnet. This standard has an opening near the bottom with sliding sleeve to cover it. The gauze is of much larger diameter than those of Nos. 12 and 13.

Lamps No. 15 and 17 are respectively round and flat-wick Koehler's, burning gasoline.

Lamp No. 18 is a flat-wick Cremer, burning gasoline, differing from the Koehler and Wolf in the construction of the bonnet. The air enters through slots in the circumference of the base of the bonnet and not through perforations along the sides. It has a corrugated expansion ring between the base of bonnet and flange of the gauzes similar to the Koehler.

Lamps No. 19 and 20 are respectively standard round- and flat-wick Wolf's.

The five gasoline lamps are all fitted with Pyro igniters.

The illuminating power of some of the types tested as quoted by other authorities is given in Table 2. A wide variation in some cases will be noted. It may be proper to mention at this point that some of the results obtained were checked by testing the same lamps with a Bunsen type photometer, the results agreeing closely. The greatest

difference will be noted in Nos. 10 and 12. No. 10 was an aluminum Beard-Deputy with Beard-Mackie sight indicator. To check results on this, No. 10*a* was tested with one oil. No. 10*a* lamp was identical except that it was made of brass, the indicator was omitted, and it did not have a bonnet. The results on No. 12, A-H-G lamp, were checked with a duplicate lamp, the results agreeing within 5 per cent. The good lighting power of No. 13 over No. 12 is undoubtedly due to the better air supply of No. 13. As noted in Table 1, this lamp has six air tubes each with an air hole at the base, whereas No. 12 has four tubes with air inlets near the base in two only. A considerable difference will also be noticed between the observed candlepower of the Wolf type lamps and the generally accepted candlepower with a value of 1.

*Table 2.—Illuminating Power of Certain Lamps According to Different Authorities*

(Expressed in Candlepower)

| Type | Authority | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | Marsaut | Beard | Coal & Metal Miner's Pocket Book | Mines Accident Commission | Crossley | J. A. Longden | Prussian Fire Damp Com. | Lupton, Parr & Perkins | Average |
| Davy (common) | 0.18 | 0.15 | 0.16 | 0.126–0.15 | 0.125 | 0.10 | ..... | ..... | 0.141 |
| Clanny (various types) | . | 0.25–0.50 | 0.20 | .... | ... | .. | .... | .... | 0.317 |
| Clanny, unbonneted | 0.47 | ..... | .... | 0.50 | 0.23 | 0.21 | 0.62 | .... | 0.416 |
| Clanny, bonneted | .. | ..... | .... | 0.31–0.64 | ... | 0.25 | .... | .... | 0.40 |
| Evan Thomas No. 7 | .. | 0.40 | 0.45 | 0.46 | ... | .. | .... | .... | 0.437 |
| Marsaut (2 gauze) | .. | 0.60 | 0.55 | .... | ... | .. | .... | .... | 0.575 |
| Ashworth-Hepplewhite-Gray | .. | 0.65–0.79 | .... | .... | 0.65 | .. | .... | 1.00 | 0.77 |
| Beard, Eloin-Marsaut (2 gauze) | .. | 0.75 | .... | . . | ... | .. | .... | .... | 0.75 |
| Wolf | .. | 0.80–1.00 | 0.90 | .... | ... | .. | 0.66 | 0.40 | 0.75 |

## *Conditions of Test*

After being cleaned, filled, and lighted, the flame of each lamp was adjusted to a normal height suitable for working conditions, as near as it was possible to judge. This was ordinarily done by turning up the flame until it would begin to smoke and then slowly lowering it until it stopped smoking. The lamps were then allowed to stand 20 to 30 min.

or until all parts were thoroughly warmed up, before the first readings were taken. The flames were again adjusted just before the readings were made. They were also maintained at approximately a working height throughout the time of 8 hr. and carefully adjusted at each observation.

### *Illuminants*

The oils used were Eureka Safety Lamp oil, Mineral Sea oil, pure sperm oil, pure lard oil, a mixture of lard oil and kerosene, and gasoline. The first two are mixed oils sold on the market under the trade names given. Eureka Safety Lamp oil is apparently a mixture of lard or cotton seed, possibly both, and a petroleum oil. Mineral Seal is a mixture of fish or sperm oil and petroleum oil with possibly other ingredients. The mixture of lard oil and kerosene was prepared for the purpose of this test and was composed of 50 per cent. lard oil and 50 per cent. kerosene by volume. Pure lard oil was tried in all the lamps, but even after they were warmed up such constant attention was required to keep them burning properly that any results obtained would have been of no practical value.

The specific gravities of the oils and the corresponding gravity on the Baumé scale are given in Table 3.

TABLE 3.—*Specific Gravity of Safety-Lamp Oils at 18° C.*

| Oil | Specific Gravity, Water = 1.00 | Degrees, Baumé |
|---|---|---|
| Eureka Safety Lamp | 0.863 | 32 |
| Mineral Seal | 0.818 | 41 |
| Sperm | 0.900 | 25 |
| Lard | 0.913 | 23 |
| Lard 50 per cent.; kerosene 50 per cent. | 0.860 | 32 |
| Gasoline | 0.695 | 72 |

### *Conclusions*

The average candlepower for the different oils indicates clearly the advantage of a mixed oil or the addition of petroleum oil. The average illuminating powers of lamps Nos. 1 to 14 for the different oils, excepting gasoline, are given in Table 4.

TABLE 4.—*Average Illuminating Power of Different Oils*

| Oil | Average Candlepower | Specific Gravity of Oil |
|---|---|---|
| 50 per cent. lard, 50 per cent. kerosene | 0.333 | 0.860 |
| Eureka Safety Lamp | 0.318 | 0.863 |
| Mineral Seal | 0.316 | 0.818 |
| Sperm | 0.296 | 0.900 |

There is apparently no definite variation of candlepower with the specific gravity of the oil, for although sperm oil is the heaviest and has the lowest average illuminating power, Mineral Seal, the lightest, is intermediate in illuminating power between sperm and the other two heavier oils.

In addition to better lighting, the mixed oils burned more freely and required less attention to the wick and flame, but showed a slightly greater inclination to smoke than the pure sperm oil. The better illuminating power of the mixed oils is especially noticeable in Lamps Nos. 4, 6, 10, and 11. The effect of a bonnet in decreasing illuminating power is indicated to some extent in Lamps Nos. 10 and 10*a*, identical except for a bonnet. With the mixed lard and kerosene a difference of 0.04 candlepower is noted.

The effect of admitting the air below the flame or even at the base of the bonnet, instead of higher up, is shown conclusively in lamps Nos. 9 and 11, constructed so that air could be admitted in different ways. In No. 9, the results given in Table 1 were obtained with air entering at base of bonnet; the same lamp with air entering near the top of the bonnet and passing downward between bonnet and deflecting ring gave an average candlepower of 0.24 with Eureka oil, 0.07 less than the value given in Table 1. With No. 11 the results are even more pronounced. With the inlet at the bottom closed, and air entering at base of bonnet and gauze, the average candlepower with Eureka oil was 0.23, while with air entering below the flame it was 0.36, a difference of 0.13 or 36 per cent.

In testing the lamps with flat wicks, the candlepower was in all cases taken in a line normal to the width of the flame. The decrease in lighting edgewise to the flame was much less than anticipated. For the oil lamps the candlepower edgewise varied from 5 to 20 per cent. less with an average of 10.5 per cent. For the gasoline lamps the illumination edgewise was 4 to 7 per cent. less.

With the exception of gasoline, the decrease of candlepower during the 8 hr. seems to vary inversely as the illuminating power of the oil. Sperm oil gave an average candlepower for 14 lamps when freshly lit of 0.301 and after 8 hr. of 0.304, a gain of 1 per cent.; Eureka oil gave at the beginning an average of 0.323 cp. and after 8 hr. 0.308 cp., a decrease of 4.3 per cent.; Mineral Seal at beginning an average of 0.340 cp. and after 8 hr. 0.314 cp., a decrease of 7 per cent.; a mixture of 50 per cent. lard and 50 per cent. kerosene at the beginning gave an average of 0.403 cp. and after 8 hr. 0.296 cp., a decrease of 26.5 per cent. To determine whether the decrease in candlepower was due to the lighter part of the oil burning off first, leaving a heavier residue, the specific gravity of the oil remaining in the lamps after burning the mixture of kerosene and lard oil was determined. This was found to be 0.865 or 0.005 greater

than the original oil. As this is only 0.58 per cent. gain, it is within a possible experimental error, and it would be hardly safe to assume from it that there was a separation of the oils. It would be more reasonable to conclude that the decrease in illumination was due to the more rapid deposit of soot on the gauze and a film of unburned carbon on the glass when burning oils containing petroleum products, than when burning pure sperm oil.

With the gasoline lamps, the illuminating power decreased in 8 hr. from an average of 0.65 to 0.62 cp. or 4.6 per cent.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Iron Pyrites Deposits in Southeastern Ontario, Canada

BY P. E. HOPKINS,* B. A., B. S., TORONTO, ONT.

(Arizona Meeting, September, 1916)

## *Introduction and History*

In speaking of the economic geology of southeastern Ontario, W. G. Miller and C. W. Knight[1] say that "there occurs in southeastern Ontario a variety of minerals and rocks of economic value, probably as great as in any district of like size on the North American continent. Some of these deposits, including marble and trap, are inexhaustible. Others, including talc and iron pyrites, have proved to be of considerable economic importance. From time to time, during the last 50 years, the following minerals and rocks have been mined or quarried with varying success: gold, iron pyrites, zinc blende, copper pyrites, galena, mispickel, magnetite, hematite, talc, actinolite, mica, marble, ophicalcite, feldspar, fluorite, apatite, corundum, graphite and sodalite. All of the economic materials, with the exception of fluorite, appear to be of pre-Cambrian age. The fluorite veins penetrate the Ordovician, Black River, limestone."

Accompanying that report was an article by the writer on the Queensboro Pyrite Area which includes one of the two working pyrite properties in southeastern Ontario.

In the present paper will be given a brief description[2] of all the known pyrite deposits in the area which may at some time possess an economic value, with fuller descriptions of the two working mines—The Canadian Sulphur Ore Co.'s mine near Queensboro, and the Nichols Chemical Co.'s property at Sulphide.

The earliest mining of iron pyrites in Ontario was done in 1868 on the Billings property near Brockville. The mines were closed down in 1879 under the assumption that they were exhausted. Many other

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[1] The Pre-Cambrian Geology of Southeastern Ontario, *Annual Report, Ontario Bureau of Mines*, vol. 22, part 2 (1913).

[2] The information regarding the various pyrite prospects in southeastern Ontario is summarized from E. L. Fraleck's comprehensive report on Iron Pyrites in Ontario, *Annual Report, Ontario Bureau of Mines*, vol. 16, part I, pp. 149–201 (1907).

pyrite deposits have been worked for gold, iron or copper at some time. The steady pyrite industry of the Province began in 1900 when ore from the Bannockburn mine was produced. Mines in Hastings County have been steady producers since that time. An acid-making plant has been in operation at Sulphide since 1907 by the Nichols Chemical Co. for the

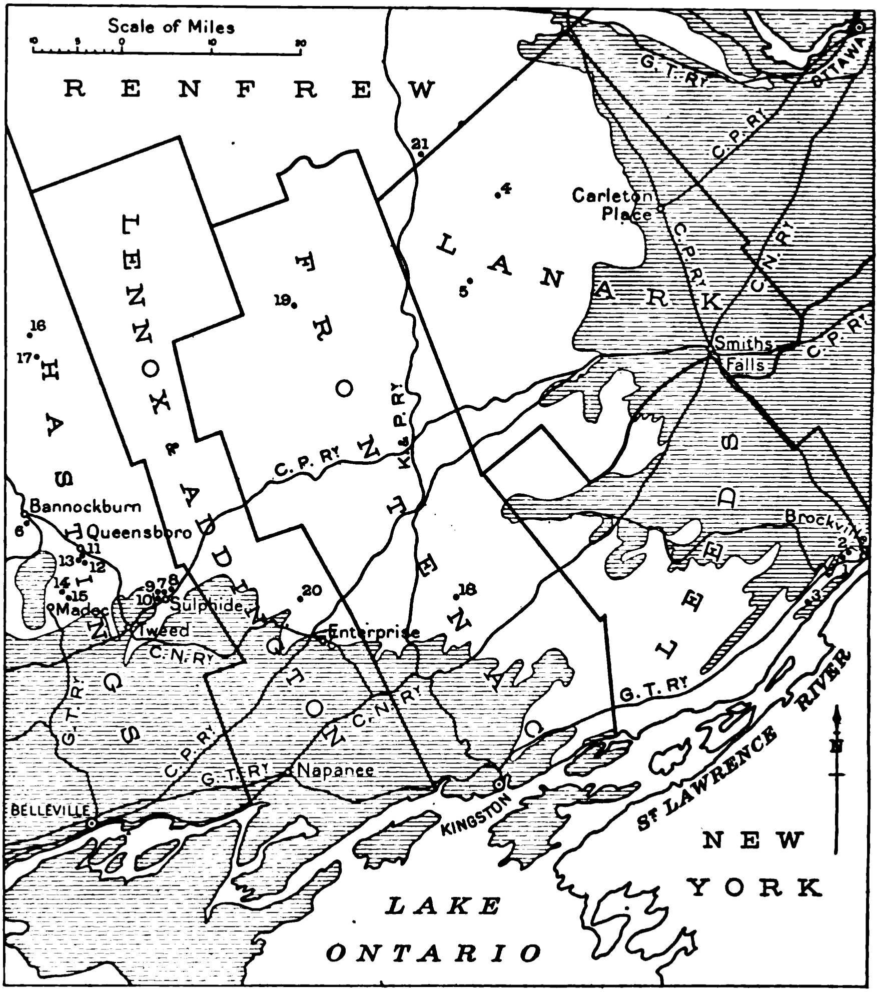


FIG. 1.—SKETCH MAP SHOWING PRE-CAMBRIAN AND PALEOZOIC (HATCHED) AREAS IN SOUTHEASTERN ONTARIO. THE NUMBERS INDICATE THE LOCATIONS OF PROPERTIES DESCRIBED.

treatment of its ore at Sulphide. The company also buys the ore mined from other properties in the neighborhood. Another plant for treating custom ore is operated by the Grasseli Chemical Co. at Hamilton. These two plants treat the bulk of the eastern Ontario production, the remainder being shipped to the United States.

Recently a large percentage of the production has been coming from the Vermilion Lake deposits[3] in northwestern Ontario, the ore being shipped to United States ports on the great lakes. Another property, the Goudreau Lake deposits,[4] has been recently developed and expects to

TABLE 1.—*List Showing the Locations of Pyrite Deposits in Southeastern Ontario*[5]

*Brockville Section.*

† 1. Brockville Chemical Co. (Billings property); lot 19, con. 2, Elizabethtown township.

† 2. Sloan prospect; lot 18, con. 2, Elizabethtown township.

3. Shipman prospect; about 6 miles west of the Billings (No. 1).

*Lanark County.*

† 4. McIlwraith mine; lot 5, con. 4, Darling township.

5. Ladore prospect; lot 19; con. 7, Dalhousie township.

*Hastings County.*

‡† 6. Bannockburn (or Jarman) mine; lot 25, con. 6, Madoc township.

* 7. Hungerford mine (Nichols Chemical Co.); lot 23, con. 12, Hungerford township.

† 8. Canada mine (formerly Oliver Prospect); lot 26, con. 12, Hungerford township.

9. Hungerford Western Extension; parts of lots 21 and 22, con. 12, Hungerford township.

† 10. Ontario Sulphur Mines, Ltd.; northwest quarter of east half of lot 21, con 12, Hungerford township.

† 11. Queensboro mine; lot 11, con. 11, Madoc township.

* 12. Canadian Sulphur Ore Co. (formerly Wellington prospect) N. ½ lot 9, con 10, Madoc township.

† 13. Davis or Palmer prospect; lot 10, con. 10, Madoc township.

14. Farrell prospect; 2 miles northeast of Madoc village.

‡ 15. McKenty prospect; 2 miles east of Madoc village.

16. Little Salmon Lake deposit; lot 23, con. 7, Cashel township.

17. Gunter property; lot 23, con. 4, Cashel township.

*Other Eastern Ontario Prospects.*

18. Snooks prospect; lot 7, con. 14, Loughborough township, Frontenac Co.

19. Stalker prospect; lot 42, con. 6, Clarendon township, Frontenac Co.

20. Foley prospect; 5½ miles north of Enterprise Sta., Lennox, Addington Co.

21. Caldwell prospect; lot 1 con. 1, Blithfield township, Renfrew Co.

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* Mines now working (April, 1916) and shipping pyrites.

† Properties which have shipped pyrites.

‡ Properties which have shipped hematite or limonite.

[3] E. S. Moore: Vermilion Lake Pyrite Deposits, *Annual Report, Ontario Bureau of Mines*, vol. 20, part I, pp. 199–209 (1911).

T. F. Sutherland: Northern Pyrites Company, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, pp. 94–95 (1915).

[4] A. L. Parsons: Goudreau Pyrite Claims, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, p. 211 (1915).

T. F. Sutherland: Madoc Mining Company, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, p. 107 (1915).

[5] The number of each property refers to the corresponding number showing its position on the accompanying map.

commence at once supplying large tonnages. The Helen mine,[6] operated by the Algoma Steel Corporation, produces some pyrite which is treated in its plant at Sault Ste. Marie.

The iron pyrites resources of Ontario are of considerable extent and value, in the last 15 years 538,755 tons, worth $1,438,122, having been produced, the greater part coming from southeastern Ontario. During the coming years there will undoubtedly be a steady increase in production. The war has had a stimulating effect on the demand of the United States for pyrite from Ontario.

### *Brockville Section*

*The Brockville Chemical Co., No.* 1,[7] began mining for pyrite on the Billings property in 1868. The ore occurred in a series of lenses conformable to the lamination of a highly foliated pink granite gneiss. The lenses, which consist of pyrite and calcite in parallel lines, strike northeast and dip to the southeast. The richer shoots of ore were gouged out and no timbering was done. The main pit was sunk 250 ft. The ore was used for making acids in Brockville, the sulphuric and mixed acids being used at the fertilizer and dynamite works in and near Brockville. Operations of all kinds ceased in 1880. The evidence of the men who worked in the old pits is to the effect that they were never completely exhausted.

*Sloan Prospect, No.* 2.—A band of gossan strikes in a northeast direction across the property and dips to the southeast. The 20-ft. inclined shaft passes through 6 or 8 ft. of gossan. There is a width of 3 ft. of solid pyrites on the foot wall, the remainder of the shaft being in alternating bands of pyrite and crystallized calcite in equal amounts. Eighty tons of ore, running 40 per cent. sulphur, were shipped to Buffalo and Capelton.

The Buffalo-Brockville Mining Co. shipped a small tonnage from this lot during 1911 and 1912.

*Shipman Prospect, No.* 3.—The pyrite, which is much intermixed with pyrrhotite and country rock (gneiss), has been mined from an irregular pit 40 ft. long and 30 ft. wide.

### *Lanark County*

*McIlwraith Mine, No.* 4.—The deposit, which is covered by 14 ft. of gossan, strikes north of east along a contact between diorite on the

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[6] A. L. Parsons: Helen Mine, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, pp. 202–205 (1915).

[7] The numbers following mention of the pyrite properties refer to corresponding numbers showing their positions on the accompanying map, and in Table 1.

south and crystalline limestone on the north, and dips 60° to the south. It was first opened for gold. In 1899 and 1900 the shaft was deepened to 75 ft. and a 150-ft. tunnel run along the strike of the deposit, disclosing a length of over 90 ft. of clean high-grade pyrite inclosing lenses of quartz. A 12-ft. crosscut to the south did not pierce the width of the deposit. Three carloads of ore were shipped. Samples from the dump and tunnel, by E. L. Fraleck, gave 38.86 and 42.60 per cent. of sulphur respectively.

*Ladore Prospect, No.* 5.—A heavy fahlband strikes north of east along the contact of a coarse amphibolite and a fine-grained gray granite. The trenches and shallow pits expose a gossan in the form of bog iron ore, but pyrite in quantity was not located. The fahlband continues into the adjoining lot to the east along a contact of crystalline limestone and granite.

*Bannockburn Mine, No.* 6.—In 1898, the property was opened as an iron mine, 11 car loads of limonite, running about 38 per cent. in iron and low in sulphur, having been shipped. This ore was merely a gossan 8 to 15 ft. deep which capped iron pyrites deposits. The pyrites occurred as two lenses at right angles to each other, but conforming in strike and dip with the inclosing rock, a chloritic schist. Limestone covers the apex of the fold of the lenses. The south lens, which is 160 ft. long and 8 to 15 ft. wide, was mined to a depth of 275 ft. During the 6 years of operation about 580 tons of pyrite per month were shipped, all of which went to the General Chemical Co. at Buffalo. The ore did not fall off either in grade or quantity with depth, but, owing to the hazard of open-pit mining, operations were abandoned in August, 1906.

*Hungerford Mine, No.* 7.—This property was opened 40 years ago as a gold property, and a smelter was erected to extract gold from the barren pyrite. The Nichols Chemical Co.[8] re-opened the mine in June, 1903. Owing to some difficulty about the title, the mine was closed down in August, 1904, but operations were resumed in August, 1905, and have since been continuous. Since 1907 acid works have been in operation for the treatment of company ores, and other ores in the vicinity.

Passing through this property, and extending beyond, is a large fahlband striking 25° north of east and traceable for 2 miles. Level farm land to the south is underlain by garnetiferous crystalline schist cut by massive diorite, into which, 500 yd. north of the deposits, has been intruded a pink hornblende granite that rises above the country in a series of rugged hills, locally called the Bald Mountains. The granite has protected the deposits from denudation. The deposits are strung along the contact of the diorite and the schist, the strike of lenses, contact, fahlband, and schist being identical.

The pyrite occurs in three parallel deposits striking with the schist

[8] W. H. Nichols, President, 25 Broad St., New York.

and dipping 60° to the south. The middle one, which does not outcrop on the surface, lies 85 ft. from the south vein and 45 ft. from the north deposit. The north deposit, upon which most of the work has been done, varies in width from 6 to 22 ft. It has been exploited to a length of 620 ft. and to a depth of 575 ft., and the ore still continues. The length as indicated on the surface is about 500 ft. There are now two shafts on the property and about 3,500 ft. of drifting has been done on the orebodies on the six levels.

The ore is coarsely granular and makes a large percentage of fines. The main impurity is calcite, although there is also some quartz present. A small quantity of pyrrhotite occasionally occurs, mainly in the north lode next the foot wall. The average percentage of run of mine ore is about 35 per cent., the fines being much higher.

The acid works have been successfully operated since their completion in July, 1907, and machinery has been installed at various times to increase the capacity and to make new acids. At present sulphuric, hydrochloric, nitric and mixed acids are made by the contact process and shipped in the company's tank cars to various parts of Ontario and Quebec.

Electric power supplied by the Seymour Power and Electric Co. is used throughout the mine and acid works.

*The Canada Mine, No.* 8, which was formerly the Oliver prospect, adjoins the mine operated by the Nichols Copper Co. on the east, and is located on the same fahlband. The lode strikes east and west and dips 50° to the south. During part of 1907, the Canadian Pyrites Co. sank an inclined shaft on the deposit to a depth of 110 ft. and did some drifting on the 85-ft. level, together with some diamond drilling. The deposit varies from 4 to 7 ft. in width. The ore on the dump is pyrite with a little pyrite and pyrrhotite, which will grade upward of 40 per cent. in sulphur.

*The Hungerford Western Extension, No.* 9, was fairly well prospected in 1906 by means of surface trenches at regular intervals along the strike of the fahlband. The western lens had been exploited by surface trenches to a length of 500 ft., exhibiting, near the line between the lots, a width varying from 16 to 18 ft. of ore, which will grade from 42 to 44 per cent. sulphur. The only impurity consists of small included lenses of calcite.

The eastern lenses are presumably continuations of the Hungerford mine orebodies.

A gossan 40 ft. wide occurs on the south end of the property, but not enough work has been done to determine the extent of the deposit.

*The Ontario Sulphur Mines, Limited,*[9] *No.* 10, commenced work in

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[9] Formerly the Craig property.

March, 1908, and continued until the end of 1911, save for 2 months in the summer of 1910. The pyrite deposit on which work has been done is located about ½ mile west of the Hungerford mine. It appears to be a lens pitching toward the southeast. A shaft has been sunk 300 ft., with 225 ft. of drifting on the 100-ft. level and 250-ft. on the 200-ft. level. According to A. W. G. Wilson,[10] "The total shipments from the property up to the first of May, 1911, have been 4,821 long tons of ore averaging 36½ per cent. sulphur." In one place the deposit is 30 ft. wide.

The Sulphide Chemical Co. operated the property from the spring of 1914 until the following November, during which time the mine was dewatered and considerable ore was raised and shipped.[11] No work has been done since.

*The Queensboro Mine (Blakely), No.* 11, up to the autumn of 1906 shipped 65 carloads of pyrites running about 45 per cent. sulphur. Mine operations ceased in 1908. The pyrite occurs as a series of lenses up to 15 and 20 ft. wide along the contact of a garnetiferous schist (Grenville in age) and an intrusive pink felsite (post-Hastings in age). The ore is dense, the only impurity being thin veinlets of quartz. Cutting a pyrite lens is a small quartz vein containing copper pyrites and argentiferous jamesonite. In another place some zinc blende is interbanded with the pyrite. The main shaft is 135 ft. deep with about 175 ft. of drifting on the 50- and 85-ft. levels.

*The Canadian Sulphur Ore Co.'s Pyrites Mine,*[12] *No.* 12, was discovered in 1906 by Stephen Wellington while prospecting for iron. Under the gossan, merchantable iron pyrites was discovered, from which a car load of iron pyrites was shipped in 1908. Later, the Canadian Pyrites Syndicate bought the property, installed a small plant and shipped a few hundred tons of pyrite. In the spring of 1910 the property was handed over to the present company, which began shipping ore 3 months later, and has continued to the present. The mine is equipped to produce 100 tons of iron pyrites per day, yielding 40 per cent. of sulphur. Since Dec. 11, 1912, the mine has been run by electricity supplied by the Seymour Power Co. A branch line 2½ miles in length from the Bay of Quinte Railway near Queensboro to the mine was completed in 1913. The ore is shipped to the Nichols Chemical Co.'s acid plant at Sulphide, 11 miles southeast, and to the chemical companies at Hamilton and Detroit.

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[10] A. W. G. Wilson: Pyrites in Canada, *Publication No.* 167, *Canada Mines Branch,* p. 66 (1912).

[11] A. W. G. Wilson: *Annual Report, Ontario Bureau of Mines* vol. 23, part I, p. 174 (1914).

[12] Alex. Longwell, President, [illegible] ilding, [illegible] For a fuller description see P. E. Hopki[illegible] tes [illegible] *nual Report, Ontario Bureau of Mine*[illegible] 13)

12

The pyrite is mined by underground and open-pit methods. The development work consists of three shafts and two open cuts, with some diamond-drill borings. Nos. 1 and 2 shafts, which are 75 and 100 ft. deep respectively, have been abandoned for some time. The work in late years has been confined to shaft No. 3 and the two open pits. The vertical shaft, No. 3, is 250 ft. deep with about 800 ft. of drifting on the 60-, 120-, and 200-ft. levels. The pyrite deposits are marked by gossan outcrops from 2 to 30 ft. in depth. Beneath are the pyrite deposits, which occur as lenses in contact with rusty schist to the south and white quartzite to the north (both Grenville in age) near an irregular post-Hastings intrusion of gray felsite. The strike of the deposits is slightly north of east, while the dip is almost vertical, inclining slightly to the south. Lenses vary in width up to 25 ft., but horses of country rock are frequently inclosed in the pyrites.

The ore is high grade, very little cobbing, if any, having to be done. Ores have been shipped running 40 to 48 per cent. sulphur.

The deposits are free from impurities such as arsenic, zinc, lead, copper and calcium. The pyrite burns satisfactorily, and is in good demand by sulphuric acid makers.

*The Davis or Palmer Deposit, No.* 13, is in the Grenville limestone. On the surface the pyrite is 2 ft. in width; 9 ft. down there is said to be a deposit 15 ft. wide. A few carloads of pyrites were shipped from a pit sunk on the property.

*The Farrell Deposit, No.* 14, lies in and conforms with the schist which strikes northwest. Test pits for a distance of 200 ft. show either gossan or pyrite. A shaft has been sunk to a depth of about 25 ft. A sample collected by E. L. Fraleck, representing an average of 75 per cent. of the dump (which consists of about 40 tons), yielded 40.64 per cent. of sulphur. The deposit maintains a uniform width of 5 ft., the only impurity being crystalline limestone.

*The McKenty Prospect, No.* 15, shipped hematite 40 years ago. A pit at one time 60 ft. deep has caved in. An examination of the cull dump reveals the fact that all large lumps of apparent hematite have, when broken, a core of pyrites. In E. L. Fraleck's opinion, this is one of many instances throughout eastern Ontario where hematite constitutes the gossan capping of a sulphide orebody.

*The Little Salmon Lake Deposit, No.* 16, occurs in a chlorite schist which strikes east and west, the main rock in the area being a white crystallized limestone, probably of Grenville age. A trench, 40 ft. long, uncovered pyrite 15 ft. in width. An average of 75 per cent. of the pyritiferous material yielded 38.83 per cent. of sulphur.

On the *Gunter Property, No.* 17, a shaft, 20 ft. deep, has been sunk on a deposit consisting of alternating bands of quartz and pyrite 5 ft.

wide. A sample representing two-thirds of the dump yielded 39.50 per cent. of sulphur.

*Snooks Prospect, No.* 17.—A fahlband strikes northeast through a coarse, impure crystalline limestone, and can be traced across the adjoining lot 6 to Desert Lake. On the road allowance, 7 ft. of massive pyrite and 25 ft. of pyrite mixed with crystalline limestone were uncovered in obtaining material for the road.

On the *Stalker Prospect, No.* 19, is a well-defined fahlband, containing some hematite, and striking east and west. A small test pit has been sunk on a lens of pyrite which shows at that point a width of 6 ft.

*The Foley Deposit, No.* 20, occurs in an outlier of crystalline limestone surrounded on all sides at short distances by granite. The irregular deposit consists of small masses of pyrite and pyrrhotite in about equal proportions. The work consists of a pit, 80 ft. long, 40 ft. wide and 10 to 15 ft. deep, sunk on pyrite and pyrrhotite in about equal proportions intermixed with pyroxene, calcite, mica and molybdenite.

*The Caldwell Prospect, No.* 21, was opened in the fall of 1915 by Thomas B. Caldwell of Lanark. About 500 tons of ore have been mined, but the sulphur contents are not known.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Method of Mining Talc

BY F. R. HEWITT, ASHEVILLE, N. C.

(Arizona Meeting, September, 1916)

THE methods of mining talc are simple, and in western North Carolina are almost entirely by open cut and quarry. The larger part of the talc of this section lies in various-sized "veins" inclosed in quartzitic walls, the majority of which have by folding been thrown into a perpendicular position, or nearly so, and the exposed edges of which have been covered by débris from erosion of the mountain above. This covering of loose rocks and earth is from 5 to 25 ft. thick, and in most cases has to be removed because it is difficult to support. Sometimes the covering is heavy enough to "catch" and timber successfully, and in this case the talc can be worked out by the usual method of following the "vein," using stulls for holding up the walls until the vein is worked out, and then allowing the cut or drift to fall in.

In some cases it proves more convenient to sink shafts and run drifts, following the talc "veins" until exhausted. The talc deposits of this section are badly broken and faulted both laterally and perpendicularly, and in some cases are found many feet below the level of drainage, necessitating pumping at considerable expense.

My own experience in mining talc has been that the chief point is to get a reliable "vein," and that then the mining is not as difficult, or any more so at least, than that of any common ore. If the "veins" are pure talc it is simple; if the talc is admixed with foreign matter, such as tremolite in excess, or stains of iron or manganese, as is often the case, one had better abandon the deposit.

## *Methods of Manufacture*

Talc, being a soft mineral, is not difficult to reduce to the condition desired by the trade. Powdered talc is admitted under low duty from France, Italy and Austria, and some is being imported from Asiatic points. Talc is usually ground and bolted in any simple reduction mill. That mined in this section is hand-sorted and the quality that will make good powdered talc is ground for use in the manufacture of talcum powder, cosmetics, etc., while all that is hard enough and has sufficient strength

is used for crayons and blanks for gas burners and electrical work, being sawed into the shapes desired. The rest that is clean and pure is ground in mills, of which many different kinds are used. Some people like the mill stone, while others prefer the many kinds of high-speed beater mills for grinding.

The manufacturing process can be summed up as follows:

More or less hand-sorting may be necessary, and in grinding the talc must be reduced sufficiently to pass through a 170- to 200-mesh sieve, silk cloth reels being commonly used.

Among the objectionable impurities which it is almost impossible to remove are lumps of tremolite and pyrite. They destroy the value of the talc by changing the color and making a coarse and hard gritty product which is not saleable.

### *Uses*

Most of the talc sold is used in the paper, rubber and paint trade, while the lower grades go largely into roofing. About one-quarter of the production goes into talcum powder. Not a little is employed as a body or carrier of medicinal chemicals used in tablet form.

Much of the talc or soapstone is cut with small saws into crayons of various sizes for use in the iron-working trades and on blackboards. A growing business is the manufacture of gas burners and small blocks for electrical work, which, after shaping and burning, are known as lava goods.

The two last, crayon and burner manufacture, are the most important uses of talc, but the greatest difficulty is to find mineral of the proper quality, as it has to be solid, firm, and free from grit and other foreign matter.

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# An Explanation of the Flotation Process

BY ARTHUR F. TAGGART* AND FREDERICK E. BEACH,† NEW HAVEN, CONN.

(Arizona Meeting, September, 1916)

## INTRODUCTION

THE flotation process for the concentration of ores is a method by means of which one or more of the minerals in the ore (usually the valuable ones) are picked up by means of a liquid film and floated at the surface of a mass of fluid pulp. Here they are separated from the other minerals, which remain immersed in the body of the pulp. In general, the minerals which are floated are sulphides of metallic luster, but some other minerals of metallic luster such as graphite and some sulphides with adamantine luster, such as sphalerite and cinnabar, are amenable to treatment by the process.

The importance of flotation lies in the fact that it is primarily a "slimes process" by means of which the particles of valuable mineral, too fine for efficient gravity concentration, are saved with a high percentage of recovery. Recoveries in the mills treating low-grade copper sulphide ores have been advanced 10 to 20 per cent. by the installation of the process and similar increased savings have been accomplished by the same means in mills treating sulphide ores of zinc and lead.

When finely ground ore containing sulphides mixed with a siliceous or earthy gangue is brought gently onto the surface of a body of water, in a direction forming an acute angle with the surface of the water, a considerable portion of the sulphide constituent of the ore floats on the surface of the liquid, while the gangue sinks. This is the so-called film flotation, exemplified by the Wood and Macquisten processes.

When gas bubbles are introduced into a fluid pulp composed of finely ground ore and water, to which has been added (1) a small amount of certain oils, or (2) a small amount of certain acids or acid salts, or (3) a small amount of certain alkalis or alkaline salts, or (4) a small amount of a mixture of oil with acid or alkali, the sulphide particles in the ore are brought to the surface on the gas bubbles. These collect in a froth

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heavily laden with sulphide particles. The gangue particles are not brought up by the bubbles, but remain in the mass of the pulp. This is the so-called froth flotation. Its variations, acid-froth, agitation-froth, and pneumatic-froth processes are discussed in detail later.

The points to be explained in the operation of these processes are: (1) The flotation of solid particles in a liquid the specific gravity of which is less than that of the solid; (2) the preferential flotation of the sulphide portion of the mass; (3) the functions of the reagents used.

## Theory

The theory here presented in explanation of the points listed in the preceding paragraph appeals to the following physical phenomena: Surface tension, adsorption, adhesion and viscosity. The first three of these are closely related.

### *Surface Tension*

Every liquid surface in contact with a gas or its vapor, behaves as if it were under tension. The value of this contractile force per unit width can be measured. Its value for water, 74 dynes per centimeter, is higher than for any other well-known liquid. (Liquid metals and fused salts are of course excepted.) This fictive tension is a convenient conception for many discussions and may be explained in terms of the intermolecular attractions of the substances forming the boundary. Another and very useful way of considering the phenomenon is to regard each unit of surface as having associated with it an amount of potential energy which is numerically equal to the surface tension. This relation is established as follows:

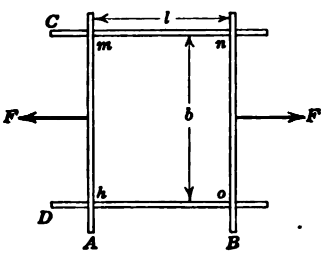

Fig. 1.

If two wires, $A$ and $B$ (Fig. 1), are so placed as to slide on two fixed wires, $C$ and $D$, and if the wires, $A$ and $B$, having been in contact, are separated a distance $l$ against the pull of one of the surfaces only of a film, *mnoh*, by applying the force $F$, the work per unit area will be

$$\frac{W}{A} = \frac{Fl}{lb} = \frac{F}{b} = T$$

which is obviously equal, numerically, to the force per unit width or the surface tension, $T$, of the one surface considered. Applying this conception of surface energy to different cases of contact, we can develop some statements of the relations of forces which are important in explaining many of the phenomena observed in flotation.

Consider first that two blocks of liquid, which have become rigid without any change in their other properties, are drawn together by the mutual action of their forces of molecular attraction until they perfectly unite over an area $A$. They would do an amount of work that we will represent by the letter $W$. Now consider that the same two bodies when brought near together, but not into contact, become liquid and unite over the area $A$. The work done is again $W$, but in this case it can be considered as due to the shrinkage of the surface by an amount $2A$, whence $2AT = W = A(T + T)$, where $T$ is the surface tension of the liquid. If two different liquids whose surface tensions are $T_1$ and $T_2$ respectively were brought together, the work due to the molecular attractions would be $(T_1 + T_2)A = W$, provided there was complete union, but if there is not complete union, there will be an interfacial tension $T_{12}$, and the energy equation becomes

$$(T_1 + T_2)A - T_{12}A = W$$

or

$$T_{12} = T_1 + T_2 - \frac{W}{A}$$

*i.e.*, the interfacial tension $T_{12}$ is the excess of the sum of the two tensions over the work which is done by them in allowing a unit area of the two liquids to come into contact. If a liquid is brought into contact with a solid, the energy equation is $T_{LS} = T_L - W_1$ (gas or vapor effects being excluded),

where $T_L$ = the surface tension of the liquid,
$W_1$ = the work done in bringing a unit area of the liquid and solid into contact,
and $T_{LS}$ = the interfacial tension.

If, therefore, $W_1 = T_L$, the solid has the same attraction for the molecules of the liquid as the molecules of the liquid have for each other and there will be no interfacial tension. If $T_L > W_1$ the interfacial tension $T_{LS}$ will be positive; if $T_L < W_1$ there will be negative interfacial tension or a surface pressure. In the latter case the liquid will tend to spread over the solid.

## *Angle of Contact*

When, as is the common case in the flotation process, there are three substances in contact, a system of forces as shown in Fig. 2 is brought into play. If $O$ does not move indefinitely to the right or to the left, equilibrium will be attained when

$$T_{SL} = T_{GS} + T_{GL} \cos \theta$$

or

$$\cos \theta = \frac{T_{SL} - T_{GS}}{T_{GL}}$$

where $T_{GS}$, $T_{GL}$, and $T_{SL}$ are the interfacial tensions or pressures at the gas-solid, gas-liquid, and solid-liquid contacts respectively. From this equation is deduced the important conclusion that as $T_{SL}$ increases with respect to $T_{GS}$, the angle of contact $\theta$ becomes smaller (the gas and liquid being the same), or, in other words, the angle of contact is a measure of the tendency of one fluid to replace another on the surface of a solid. We have examined the angles of contact of the water-air, oil-air, and oil-water surfaces against a number of the common minerals. We have found, in general, that the air-water contact angle is less for gangue minerals than for sulphide minerals; that the air-oil contact angle is less for sulphides than for gangues and less for any given sulphide than the air-water contact angle; and that the water-oil contact with solids takes the form shown in Fig. 3.

We found further that the invariable effect of oiling a solid surface is to reduce the air-water contact angle. This latter phenomenon is un-

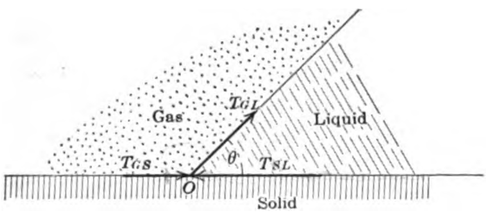


FIG. 2.

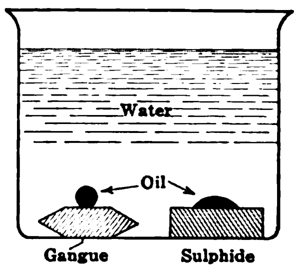


FIG. 3.

doubtedly aided by the reduction of the surface tension of the water due to contamination by the oil. This is further discussed later.

The conclusions forced by observation of the above phenomena are:

1. That water has a smaller tendency to displace air on the surface of sulphide minerals than on the surface of gangue minerals.

2. That the tendency of oil to displace air is greater at the surface of sulphide minerals than at the surface of gangue minerals.

3. That oil tends to displace water on the surface of sulphides and that water tends to displace oil at the surface of gangue minerals.

4. That water displaces air more readily on an oiled solid surface than on a clean surface of the same solid.

5. That these tendencies toward displacement are due to the interfacial tensions or pressures existing between the various substances, and that the resulting action of these interfacial forces is a manifestation of the tendency toward reduction of the total potential energy of the system. Wherever an increase in the solid-fluid interface will decrease the potential energy, such a change will occur.

These conclusions suggested the following confirmatory experiment.

A ring, 6.17 cm. outside diameter and specific gravity of 1.38, made of aluminum tubing, 0.63 cm. diameter, was cleaned and floated without trouble on the surface of pure water. The shape of the water surface at the air-water contact is shown in Fig. 4. The ring was then oiled slightly. The air-water contact angle was reduced, as shown in Fig. 5, to such an extent that it was impossible to float the ring. The same was true, as might be expected, when a cylinder of aluminum replaced the ring. A similar cylinder of glass tubing exhibited such a small air-water contact angle that it could not be floated.

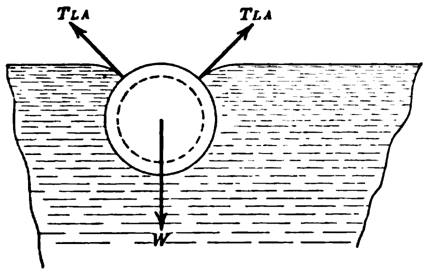


FIG. 4.

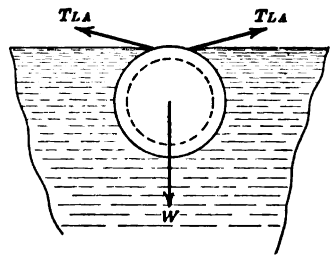


FIG. 5.

*Adsorption*

The surface layer between two physical phases is the seat of conditions of density and viscosity, also of apparent forces or energy manifestations, which are notably different from those in the bulk of either phase. On philosophical grounds it is impossible to consider that a real physical discontinuity occurs at the boundary between two media. In other words, there must be a very thin layer of transition in which there is a rapid but continuous change in the concentration of the components. This change in the concentration of a component at the interface is called adsorption, and may occur even between two phases which are ordinarily regarded as immiscible.

Adsorption at a gas-liquid interface may be demonstrated as follows: If a solid, which has been heated in a vacuum, is introduced into a measured volume of a gas over mercury in a calibrated tube, an amount of the gas will be adsorbed, as is shown by the change in pressure and volume compared to the space originally occupied.[1] These additional facts are established:

(*a*) The amount of the gas adsorbed at constant temperature increases with the pressure.

(*b*) It is different for different gases.

(*c*) It is different for different solids.

(*d*) It increases as the temperature decreases.

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[1] Freundlich: *Kapillar Chemie*, p. 92.

(*e*) There is an energy transformation which is indicated by the heat developed through adsorption, analogous to the Pouillet phenomena mentioned later.

(*f*) Chemical reactions are assisted by the adsorbed layer.

It follows that the gas layers must vary in density, falling off rapidly with increasing distance from the solid. Quincke assumes that the density of the gas next to the solid is equal to that of the solid and concludes that the amount adsorbed will increase with the density of the solid. From these facts we conclude:

1. That gases and solids exhibit selective adhesion and that, therefore, gas bubbles will attach themselves more persistently to some substances than to others.

2. That this selective adhesion is a manifestation of a definite amount of energy possessed by each unit area of a gas-solid contact, and that this potential energy is capable of variation.

3. That chemical reactions which diminish this potential energy are aided by adsorption.

Adsorption at a liquid-solid surface manifests itself in a vacuum, or where the vapor phase is negligible, by the way in which the liquid spreads or gathers itself together on the solid; in other words, in the way in which the liquid wets or adheres to the solid. It is further manifested by an evolution of heat, known as the Pouillet phenomenon. A calculation of the condensation necessary to evolve this amount of heat, in the case of water against glass, indicates that the specific gravity of water in the adsorbed layer is increased to about 2.1.[2]

Adsorption of the gas at a gas-liquid surface is indicated

1. By the effect on the surface tension. The surface tension of a freshly formed mercury surface does not change in a vacuum, but falls off in the presence of different gases for about an hour. Certainly the density of a liquid cannot be constant at the boundary but must go over continuously into that of the gas.

2. By the increase in the solvent power of the surface.[3]

3. In the case of contaminated liquids, by the concentration of one or more of the components of the liquid at the gas-liquid surface. Every unit area of such a boundary possesses a definite potential energy which always tends to a minimum. If, therefore, the surface tension of a solution depends upon the presence of any component, such a change of concentration of that component will occur as will reduce the potential energy, *i.e.*, the interfacial tension. In other words, any component which reduces surface tension will be found in excess at the surface of a solution. For example, the surface tension of water is greater than that of alcohol. Experimentally, a drop of alcohol on a thin film of water

[2] Lewis: *Phil. Mag.*, 20, p. 502, 1910.

[3] Pockels: *Nature*, Mar. 12, 1891, p. 439.

rapidly reduces the surface tension of the water and the latter draws away from the alcohol. On the contrary, a drop of water on a thin film of alcohol spread over glass does not at once diffuse into the film but remains gathered in a heap. Such diffusion would increase the surface energy of the system, hence the water concentrates away from the gas-liquid surface. The greater viscosity of the surface of a solution above that of the bulk, or of that of a pure liquid, has long been recognized[4] by its damping effect upon a swinging magnetic needle and may properly be ascribed to gas-liquid adsorption. Closely connected with this is the formation of elastic solid skins or very viscous layers at a free surface, as, for example, in the case of solutions of peptone and dye stuffs. A peculiarity of saponine solution is the rigidity of its surface while the interior remains more mobile.[5] The surface of a freshly formed fairly concentrated solution of fuchsin is quite mobile, but in the course of a few

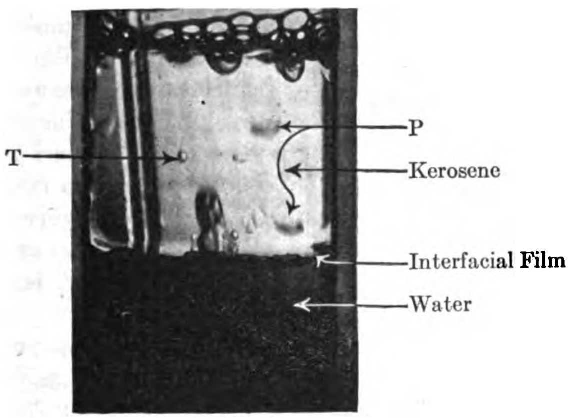


FIG. 6.

hours it changes to a reflecting skin with solid properties. Similar results are obtained with methyl-violet and peptone. In the case of the latter substance the skin is highly elastic. There seems to be no doubt that true adsorption is present here. In the case of crystal-violet, which closely resembles methyl-violet in its properties, a solution of 1 g. to the liter lowers the surface tension from 75 to 69.9 dynes per centimeter. Other causes for the production of a solid layer may, however, be present, for many of these substances in concentrated solutions stiffen into gelatines and since the concentration of the contaminant is great at the surface and the solubility has also a different value, the solid remains persistently. Either a chemically irreversible change or a transformation into a more difficultly soluble phase at the surface is clearly the explana-

[4] Daniells: *Physics*, p. 258, Fr. p. 76.
[5] Boys: *Soap Bubbles*, p. 115.

tion of the persistence of the froth in albumen solution and the like. The properties of such surfaces apparently pass over imperceptibly into those of colloids.

Adsorption at the boundary between two liquids is evidenced by the effect on the interfacial tension just as in the gas-liquid solution surface, although the process is not one usually described as ordinary solution and one may also have to reckon with chemical reactions in the transition layer. With liquid-liquid surfaces, as in gas-liquid surfaces, the adsorption frequently gives rise to very viscous layers. The presence of such a viscous layer at an oil-water interface is easily shown by pouring any clear oil, kerosene, liquid vaseline, etc., onto water and then bubbling a gas through the water. Such an experiment is shown in Fig. 6. The interface has all the appearance of an elastic skin. Bubbles rising through the water and striking the under side of the interface stretch the film (see *H*, Fig. 7), and rising farther drag away a mass of water surrounded by this viscous layer. The system now appears as shown at *M* and rises to the oil-air surface on account of its lower specific gravity. Here the film, together with the excess water carried up as shown at *C*, breaks away and falls back through the oil, not in spherical form, as would be the case were the water drop not surrounded by a viscous film, but in hemispherical form (see *P*, Figs. 6 and 7) often trailing behind it a film with ragged edges, as it broke from the bubble. The tadpole-shaped water drops, *T*, (Figs. 6 and 7) are further evidence of the high viscosity of the oil-water interfacial film.

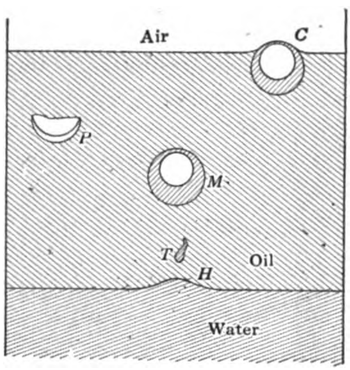

FIG. 7.

## *Viscosity*

A marked increase in the viscosity of interfacial films is produced by the presence of finely divided solid matter. This increase is apparent in the experiment just described when finely powdered sulphide is thrown into the oil and allowed to settle to the interface, where it becomes entangled in the film. When gas bubbles are introduced, as before, the return water drops, coated with a film containing the solid particles, are much more irregular in shape than previously, and their coalescence after reaching the interface requires days or weeks. An even more convincing proof of the increase in viscosity of an interfacial film is given by the following experiment. If a needle is floated at the center of a surface of pure water in a beaker 4 in. in diameter and a chip of wood is floated

near the wall of the beaker, the needle may be caused to revolve by means of a magnet without disturbing the chip. If the surface of the water is dusted over with fine ore, the whole surface together with the chip moves as though it were a rigid solid.

## SUMMARY

The potential energy at a gas-sulphide contact is less than at a gas-gangue contact; hence gas bubbles will cling with greater persistence to sulphides than to gangues.

Oil replaces water at the surface of sulphide minerals.

Water replaces oil at the surface of gangue minerals.

Water replaces gas more readily at an oiled surface of a solid than at a clean surface.

The addition of any contaminant to water lowers the surface tension.

In any body of contaminated water there will be a concentration of the contaminant at the air-liquid surface.

Adsorption at a gas-liquid surface lowers the surface tension and increases the viscosity.

Adsorption at a liquid-liquid surface produces a film whose viscosity is higher than that of the bulk of either liquid.

The presence of finely divided solid matter in a film markedly increases the viscosity of the film.

## APPLICATION TO COMMERCIAL FLOTATION PROCESSES

### *Film Flotation*

Pulp with or without oil or acid is fed gently, at an acute angle, onto the surface of a body of still water. The sulphide floats and the gangue sinks.

Possible cases:

Case 1.—Sulphide, gangue, water.

Case 2.—Sulphide, gangue, water, oil.

Case 3.—Sulphide, gangue, water, acid.

Case 4.—Sulphide, gangue, water, acid and oil.

*Case* 1. *Sulphide, Gangue, Water.*—The governing factor in the initial flotation of the sulphide and immersion of the gangue is the difference in the air-water contact angle with the sulphide and gangue surfaces respectively. If the difference is great, as in the case with galena and quartz, good separation is obtained. After a considerable amount of sulphide has been floated, if the surface flow is sufficiently impeded, the particles congregate into clumps by the well-known phenomenon of apparent attraction of floating particles,[6] a scum is formed (whose viscosity and

[6] Hastings and Beach: *General Physics*, p. 156.

resistance to rupture are many times greater than that of the original water-surface) and the floated particles are rendered more immune to immersion.

*Case 2. Sulphide, Gangue, Water and Oil.*—When oil is added in this process the phenomena are entirely different from the simple film flotation of Case 1. The oil concentrates at the surface since such concentration reduces the surface energy of the system. This adsorption of the oil at the gas-water surfaces causes the formation of a viscous film. When the mixture of sulphide and gangue is introduced at the surface, the sulphide particles tend to migrate into this layer and the gangue particles migrate into the water, for a sulphide particle in contact with oil represents the condition of least potential energy which is possible for the sulphide particle to assume in the system of oil, sulphide, and water. Likewise the gangue particle surrounded wholly by water represents the condition of least potential energy for the gangue particle to assume in this system. In this case also, the formation of a scum of floated sulphide increases the stability of the float.

*Case 3. Sulphide, Gangue, Water and Acid.*—The effects of acid are: (*a*) to diminish the surface tension of the liquid, (*b*) to diminish the gas-liquid contact angle, and (*c*) to increase the viscosity of the gas-liquid surface. The diminutions of (*a*) and (*b*) are more marked with gangue minerals than with sulphides. The result is that cleaner concentrate may be expected than in either of the previous cases, but at the expense of richer tailing.

*Case 4.—Sulphide, Gangue, Water, Oil and Acid.*—In this case a combination of results such as can be predicted from the preceding cases is obtained.

### *Froth Flotation*

In order to explain froth flotation it is necessary and sufficient that the gas of each bubble shall be inclosed by a film of contaminated water which shall possess the following characteristics:

1. Low surface tension.
2. High viscosity.
3. A variable concentration of the contaminant (reagent).
4. A preferential adhesion of the bubble film to the sulphide mineral compared to that for the gangue minerals.

We will examine first the conditions required for the formation of a froth, or the continued existence of a thin film. Solutions which form froth are preëminently aqueous solutions and the properties of the liquid film are only secondarily determined by those of the gas.

The durability of a liquid film depends upon one or more of the following conditions:

1. A low surface tension which is locally variable so as to produce stable equilibrium.

2. High viscosity, which may pass over into
3. Chemical irreversibility and the production of solid skins.

Pure liquids do not foam; for example, water or alcohol. The reason is obvious. As the film is thinned out by stretching or by draining away of the liquid, the surface tension is reduced at some part below the general constant value. As soon as this begins the thicker and more powerful parts of the film drag away from the weakened parts which at once completes the rupture. These inequalities evidently would not be so marked or rapid in their operation if the surface tension were low. Increase of viscosity would also slow up the process. High viscosity and low surface tension do not occur in pure liquids. But the most important condition for durability is some means by which the equilibrium of the forces at any point in a film may be restored, when a variation of some of the forces occurs. In the case of a compound liquid or solution this is effected by the adsorption or change of concentration of one or more of the components in the film.

The surface tension of a solution is in general notably different from that of the pure solvent, and in case of water, whose surface tension is the greatest of any liquid with which we are concerned, even a minute trace of impurity is sufficient to lessen its surface tension considerably.

Consider a film of water stretched on a vertical ring of wire. If the surface tension remained constant, as it does in the pure liquid when the thickness exceeds 0.000001 cm., the weight of the lower part would stretch down the upper part until it broke. If the water contains some component whose depletion at the weaker points increases the surface tension, equilibrium will be preserved. In the stretching of a film, and in the general running away of the liquid between the surfaces of the film, which reduces the total available amount of the contaminant, such decrease of the concentration and increase of surface tension does occur, and the film remains stable under a considerable variation of external conditions. The formation of bubbles as a result of this variation of surface tension alone is well exemplified by a simple aqueous solution of soap or of acetic acid. The running out of the liquid between the two surfaces is greatly retarded by the viscosity of the liquid, a property which may be largely influenced by the surface adsorption of one or more of the components.

When, then, gas bubbles are introduced into a liquid pulp where oil is present there is formed about each bubble a liquid film whose surface tension is less and whose viscosity is greater than that of the bulk of the liquid. Some of the solid particles of the pulp move into the film and are raised to the surface with the bubble. Since there is a concentration of oil in the film, and since the diminution in potential energy at an oil-sulphide contact is greater than at a water-sulphide contact, the contaminated layer replaces the water on the sulphide surface and the

sulphide moves into the bubble film, while the gangue, on which water displaces oil, remains in greater measure in the body of the pulp. The bubbles, therefore, as they arrive at the surface, carry an excess of the sulphide minerals. Upon their arrival at the surface, the bubbles of the contaminated liquid persist, owing: (1) to their lower surface tension; (2) to their ability to adjust this tension to a state of stable equilibrium; and (3) to their greater viscosity which is markedly increased by the presence of the solid particles.

### *Mechanical-Agitation Froth Process*

Sulphide and gangue minerals are beaten up with water and oil, with or without acid, then allowed to flow into a box containing a considerable body of liquid nearly at rest. Bubbles coated with a preponderance of sulphide particles float to the surface and form a heavy, persistent froth. The gangue particles sink.

Two cases arise:

Case 1.—Sulphide, gangue, water and oil.

Case 2.—Sulphide, gangue, water, oil and acid.

*Case* 1. *Sulphide, Gangue, Water and Oil.*—When this pulp is beaten, air is mechanically entrapped in the form of bubbles. At the surface of every bubble in the mass there is a gas-contaminated-liquid contact which results in the adsorption at this surface of the contaminant, oil, and the production of a viscous film into which the sulphide particles, circulating in the mass, pass with a diminution in the potential energy of the system. The result is that in a very short time after the air bubble is entangled in the pulp, it is surrounded by a viscous sheath composed of an oil-water interfacial film in which are entangled a large number of sulphide particles. The presence of the solid particles greatly increases the viscosity of the bubble sheath. When the solid-coated bubble arrives in the settling box or spitzkasten it rises to the surface. Here the bubble persists, or, bursting, transfers its load to other bubbles. This bubble persistence is due to a combination of several factors. The oils used have, in general, a slower evaporation rate than water. The tension of the bubble film is lower than the tension of a pure water bubble. The bubble has the power of adjusting itself to its tension, within limits, without bursting. The presence of the large amount of solid matter enormously increases the viscosity of the film.

*Case* 2. *Sulphide, Gangue, Water, Oil and Acid*—The addition of acid has the twofold effect of further lowering the surface tension and increasing the adhesion ratio $\frac{\text{oil-solid}}{\text{water-solid}}$. The result is, in general, cleaner concentrate with or without an increase in the sulphide content of the tailing.

Heating the pulp has, in some cases, a beneficial effect. Where this is true it is probably due to (*a*) decreased surface tension and consequent increased stability of the froth; (*b*) increased number of air bubbles formed by the air released from solution; (*c*) in the case of viscous oils, the greater area over which oil is spread and consequently the greater number of bubbles with a viscous oil-water interfacial film; (*d*) probable increase in solubility of the oil and consequent greater diffusion, resulting in more widespread adsorption in bubble films.

*Pneumatic Froth Process*

Sulphide and gangue minerals mixed with water and oil, with or without acid, are run into a tank with a porous bottom through which air is forced. The air bubbles rise to the surface with a coating of solid particles, preponderantly sulphide, while the gangue particles sink.

The principles involved in this method are the same as explained in the agitation-froth process. The only difference is in the method of introducing air. The result of this difference is that the bubbles in the pulp are much larger than in the agitation froth method; they arrive at the surface less heavily loaded in proportion to their area; the bubble films are, therefore, less viscous, and the froth less persistent.

*Potter-Delprat Process*

Sulphides and carbonates, with or without other gangue minerals, are treated with hot, dilute sulphuric acid. Bubbles of carbon dioxide and hydrogen sulphide are formed which rise to the surface with a sulphide coating and there form a froth. That part of the gangue not dissolved remains immersed. In this method as in the other froth methods, gas bubbles are formed which are surrounded by films of contaminated water, the contaminants in this case being sulphuric acid, lead sulphate, calcium sulphate and other salts formed by the action of the sulphuric acid. The films have a higher viscosity and a lower surface tension than is possessed by the bulk of the liquid. The sulphides move into the bubble films because the system composed of sulphide and this contaminated layer has a lower potential energy than the system composed of the pulp in the bulk of the liquid. The writers at first suspected that the selective action in this case might be due to preferential gas adsorption at the gas-sulphide contact, as opposed to a gas-gangue contact, but microscopic examination of mineral froths collected from the process showed that the solid particles in the froth were completely within the films and at no point in contact with gas. The persistence of the froth is due to the factors explained in connection with the other froth processes.

While the writers have made no appeal to electrostatic forces or to

colloidal phenomena in this discussion, they realize that the potential energy existing at the contact of dissimilar substances may well include electrical forces and that migration of the suspended solid particles under the influence of electrical charges, similar to the migration of colloids, may account for some of the selective action of the bubble films. But the agitation of the pulp in the mechanical- and pneumatic-froth processes and the generation of carbon dioxide gas on carbonate particles in intimate contact with sulphides in the acid-froth process, are sufficient, in their opinion, to bring every sulphide particle into contact with a bubble film. Once in contact, the preferential adhesion of the contaminated layer to a sulphide surface in the presence of water is sufficient to account for the persistent attachment of the sulphides to the bubble films, while on the other hand, the replacement of gas or oil by water on the surface of gangue particles explains the wetting and continued immersion of the latter.

The writers have a considerable bulk of experimental data on which many of the statements in the foregoing explanation are based. These, together with the data from other experiments which are projected, and photographs of many of the phenomena mentioned, they hope to present in a later paper.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Some Miscellaneous Wood Oils for Flotation.*

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(Arizona Meeting, September, 1916)

THE testing of flotation oils has occupied a large part of the time of the testing departments of various companies using the flotation process in the beneficiation of their ores. The great differences in ores and in oils has made such work necessary as it has not been possible to select the oil best suited for the purpose without trial of many kinds. In fact, individual shipments of flotation oils from the same manufacturer will often differ so radically that the mill men have been forced to test every lot of flotation oil before its use in the mills.

Further, although almost every imaginable product of the distillation of wood, from alcohol to pitch, has been tested in a more or less perfunctory manner by various people, few companies have allowed the results of their tests to be made public.

This work was undertaken for the purpose of obtaining reliable data regarding the value as flotation agents of a number of products that are obtained from the distillation of wood by different methods, and incidentally to determine whether some of the products not used at present could be used. Large quantities of such materials are being wasted or burned in the wood-distillation plants of the United States, and if some use other than fuel could be found for these materials, such a use would represent a distinct saving to the country as a whole, would increase the profit of wood-distilling plants now operating and perhaps make it possible for others to operate which at present cannot do so except at a loss. At the same time it was thought that if such products can be used as flotation agents, then their use would serve to increase the number of oils and other products available for flotation purposes. It was also hoped that it might be possible to discover a substitute for some of the most effective but comparatively high-priced oils which are now used for flotation

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* By permission of the Director, U. S. Bureau of Mines.

This work is the result of coöperation between the Forest Products Laboratory of the Department of Agriculture, the Metallurgical Research Department of the University of Utah, and the U. S. Bureau of Mines. The authors are members of the three organizations.

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purposes. Hence these fractions occupy the major portion of the following list.

In taking up this work, the Forest Products Laboratory of the Department of Agriculture, which is interested in finding uses for byproducts from wood distillation, was represented by R. C. Palmer. The Bureau, which is interested in finding supplies of suitable flotation oils, was represented by O. C. Ralston, and the Department of Metallurgical Research of the University of Utah, which is doing coöperative work with the Bureau of Mines, was represented by G. L. Allen.

Three ores were chosen for tests of the series of oil samples herein described.

Ore A was from the mine of the Arthur Zinc Mining Co. near Ruby Valley, Nev. It was chosen on account of its simplicity. It consisted mainly of galena, sphalerite and quartz minerals, which were crystallized in such a way that they could be broken away from each other by crushing to the size necessary for flotation. Ground quartz was mixed with it to make it sufficiently low-grade. Any marked tendency of one mineral to float in preference to another could be observed with such a "free" ore. Consequently, the value of the oil for differential flotation of mixtures of galena and sphalerite could be observed; also the conditions which allowed the best extraction or the cleanest concentrate of any of these minerals with any given oil.

Ore B was the heavier sulphide ore from Cripple Creek, Colo. This siliceous ore contains some pyrite, some gold and silver as the tellurides, sylvanite and calaverite, and other tellurium minerals. The tests on this ore would allow observations on the flotation of pyrite as well as of the famous Cripple Creek gold telluride minerals. Wherever the pyrite contains gold it would be desirable to float it, while barren pyrite ought to be kept with the gangue. Hence the tests should be carried on with an ore containing both types of minerals, as was done in this case. This sample was furnished by the Golden Cycle Mining & Milling Co.

The ore C was the Utah Copper Co.'s milling ore from the mine at Bingham. A great many copper minerals are present in this ore—chalcopyrite, cupriferous pyrite, etc. The gangue is siliceous, and the ore is well known. It is a hard ore to treat by flotation and nearly any oil that will treat it is almost sure of success with most other copper ores. One peculiarity of the Utah Copper ore is that it is much easier to float in an alkaline solution.

Each of the ores was tested with three different tests—an acid test, an alkaline test, and a test run in neutral water. The water used in all the flotation tests was the Salt Lake City water. It contains a small amount of lime so that the water used in the "neutral" test was in fact slightly alkaline. This method of testing was adopted for the reason that it is known that some oils will refuse to work at all in neutral solution

although they do good work in acid solution. Others are more successful in alkaline solutions. The most advantageous amounts of acid or of sodium carbonate were not determined. The procedure followed, in order to facilitate work, was to add the reagent drop by drop to the charge in the flotation machine until it could be seen that it was causing some difference in the appearance of the froth or of the gangue. Further additions of the reagent were often made after most of the froth resulting had been skimmed off, and it frequently happened that a further amount of excellent froth would result.

Although such a method of obtaining the best conditions is not satisfying to certain types of mind, and is very aptly characterized as the "American" type of research, it is practically necessary to follow such a method in order to get anywhere with such testing. So many other variables are involved that to painstakingly follow out each one for only one oil, while the others are kept constant, is an immense undertaking. A flotation testing man must be an opportunist and seize on the best conditions for flotation as he is able to see them in his machine when varying the conditions, such as amount of agitation, depth of pulp in machine, amount of oil, amount of acid or alkali, temperature, etc.

The Janney test machine described in the *Mining and Scientific Press* for Jan. 1, 1916, was used in all this work. Our reason for using this machine is that it permits of quick work and is so designed that practically every particle of flotation concentrate and tailing can be recovered and weighed, while many other machines have inaccessible corners where some of the material escapes.

The extractions were calculated by the following well-known formula which does not require weights of the products involved.

$$\text{Extraction} = \frac{100\,(h-t)c}{h(c-t)}$$

where the analyses of the head, or original ore, tailing, and concentrate are represented by the letters in the equation. This formula assumed that any middling produced can be separated into concentrate and tailing.

Each test was "roughed" and "cleaned." The rough tail was final but the concentrate was poured back into the machine with more water and a clean concentrate and middling produced.

The oils tested in these experiments were furnished by the Forest Products Laboratory,[1] Madison, Wis. Most of them were either commercial samples or were produced in the laboratory in semi-commercial apparatus under conditions entirely comparable with plant practice.

The following is a somewhat detailed description of the different oils.

No. 23 was an authentic commercial pine oil, specific gravity 0.9385

[1] Maintained by the Forest Service, U. S. Department of Agriculture in coöperation with the University of Wisconsin.

at 21° C., with a sulphonation residue of 0.8 per cent. The oil was produced by the steam distillation of fat southern pine wood, the crude oil being redistilled to separate the turpentine (boiling point 155 to 170° C.), from the pine oil (boiling point 190 to 220° C.). As there is some confusion in the mining industry in regard to the name "pine oil," many other oils produced from dead pine wood also being known as "pine oil," it may be well to describe the oil to which the name is correctly applied. There is a resin in the sapwood of the living pine, which flows out when the tree is wounded. This resin is the source of the ordinary turpentine and rosin of commerce. When the sapwood turns to heartwood, the resin becomes hard and during the ageing process another oil is formed which did not exist in the sapwood. This oil is heavier than turpentine, has an entirely different composition, and a much higher boiling point, and it is this oil that is correctly termed "pine oil." This oil can be removed from the dead pine wood either by the action of steam or dry heat. Dry heat carefully controlled so as not to exceed the maximum boiling point of the oil will remove it in practically as pure condition as when the wood is treated with saturated steam. Pine oil is produced either as a yellow or water-white oil, the former being produced in the first separation of the crude turpentine-pine-oil mixture, and the latter by redistillation of the yellow oil. If the wood is heated at a much higher temperature, as it is when the temperature is not controlled during the early stages of the destructive distillation process, the natural oils (turpentine and pine oil) in the dead wood are contaminated by tar and tar oils produced by the decomposition of rosin and wood. It is practically impossible commercially to separate a pure pine oil from this mixture.

Chemically, pine oil has been found to be composed chiefly of a terpene called terpeneol, although other terpenes such as borneol, fenchyl alcohol, and cineol, are also present. Since the detection of mineral oils as adulterants in pine oil is of special interest it may be well to point out that the sulphonation test may be successfully used for the detection of mineral oil. A pure pine oil should give a sulphonation residue of less than 1 per cent. Special care must be taken, however, in carrying out the test, and it is suggested that the test be made as follows:

Take 25 c.c. of 37 normal sulphuric acid in a Babcock bottle and add 5 c.c. of the oil to be tested, 1 c.c. at a time, shaking vigorously after each addition. When the oil is all added, place the bottle in a water bath and heat at 100° for 1 hr., shaking cautiously from time to time to insure a perfect mixture of the oil and acid. Cool and fill the bottle with ordinary sulphuric acid and either allow it to stand for 24 hr. or centrifuge it in order to read the residue. It is absolutely essential to have the strength of the acid exactly 37 normal; the cooking of the mixture for 1 hr. is equally important for securing low residue with pure pine oil. The index of refraction of the residue is also of value. With pure pine oil the residue

should have an index of over 1.5, while a mineral oil will have a residue with an index of less than 1.48.

A pine oil was included in these tests for the purpose of comparing an oil of known flotation value with the other oils. The sample had been shaken repeatedly with hot water in order to remove any water-soluble constituents (probably the fenchyl alcohol), to determine whether the removal of water-soluble material had any noticeable effect on the value of the oil for flotation. About 5 per cent. was found to be soluble in water.

No. 11 was a commercial wood oil produced in the destructive distillation of hardwood, the particular species in this case being maple and birch. The specific gravity of the oil was 0.934 at 21° C. The chemical composition of this oil is unknown, although it probably contains a number of complex ketones. In the destructive distillation of hardwood a heavy tar settles out of the crude pyroligneous acid, the watery distillate containing the wood alcohol and acetic acid, and this tar has included in it some of the pyroligneous acid. To remove this acid the tar is usually placed in a wooden still and either washed by blowing in exhaust steam or is simply heated with steam in closed coils. In either case a watery product distills over and with it is carried this light tar oil or "wood oil," as it is called in the plant. This oil is practically a waste product and is burned for fuel in most plants. It is an excellent solvent, however, and a small amount is sold for that purpose. A cord of hardwood gives on an average about 2 gal. of this oil and the production in all the hardwood distillation plants is probably close to 2,500,000 gal. annually. It could probably be bought for $0.10 to $0.15 per gallon f.o.b. plant, as this is the price usually paid for it in the crude condition for a solvent.

No. 18 was a crude hardwood tar just as it settled out of the pyroligneous acid, as described above. This tar was produced in a semi-commercial laboratory distillation plant by the distillation of equal proportions of beech, birch and maple. It still contained the pyroligneous acid and wood oil. On distillation the sample gave 12 per cent. pyroligneous acid, 8 per cent. wood oil, 42 per cent. wood creosote oil, and 38 per cent. wood pitch. The specific gravity of the combined wood oil and wood creosote was 1.067 at 21° C. When the distillation of the crude tar is continued with direct heat after distilling off the watery distillate and light oil (wood oil) the oil obtained is similar to the hardwood creosote now on the market for flotation. The object in testing a crude hardwood tar of this kind was primarily to determine to what extent the pitch present in the tar acted as a deleterious adulterant. Excluding the wood oil, about 15 gal. of this tar is produced per cord of hardwood, making available per year over 20,000,000 gal. of this crude tar. This tar with the wood oil and pyroligneous acid removed is generally used for fuel at the distillation plant, where it is worth about $0.03 per gallon for this purpose,

in comparison with the normal prices for other kinds of fuel. It could probably be bought for \$0.05 to \$0.06 per gallon f.o.b. plant.

No. 15 was an entirely different tar product produced in the distillation of hardwood. After the pyroligneous acid has been allowed to stand to settle out the oily tar, it contains a tar-like substance dissolved in it which has to be removed before the liquor can be worked up into alcohol and acetic acid. Since this tar is non-volatile it is removed by completely distilling the pyroligneous acid, the tar being left as a residue in the still. This sample was produced in the laboratory distillation plant from beech, birch and maple. It is a heavy, non-volatile, viscous, tar-like substance, specific gravity about 1.4. Its composition is unknown. It is about 50 per cent. soluble in water, the remainder dissolving in the pyroligneous acid because of the solvent action of the alcohol and acid constituents. The yield of this "dissolved tar" or "residual tar" or "acid water tar" or "copper still tar," which are the various names given to it in the plants, is from 6 to 8 gal. per cord, making a total production in the United States of about 9,000,000 gal. per year. It is strictly a waste product at present, although worth about \$0.025 per gallon as fuel. It could probably be obtained for \$0.04 to \$0.05 per gallon at the plant.

No. 16 is a 15 per cent. water solution of the water-soluble portion of No. 15, just described. The specific gravity of the solution was 1.061 at 21° C. This sample was prepared to determine the value of the water-soluble portion as compared to the whole tar.

There have been many references in the literature to the use of pyroligneous acid for flotation, but no data have been given showing the actual value of this material. These tests included several samples of pyroligneous acid to determine this point.

No. 14 was a semi-commercial pyroligneous acid produced by distilling maple. The specific gravity of the sample was 1.036 at 21° C. It contained 9.7 per cent. dissolved tar, 4.8 per cent. wood alcohol, 11.9 per cent. acetic acid. The value of this liquor for its alcohol and acid content is over \$0.05 per gallon.

No. 12 was a commercial pyroligneous acid from a plant distilling birch and maple, from which the dissolved tar had been removed by distillation. The specific gravity of the sample was 1.008 at 21° C. It contained 3.1 per cent. wood alcohol and 5.9 per cent. acetic acid, making it worth a little over \$0.03 per gallon for its content of these products.

No. 10 was commercial crude pyroligneous acid produced in the destructive distillation of southern yellow pine. The specific gravity of the sample was 1.034 at 21° C. It contained much less alcohol and acetic acid than the hardwood pyroligneous acids, the composition being 1.8 per cent. alcohol, 2.85 per cent. acid and 7 per cent. dissolved tar. Except under abnormal conditions it does not pay to refine pine-wood pyroligneous acid in competition with the hardwood product and it is thrown to

waste. This particular sample would not be worth over $0.0175 per gallon for the alcohol and acid in it. A cord of pine (4,000 lb.) will yield from 125 to 150 gal. of crude pyroligneous acid. It would certainly not pay to buy the crude acid at even $0.02 per gallon f.o.b. plant on account of the freight charges on the large volume of water in it. The concentration of the crude distillate would simply give a thick tar containing a small amount of the acid and none of the alcohol, but if the tar is the valuable constituent this would seem to be the only feasible method of marketing softwood pyroligneous acid for flotation.

### *The Results*

The results of the flotation of these three ores with the above set of oil samples are contained in Tables 1 to 3. In Table 1 the analyses of the concentrate and tailing for each test are given. In the other two tables only the analyses of the concentrate samples are given. Accompanying the analyses of the concentrate samples in each case are the percentages of extraction obtained. The test in neutral, acid and alkaline solutions for each oil is given.

It can be seen that many of the oils give fully as satisfactory results as pine oil for practically all the minerals tested. Oils 11, 18, and 15 are the ones of most interest on account of being immediately available at low prices and because they can be used in small amounts with success. For all-round work, No. 18 seems to have given good extractions of all the metals tested, together with fair grades of concentrate. It will be remembered that this is the crude hardwood tar which has settled out of the pyroligneous acid.

It is a pleasant surprise to find that No. 16, the 15 per cent. solution of No. 15 in water, is so efficient in comparison with No. 15. Here is something which is really not an oil because it is completely soluble in water. It is satisfactory for all the minerals except the copper minerals of ore C.

It will be noticed that the best extractions of the zinc, and generally the highest-grade zinc concentrates, were obtained in solutions that had been acidified. Differential flotation of galena in the presence of sphalerite is seen to be most marked in alkaline or in neutral solutions. No. 14, the hardwood pyroligneous acid, seems to be best adapted for this.

The extractions of the gold in the Cripple Creek ore and the grades of concentrate are surprisingly good. Oils 18, 15 and 16 seem to be capable of producing the highest-grade concentrates. By further treatment of the tailing it should be possible to get even higher extractions, although the grade of the total concentrate would be cut down. No attempts to run tests in this manner were made, as the object of the work was the finding of oils best adapted to do certain things. The relatively high

TABLE 1.—*Tests on Ore A in Neutral, Acid and Alkaline Solutions with Different Oils*

Analysis Head: Lead, 3.4 Per Cent.; Zinc, 4.74 Per Cent.

| Oil No. | Character of Oil | Remarks | Total Time, Min. | Total Oil, Lb. per Ton | Total Acid, Lb. per Ton | Total Alkali, Lb. per Ton | Lead: Per Cent. | Lead: Per Cent. Extraction | Zinc: Per Cent. | Zinc: Per Cent. Extraction |
|---|---|---|---|---|---|---|---|---|---|---|
| 23 | Pine oil | Concentrate | 10 | 0.375 | .... | ... | 55.9 | 53.1 | 12.4 | 62.2 |
| | | tailing. | 10 | 0.375 | .... | ... | 1.6 | .... | 2.3 | |
| | | | 10 | 0.375 | 11.14 | ... | 32.3 | 87.9 | 37.9 | 78.2 |
| | | | 10 | .... | 8.36 | ... | 0.4 | .... | 1.1 | |
| | | | 15 | 0.375 | .... | 8.0 | 48.0 | 89.5 | 25.5 | 79.1 |
| | | | 10 | .... | .... | 2.0 | 0.4 | .... | 4.4 | |
| 11 | Hardwood "wood oil" | ........... | 17 | 1.11 | .... | ... | 57.2 | 92.6 | 20.4 | 58.2 |
| | | | 10 | 0.18 | .... | ... | 0.3 | .... | 2.3 | |
| | | | 16 | 0.94 | 3.68 | ... | 37.9 | 88.7 | 34.2 | 72.6 |
| | | | 15 | 0.17 | 3.68 | ... | 0.4 | .... | 1.4 | |
| | | | 15 | 1.29 | .... | 4.0 | 48.0 | 97.2 | 32.0 | 64.8 |
| | | | 5 | .... | .... | ... | 0.1 | .... | 1.8 | |
| 18 | Crude hardwood "settled tar." | ........... | 17 | 1.35 | .... | ... | 48.6 | 97.2 | 28.0 | 68.4 |
| | | | 3 | .... | .... | ... | 0.1 | .... | 1.7 | |
| | | | 19 | 1.24 | 3.68 | ... | 37.7 | 98.2 | 33.4 | 76.5 |
| | | | 6 | .... | .... | ... | 0.1 | .... | 1.2 | |
| | | | 20 | 1.70 | .... | 2.0 | 48.7 | 98.0 | 27.9 | 63.4 |
| | | | 4 | .... | .... | ... | 0.1 | .... | 1.9 | |
| 15 | "Dissolved hardwood tar." | ........... | 20 | 1.44 | .... | ... | 59.2 | 79.4 | 16.2 | 30.2 |
| | | | 7 | .... | .... | ... | 0.7 | .... | 3.5 | |
| | | | 15 | 1.44 | 5.52 | ... | 32.8 | 88.7 | 13.0 | 69.8 |
| | | | 10 | 0.72 | 3.68 | ... | 0.4 | .... | 1.9 | |
| | | | 20 | 1.44 | .... | 4.0 | 66.7 | 73.0 | 38.1 | 30.3 |
| | | | 8 | .... | .... | ... | 1.0 | .... | 3.4 | |
| 16 | Water solution of No. 15 | ........... | 13 | 1.80 | .... | ... | 55.1 | 90.2 | 22.0 | 43.7 |
| | | | 5 | .... | .... | ... | 0.4 | .... | 2.9 | |
| | | | 19 | 1.80 | 7.36 | ... | 30.3 | 92.2 | 40.5 | 77.2 |
| | | | 4 | .... | .... | ... | 0.3 | .... | 1.2 | |
| | | | 19 | 1.80 | .... | 4.0 | 53.7 | 77.4 | 23.3 | 44.7 |
| | | | 4 | .... | .... | ... | 0.8 | .... | 2.9 | |
| 14 | Crude hardwood pyroligneous acid. | ........... | 20 | 14.00 | .... | ... | 54.3 | 86.2 | 22.7 | 51.5 |
| | | | 3 | .... | .... | ... | 0.5 | .... | 2.6 | |
| | | | 15 | 14.00 | 5.5 | ... | 29.5 | 72.2 | 41.5 | 70.0 |
| | | | 5 | .... | .... | ... | 1.0 | .... | 1.5 | |
| | | | 17 | 12.0 | .... | 4.0 | 62.0 | 54.8 | 9.8 | 42.7 |
| | | | 6 | .... | .... | ... | 1.5 | .... | 3.4 | |
| 12 | Tar-free hardwood pyroligneous acid. | ........... | 13 | 20.0 | 3.68 | ... | 23.5 | 42.7 | 31.7 | 41.4 |
| 10 | Crude softwood pyroligneous acid. | ........... | 17 | 10.0 | .... | ... | 53.8 | 73.3 | 23.9 | 41.3 |
| | | | 13 | 4.75 | .... | ... | 1.0 | .... | 3.0 | |
| | | | 18 | 12.0 | 3.68 | ... | 31.6 | 64.7 | 35.2 | 57.3 |
| | | No cleaner | 7 | 4.00 | .... | .... | 1.3 | .... | 2.2 | |
| | | test. | 12 | 16.00 | .... | 4.0 | 49.1 | 57.8 | 17.9 | 15.0 |

grade of the heads in the tests on the Cripple Creek ore leaves tailing in all these tests too high to compare favorably with roasting and cyaniding. Hence, further work in adaptation of these oils to give higher extractions would have to be done. This is a question of mechanical manipulation with which every flotation experimenter is familiar. The extractions of the iron in the Cripple Creek ore are very poor with all the tests. As the ore was ground dry in a laboratory sample pulverizer this is not surprising, for pyrite is peculiarly sensitive to oxidation during such grinding and will not float well. Not being an important mineral in this case, the method of preparation of the sample did not matter. It is interesting to note that No. 11 and No. 10, the "wood oil" and the softwood crude pyroligneous acid, respectively, were the two oils which gave best extractions of the iron.

As was expected, the Utah Copper ore usually floated best in alkaline solutions, oils 11, 18 and 15 doing the best work.

Further work is being done, especially in the testing of the residues left by concentrating the softwood pyroligneous acid by evaporation, and many other oils are being tested in this manner.

TABLE 2.—*Tests on Ore B in Neutral, Acid and Alkaline Solutions with Different Oils*

Analysis Head: Gold, 0.608 Oz. per Ton; Iron, 5.2 Per Cent.

| Oil No. | Total Time, Minutes | Total Oil, Lb. per Ton | Total Acid, Lb. per Ton | Total Alkali, Lb. per Ton | Gold* | | Iron* | |
|---|---|---|---|---|---|---|---|---|
| | | | | | Oz. per Ton | Per Cent. Extraction | Per Cent. | Per Cent. Extraction |
| 23 | 21 | 0.375 | 0.00 | 0.0 | 4.50 | 81.5 | 8.0 | 10.2 |
| | 21 | 0.575 | 3.68 | 0.0 | 6.42 | 78.7 | 9.0 | 16.5 |
| | 21 | 0.375 | 0.00 | 1.0 | 4.76 | 79.7 | 8.2 | 9.9 |
| 11 | 24 | 1.11 | 0.00 | 0.0 | 4.72 | 82.5 | 8.5 | 31.9 |
| | 22 | 1.11 | 3.68 | 0.0 | 6.12 | 75.8 | 17.2 | 41.1 |
| | 23 | 1.11 | 0.00 | 4.0 | 5.64 | 82.3 | 13.4 | 14.8 |
| 18 | 22 | 1.8 | 0.00 | 0.0 | 10.34 | 78.2 | 13.2 | 17.7 |
| | 21 | 1.5 | 3.68 | 0.0 | 11.52 | 78.0 | 14.3 | 19.6 |
| | 22 | 1.5 | 0.00 | 1.0 | 11.60 | 81.2 | 13.1 | 6.2 |
| 15 | 23 | 2.88 | 0.00 | 0.0 | 5.06 | 82.3 | 5.7 | 0.0 |
| | 21 | 2.88 | 3.68 | 0.0 | 13.80 | 74.8 | 9.6 | 11.8 |
| | 21 | 2.88 | 0.00 | 1.0 | 5.80 | 76.0 | 7.4 | 6.2 |
| 16 | 21 | 2.4 | 0.00 | 0.0 | 10.10 | 81.3 | 8.6 | 31.5 |
| | 21 | 2.4 | 3.68 | 0.0 | 12.60 | 81.2 | .... | 0.0 |
| | 21 | 2.4 | 0.00 | 1.0 | 7.88 | 78.7 | 11.7 | 27.3 |
| 14 | 21 | 12.0 | 0.00 | 0.0 | 5.42 | 79.1 | 9.2 | 32.5 |
| | 22 | 12.0 | 3.68 | 0.0 | 7.22 | 81.7 | 12.0 | 40.3 |
| | 19 | 12.0 | 0.00 | 1.0 | 7.28 | 78.7 | 12.1 | 26.9 |
| 12 | 15 | 18.0 | 0.00 | 0.0 | 5.88 | 76.0 | 7.7 | 32.3 |
| | 20 | 20.0 | 3.68 | 0.0 | 6.76 | 81.7 | 15.0 | 12.8 |
| | 15 | 24.0 | 0.00 | 1.0 | 9.44 | 77.0 | 6.9 | 14.0 |
| 10 | 23 | 16.0 | 0.00 | 0.0 | 5.40 | 88.7 | 6.1 | 40.8 |
| | 21 | 16.0 | 3.68 | 0.0 | 4.98 | 79.6 | 6.0 | 41.2 |
| | 24 | 16.0 | 0.00 | 1.0 | 6.10 | 79.2 | 6.5 | 16.7 |

* In concentrate samples.

TABLE 3.—*Tests on Ore C in Neutral, Acid and Alkaline Solutions with Different Oils*

Heads: Copper, 1.58 Per Cent.

| Oil No. | Total Time, Minutes | Total Oil, Lb. per Ton | Total Acid, Lb. per Ton | Total Alkali, Lb. per Ton | Copper* | |
|---|---|---|---|---|---|---|
| | | | | | Per Cent. | Per Cent. Extraction |
| 23 | 17 | 0.375 | 0.00 | 0 | 3.83 | |
| | 16 | 0.375 | 7.36 | 0 | 8.61 | 28.7 |
| | 17 | 0.375 | 0.00 | 4 | 1.53 | |
| 11 | 20 | 0.92 | 0.00 | 0 | 15.95 | 29.3 |
| | 21 | 1.48 | 7.36 | 0 | | |
| | 21 | 1.29 | 0.00 | 6 | 16.15 | 32.6 |
| 18 | 20 | 1.8 | 0.00 | 0 | 16.45 | 51.2 |
| | 20 | 1.5 | 7.36 | 0 | 2.48 | |
| | 20 | 1.2 | 0.00 | 4 | 22.27 | 68.2 |
| 15 | 21 | 1.44 | 0.00 | 0 | 5.84 | 4.3 |
| | 21 | 1.44 | 7.36 | 0 | 2.77 | |
| | 23 | 1.44 | 0.00 | 4 | 18.17 | 23.1 |
| 16 | 17 | 2.1 | 0.00 | 0 | 4.50 | 33.6 |
| | 18 | 2.7 | 7.36 | 0 | 1.72 | 5.6 |
| | 23 | 2.4 | 0.00 | 8 | 7.94 | 3.9 |
| 14 | 20 | 5.0 | 0.00 | 0 | 2.29 | |
| | 18 | 5.0 | 7.36 | 0 | 1.43 | |
| | 19 | 7.0 | 0.00 | 4 | 12.90 | 13.5 |
| 10 | 19 | 8.0 | 0.00 | 0 | 1.24 | |
| | 19 | 10.0 | 7.36 | 0 | 1.43 | |
| | 12 | 8.0 | 0.00 | 7 | 2.29 | 25.3 |

* In concentrate samples.

TABLE 4.—*Screen Analyses*

| Ore A | Opening | Mesh | Weights | Assays | |
|---|---|---|---|---|---|
| | Mm. | | Per Cent. | Pb, Per Cent. | Zn, Per Cent. |
| Through.......... | 0.589 | 28 | 100.00 | | |
| On.............. | 0.417 | 35 | 0.43 | 0.00 | 0.00 |
| On.............. | 0.208 | 65 | 12.88 | 0.05 | 0.98 |
| On.............. | 0.147 | 100 | 34.46 | 0.07 | 1.37 |
| On.............. | 0.104 | 150 | 23.76 | 0.07 | 2.24 |
| On.............. | 0.074 | 200 | 10.59 | 2.19 | 6.46 |
| Through.......... | 0.074 | 200 | 17.75 | 19.08 | 14.60 |
| Ore B | | | | | |
| Through.......... | ........... | 200 | 100.00 | | |
| Ore C | | | | | |
| Through.......... | 0.295 | 48 | 100.0 | | |
| On.............. | 0.208 | 65 | 2.2 | 1.58 | |
| On.............. | 0.147 | 100 | 16.7 | 1.58 | |
| On.............. | 0.104 | 150 | 16.0 | 1.82 | |
| On.............. | 0.074 | 200 | 53.3 | 1.46 | |
| Through.......... | 0.074 | 200 | 11.8 | 1.73 | |

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Geology of the Warren Mining District

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(Arizona Meeting, September, 1916)

## CONTENTS



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## I. INTRODUCTION

The main object of this article is to present the results of observations by the Copper Queen Consolidated Mining Co.'s geological department as an addition to the already published reports on the Warren district. Since these observations relate mostly to the ore deposits, little will be found here on the general geology that has not been covered by F. L. Ransome's admirable work on the Bisbee Quadrangle.[1] A summary of this very important part of the study of the camp will be given, however, mainly to bring out the new facts discovered during the advance in underground development and the latest detailed study of structures and petrography of the mining area.

## II. PHYSIOGRAPHY

The Mule Mountains, in which the Bisbee ore deposits occur, constitute a chain with a northwest-southeast axis rising abruptly on the sides from the Sulphur Springs Valley on the northeast, as shown in Plate 1, and the San Pedro Valley on the southwest. The range starts at about the International Boundary near Christianson's ranch, with low hills covering a width of 1 or 2 miles, and extending for about 3 miles to the northwest. Here the Gold Hill overthrust fault has caused an abrupt rise in elevation and from this point on the range widens out and the hills become very much more rugged. The highest point is about 6 miles farther. The total length of the range from the International Boundary is about 23 miles and its maximum width, about opposite the town of Bisbee, is 10 miles. The range finally ends at Government Draw, which is a pass about 2 miles wide, connecting the San Pedro and Sulphur Springs Valleys. Beyond this the Tombstone hills commence.

The range is divided into two parts by Tombstone Canyon, a deep cañyon running through the southwest side of the range. It is along this canyon that the Borderland Route road takes its course, and in which the town of Bisbee is situated.

Geologically, also, as seen in Plate 1, the Mule Mountains are roughly divided by Tombstone Canyon into two parts. To the southwest is the pre-Cretaceous tract and to the northeast the Cretaceous. The difference in the two tracts is very marked physiographically. The pre-Cretaceous is very much cut up and is formed of rocks of diverse compositions, consisting of schists, granites, shales, and limestones. These rocks, when subjected to erosion, form a rugged topography with deep canyons and steep cliffs. The Cretaceous tract, on the contrary, is uniform in composition, constituted, for the most part, by soft sandstones, conglomerates,

[1] Geology and Ore Deposits of the Bisbee Quadrangle, *Professional Paper of the U. S. Geological Survey*, No. 21 (1904).

and shales, which are but little faulted. The topography carved from them is characterized by gently sloping hills and draws with almost no cliffs or deep valleys in evidence.

## III. INTRODUCTORY GEOLOGY

The oldest rocks of the range are pre-Cambrian schists and an intruded granite, which are separated from the overlying Paleozoic beds by a profound unconformity.

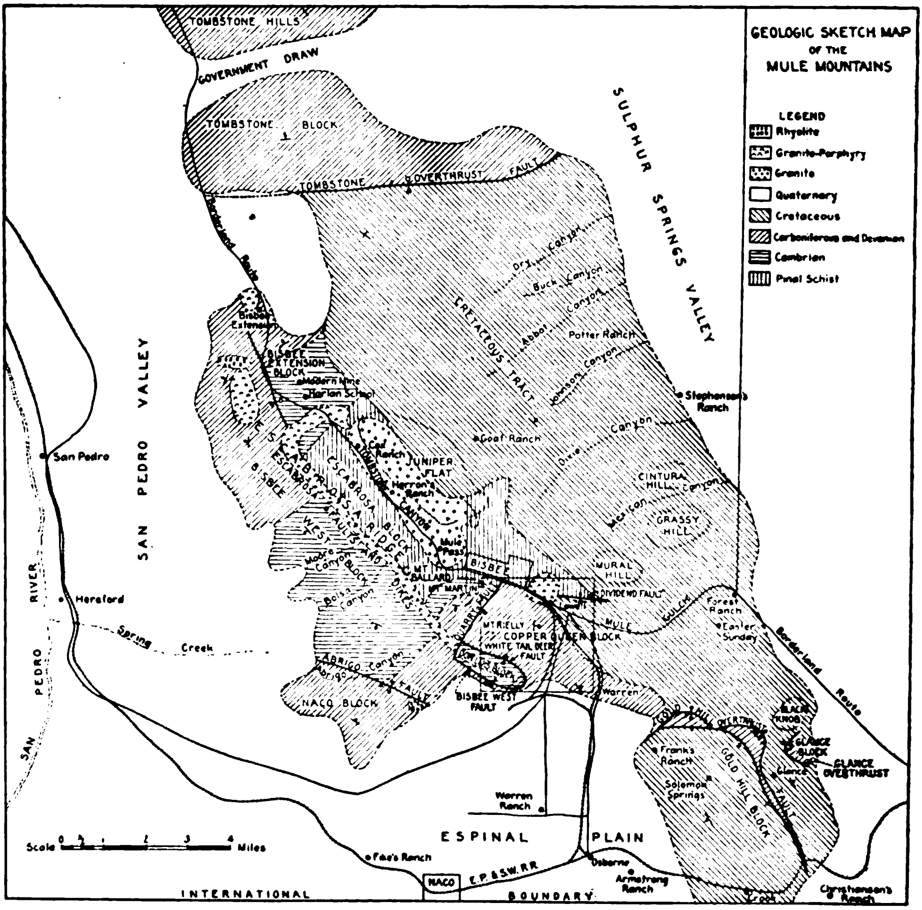



Plate 1.—Geological Map of the Mule Mountains.

The Paleozoic beds represent an era of apparently uninterrupted deposition of sediments, starting with 440 ft. of quartzites having a basal conglomerate, followed by Cambrian limestones, Ordovician-Silurian quartzite, Devonian and Carboniferous limestones. The total thickness of the Paleozoic beds is approximately 5,000 ft.

At the end of the Carboniferous era a violent uplift took place accompanied by extensive faulting and intrusion of granite porphyry. During this uplift, and intimately associated with the intrusion, mineralizing solutions arose from which originated the orebodies of the camp.

Following this came a long period of erosion, which ended with a rapid subsidence, during which time 4,500 to 5,000 ft. of Cretaceous sandstones, shales, and limestones were deposited.

At the end of Cretaceous time a gentle uplift took place, lifting the range again above sea level. This was accompanied by some intrusion of rhyolite and monzonite.

During Tertiary and Quaternary ages, this land area was subjected to erosion, resulting in the present topography, the detritus filling in both the valley of the San Pedro and of the White River, one on each side of the range.

## IV. ROCKS OF THE DISTRICT

### A. Sedimentary Rocks

Reference to Plate 2, showing a generalized geologic section, will help make clear the following description.

#### *Pinal Schist*

*Name.*—The oldest rock of the district is the Pinal Schist. It is composed of a uniform series of thinly laminated siliceous mica schists of unknown thickness. From the similarity in texture and stratigraphic relation to the later sediments, it has been correlated by Ransome[2] with the underlying schistose complex of the Pinal Range, which he called the Pinal Schist.

*Distribution and General Structure.*—The Pinal Schist, being the basal crystalline rock upon which rest all the younger formations of the district, is exposed everywhere that erosion has stripped them off, and is encountered underground on sinking through them.

The best surface exposures are northeast of Mule Gulch and to the southwest of Tombstone Canyon. Smaller exposures are also found about 1 mile to the northwest of the town of Don Luis. Underground it is exposed where the Dividend Fault is cut, both in Copper Queen and in Denn ground. It also is found as dragged-in fragments in the contact-breccia mass around Sacramento Hill.

*Lithology.*—The color varies from light to dark gray, with tinges of green on fresh surfaces, and rusty in weathered specimens. The cleavage is rather imperfect, having a shiny satin-like surface.

The microscope shows the rock to be composed mainly of quartz and sericite. Sometimes the sericite is replaced in part by penninite, chlorite,

[2] *Loc. cit.*, p. 24.

or serpentine. Folded into the schist, and forming an integral part of it, are occasional masses of diabase.

*Origin and Age.*—The age of the Pinal Schist is assumed to be the same as for the Pinal Schist of the Globe district which Ransome[3] has shown to be derived from arenaceous sediments, probably of Algonkian age. Both at Globe and at Bisbee a profound unconformity exists between it and the overlying Paleozoic beds.

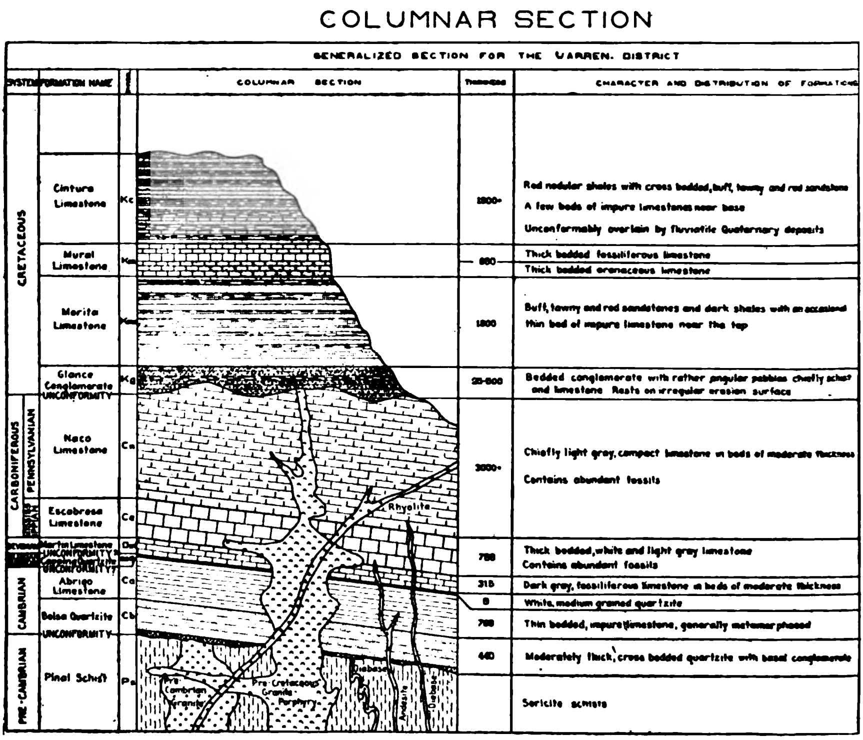



**Plate 2.—Generalized Geologic Section Showing Rock Strata of the Warren District.**

## *Bolsa Quartzite*

*Name.*—The name of this formation, as well as those for all other formations in the district, were given by Ransome[4] and have been generally adopted.

*Distribution and General Stratigraphy.*—This Bolsa Quartzite lies on an evenly eroded plain of Pinal Schist. The main exposures of this formation are southwest of Escabrosa Ridge, especially in Bolsa, Abrigo, and Quarry Canyons. In the last-named canyon a complete unfaulted

[3] *U. S. Geological Survey*, Globe Folio, No. 111, p. 2.

[4] *Loc. cit.*, p. 28.

section occurs. Underground, the formation has been cut by diamond-drill holes from the bottom of the Irish Mag shaft, and also by the workings from the Wade Hampton shaft.

*Lithology.*—At the base of the formation is a well-marked basal conglomerate, much iron-stained, containing pebbles consisting mostly of white, well-rounded quartz of up to a 4-in. diameter, and averaging about 1 in., the cementing material consisting of subangular sand. In the northwestern part of Tombstone Canyon some pebbles of porphyritic rock are found, together with some rounded crystals of feldspar. These feldspar crystals consist of pink microcline and microperthite, such as are found in the Juniper Flat granite less than a mile away. These feldspars, notwithstanding their being subject to such long exposure, are still relatively fresh.

The next 100 ft. of the Bolsa shows alternating beds of fine conglomerates and quartzites, with some arkose beds. Then follows a finer whitish quartzite with frequent cross-bedding, succeeded by some massive, dense beds of a maroon-colored quartzite. The total thickness of the Bolsa Quartzite as measured in Quarry Canyon is 440 ft.

*Age.*—No fossils have been found in the Bolsa except sparse worm tracks near the top, but as the next formation lies conformably on top of it and shows middle Cambrian fauna, the Bolsa has been assumed as lower or middle Cambrian.

### *Abrigo Limestone*

*Distribution and General Stratigraphy.*—The Abrigo formation follows the Bolsa Quartzite conformably, and is the first fossil-bearing formation in the district.

The best exposure of Abrigo limestone is found on the northern slope of Mount Martin, where a complete unfaulted section is obtainable. Large areas of Abrigo are also found on the southwestern slope of Escabrosa ridge in Abrigo, Moon, and Bolsa Canyons, but the formation is here badly faulted. Small areas are also found in Quarry Canyon, north of Don Luis, and on the southern slope of the range from Mount Reilly to Gold Hill. As the beds are, for the most part, soft shales and shaly limestones, good exposures are rare, the formation being usually covered by talus.

Underground, the Abrigo formation is cut by the lower levels of all the mines from the Higgins to the Briggs, but in most of them only the top 100 ft. of rock has been cut. At the Spray, Oliver, and Irish Mag shafts, however, the formation has been cut by their lower workings to a depth of 400 ft. below its top, and diamond-drill holes from the Irish Mag shaft have penetrated into the Bolsa. At the White Tail Deer and Wade Hampton shafts the lower beds have also been cut. No complete section, however, has been exposed underground. Measured at Quarry Canyon the thickness of the Abrigo was found to be 798 ft.

Geological section of Abrigo at Quarry Canyon:

| | |
|---|---|
| Thin shales, greenish-gray to pink-brown, highly epidotised.. | 0–72 |
| Gray broken calcareous limestone, epidotised | 72–83 |
| Shaly calcareous sandstones | 83–112 |
| Medium-grained greenish-gray limestone with epidotised beds | 112–132 |
| Pure fine-grained pinkish-gray limestone | 133–134 |
| Arenaceous pinkish-gray shales | 134–136 |
| Purer fine-grained dark-gray limestone | 136–139 |
| Light-green shales | 139–141 |
| Fine-grained limestone, thin wavy bands of epidote | 141–146 |
| Green shales | 146–151 |
| Succession of impure lime and shales | 151–183 |
| Fine-grained gray limestone, beds of epidote ½ to 1 in. thick. | 183–229 |
| Purer light-gray limestone | 229–245 |
| Coarse sandy limestone with epidotised beds | 232–245 |
| Thin greenish shales | 245–253 |
| Coarse limestone, very fossiliferous (worm tracks) | 253–261 |
| Arenaceous green limestone | 261–278 |
| Coarse limestone, thin epidotised bands, some purer, 2-ft. beds forming steps on surface | 278–412 |
| Impure limestone green to brown, thin wavy bands of epidote very characteristic on surface by the weathering out of the epidote | 413–497 |
| Shaly arenaceous limestones | 497–521 |
| Coarse gray arenaceous limestone | 521–629 |
| Coarse light-gray crystaline sandy limestone | 629–684 |
| Coarse cherty-banded crystalline white bed-forming cliff | 684–700 |
| Sandy even-grained limestone and quartzite | 700–798 |

The Abrigo formation consists of a series of shales and argillaceous and sandy limestones, with bands of epidote. These epidote bands weather out on exposure to the atmosphere, giving the formation a characteristic cherty banded appearance. In the top beds there are a few bands of true chert. Underground, this epidote appears as greenish bands giving a wavy banding which readily distinguishes it from all other formations, when the beds are fresh. About 100 ft. from the top there exists a 16-ft. bed of pure white limestone which forms a rather prominent cliff when encountered on the surface.

### *Capping Quartzite*

Lying on top of the Abrigo, and with no apparent angular unconformity, is a persistent bed of pure quartzite varying in color from white to dark red, and varying from 6 in. to 12 ft. in thickness, and locally called the Capping Quartzite.

*Distribution.*—This quartzite being more resistant than the underlying and overlying beds, is exposed wherever the Abrigo outcrops, as a small cliff or shelf. One well-marked exposure is found to the east of Black Gap, where it runs up the side of the hill apparently as a siliceous dike. Other good exposures are found on Mt. Martin, and southwest of Escabrosa Ridge.

Underground, this bed is important as a marker. Due to its character, it is resistant to metamorphism, and may be recognized easily even when the rest of the sediments have been completely changed. Its thinness, however, tends to make it elusive, as a small fault frequently will throw it out of an otherwise perfect section.

*Age.*—As one travels north through the State, this formation becomes thicker, and in the Globe district it attains a thickness of 500 to 700 ft., while that of the underlying limestone has decreased to 200 ft. Here there is a well-marked evidence of an unconformity between the limestone, which is capped by a flow of basalt of varying thickness, and the overlying Troy Quartzite, which is conglomeratic at its base. While no fossil evidence is available, its relation to the overlying and underlying formations seems to indicate that it is probably of Silurian age, and that the Ordovician is represented by a period of erosion, when the area studied was very little above sea level, since there is no apparent angular unconformity below or above this formation.

*Martin Limestone*

Lying conformably above the Capping Quartzite is 300 ft. of limestone known as the Martin limestone.

*Distribution.*—The exposures of the Martin limestone are widespread, especially along Escabrosa Ridge, north of Moore Canyon, in Abrigo Canyon, Escacado Canyon, and on the southern slopes of the hills east of Mt. Reilly. Owing to the fact that the overlying Escabrosa limestone is a prominent cliff-forming bed, the exposures are apt to be talus-covered. The best complete sections are obtained on the northern slope of Mt. Martin, and to the east of Black Gap. Underground, it is the formation most frequently cut of any in the ore zone. For this reason it is usually difficult to recognize from its lithological characteristics. The formation is so rich in fossils, however, that even when much altered, a diligent search will yield some trace of them. The formation is encountered underground in every mine in the district except the Wade Hampton, which is entirely in Cambrian, and the Denn which did not sink deep enough to strike it. The outcrop of this formation was first found in the Queen opencut near its top, and it has been encountered at progressively lower levels from here down to the 1,600-ft. level of the Lowell, and the 1,500-ft. level of the Briggs.

*Lithology.*—The Martin limestone is the most constant formation, both in respect to thickness, and character, of any in the district. Plate 3 shows a comparison of sections obtained over widely scattered parts of the district. As is seen, the formation as a whole is made up of fairly dense dark-gray limestone with occasional shaly and sandy members, but all with enough lime present to be called limestone. It is all more or less fossiliferous, and some beds are extremely so. The characteristic fossils are three brachiopods, one a spirifer, one an atrypa, and

one a schisophoria, which all attain a length of about ⅓ to ½ in.; they tend to weather out on both surface and underground exposures. When the rock is altered, these fossils tend to resist metamorphism to a

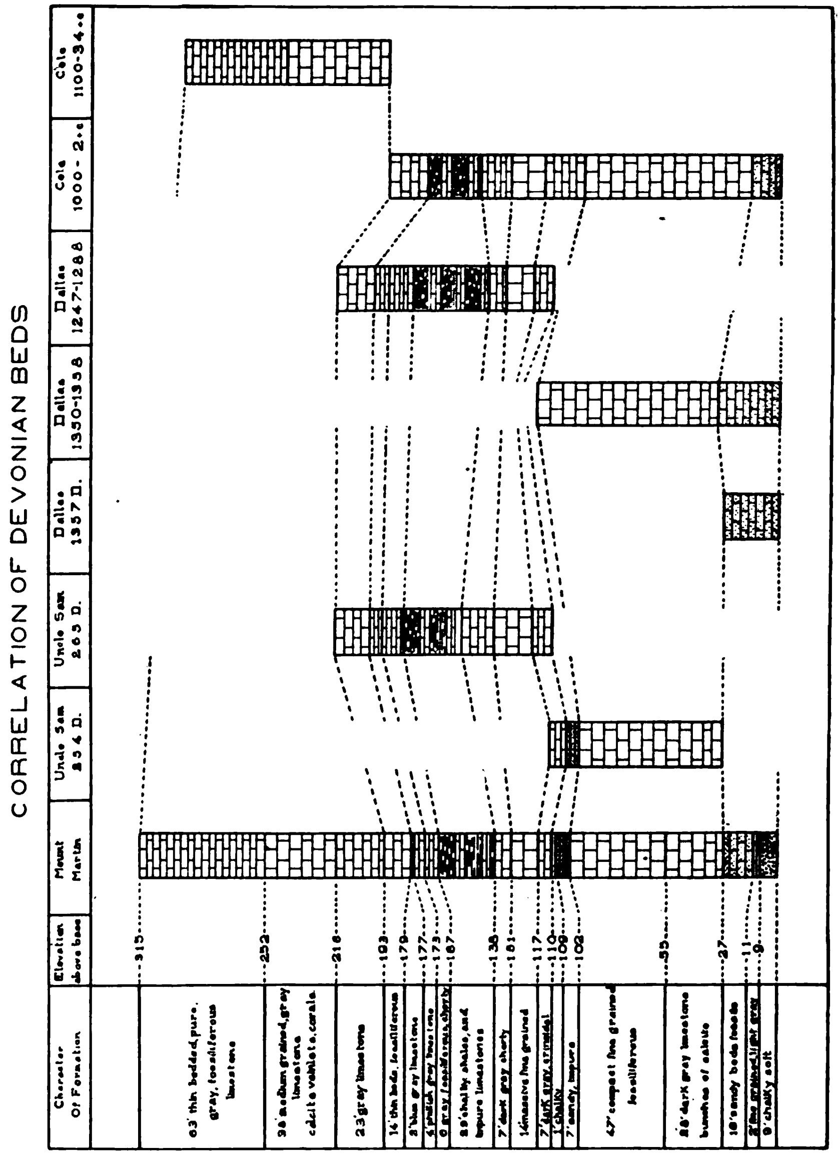



PLATE 3.—COMPARISON OF SECTIONS OF MARTIN LIMESTONE IN VARIOUS PARTS OF THE WARREN DISTRICT.

remarkable degree, and it is not at all rare to find almost perfect fossils left in a soft mass of completely altered limestone. These fossils, however, are frequently found completely pyritized when the surrounding

limestone is untouched. In one case, within a completely oxidized orebody of the Martin, fossils were found almost perfect in form but completely replaced by azurite. Fortunately, the fossiliferous beds themselves seem to be more easily replaced by ore solutions, so that where metamorphism is most complete, some fossil evidence may usually be found.

The upper beds of the formation have few brachiopods, but are almost made up of corals. On altering, these beds tend to take a pseudo wavy banding, which makes it difficult to distinguish from the Abrigo. The succession of beds in these cases is the only determining factor.

The Martin limestone is not a pure limestone by any means. Almost the whole series is more or less dolomitic, more so than any other formation in the range. It is also much more aluminous than a superficial examination would lead one to suppose. The silica content is also high. Characteristic partial analyses of unaltered specimens of Abrigo and Martin limestones from both surface and underground exposures are given in Table 1. These determinations were made at the Copper Queen assay office.

TABLE 1.—*Analyses of Unaltered Abrigo and Martin Limestones*

| | Abrigo | | | | | Martin | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Mt. Martin Section Underground | | | | | | | | | | | | |
| | Shales | 72–232 ft. above base | 232–521 ft. above base | 521–700 ft. above base | Cherty Limestone 1,357 Drift | Black Gap[4] | 500 Level Uncle Sam Mine[3] | | | 265[3] Drift | 1,300[1] Dallas | 1,200[2] Dallas | 1,000[5] Cole |
| $SiO_2$... | 68.6 | 30.8 | 35.3 | 16.1 | 49.4 | 10.3 | 39.4 | 46.0 | 21.7 | 15.5 | 45.5 | 35.5 | 25.2 |
| $Al_2O_3$.. | 9.1 | 4.0 | 3.0 | 2.1 | 4.5 | 2.0 | 9.1 | 10.8 | 6.5 | 3.5 | 12.9 | 5.0 | 5.9 |
| MgO... | 4.6 | 3.6 | 2.3 | 1.7 | 2.4 | 14.2 | 8.8 | 4.4 | 13.8 | 22.9 | 3.8 | 5.4 | 8.5 |
| CaO... | 9.8 | 33.5 | 31.5 | 38.9 | 17.8 | 28.5 | 6.0 | 10.5 | 17.8 | 10.7 | 13.1 | 26.7 | 24.7 |
| Fe..... | 5.1 | 2.7 | 2.8 | 1.5 | 2.2 | 1.2 | 3.7 | 3.2 | 2.0 | 1.8 | 3.0 | 2.1 | 1.8 |
| S...... | ... | .... | .... | 0.2 | 0.6 | 0.2 | 1.5 | 0.4 | 0.7 | 1.3 | 1.9 | 0.8 | 0.8 |
| Mn.... | ... | .... | .... | .... | .... | .... | 0.6 | 0.4 | 0.4 | 0.3 | | | |

[1] Bottom beds. [2] 100 ft. above Capping Quartzite. [3] Somewhat metamorphosed. [4] Composite of section. [5] 200 to 300 ft. above Capping Quartzite.

As is seen, the two formations are of about the same purity, but the Abrigo contains more silica than the Martin. The Martin, however, is much more dolomitic, and the lower beds are very shaly. The composite from Black Gap shows that on the whole it is a fairly pure dolomite, with only 13 per cent. of silica, alumina, and iron. The impure beds are chiefly at the bottom 100 ft. and the top 50 ft. of the formation.

*Age.*—The age of the Martin has been well established by good fossil evidence as Devonian. The best exposition on the correlation of these beds is found in the previously mentioned Ransome's report on the Bisbee Quadrangle.

### *Escabrosa Limestone*

*Distribution.*—The best surface exposures of this formation are on Escabrosa Ridge from Mt. Martin to Mt. Reilly, and on the southern slopes of the hills from Mt. Reilly to Gold Hill. The formation again appears to the northward on Escabrosa Ridge southwest of the Modern mine, and an excellent section is obtained from where the Borderland Route turns out of Tombstone Canyon into the San Pedro Valley. Underground, the formation is cut by all the shafts from the Czar to the Briggs. It is usually in the oxidized portions of the mines, and as a consequence exposures are not very good.

The best exposures found are at the Junction and Sacramento mines. At the Junction, workings cut the formation from the top to the bottom. Here, except for marbleization, the exposures are good, as oxidation has not penetrated nearly as deep as in the other mines of the camp. At the Sacramento, exposures are obtained of the bottom 300 ft., but marbleization has so altered the formation that they are not good.

*Lithology.*—The Escabrosa and Naco formations represent really one period of deposition during Carboniferous time. The division of the formation as a whole into Escabrosa and Naco is largely an arbitrary matter. Roughly it is divided, from the fossil evidence, into Mississippian and Pennsylvanian, the former locally called Escabrosa, and the latter Naco. This line of subdivision lies at approximately 800 ft. above the top of the Martin formation. In this paper the thickness of the Escabrosa has arbitrarily been taken as 768 ft. irrespective of fossil evidence. Following is a detailed section of Escabrosa taken at Mt. Martin and northeast of the White Tail Deer shaft.

| | Feet |
|---|---|
| Coarse crystalline pink non-fossiliferous limestone | 0–6 |
| Pinkish-white to gray medium-grained 6-in. to 12-in. beds | 6–60 |
| Massive grayish-white coarse fossiliferous cliff beds | 60–140 |
| Thin gray beds medium fine-grained fossiliferous | 140–145 |
| Coarse massive grayish fossiliferous cliff bed | 145–185 |
| Thin fine-grained fossiliferous gray beds | 185–312 |
| Fine-grained grayish-white cherty banded (½ in. to 3 in.) | 312–479 |
| Coarse massive white cliff beds fossiliferous | 479–556 |
| Fine-grained thin-bedded fossiliferous | 556–608 |
| Brown cherty fossiliferous limes | 608–616 |
| Grayish fine-grained thin-bedded fossiliferous | 616–648 |
| Pinkish beds medium-grained with some chert | 648–686 |
| Fine-grained thin-bedded soft grayish | 686–758 |
| Massive dark-gray medium-grained | 758–768 |

The Escabrosa limestone rests conformably on the Martin limestone wherever exposed. The 6-ft. pink non-fossiliferous bed forming the bottom member serves in unaltered ground as an excellent marker. The pure coarse crystalline bed following is the most commonly exposed

member of all, forming as it does the steep and rugged cliffs of Escabrosa and Higgins Ridges. At the Mt. Martin and the White Tail Deer sections, it contains very little chert and abundant crinoid stems. Underground exposures appear to contain considerably more chert in this horizon. In fact, the presence or absence of chert beds seems to be very local, and untrustworthy in identifying any particular horizon. Above this first 180-ft. cliff-forming horizon is a soft series of more dense fine-grained beds with some chert, followed by 127 ft. of extremely cherty banded medium-grained limestone, which seems to be very persistent. Above this is a second thinner cliff-forming horizon 77 ft. thick, followed by dense, relatively impure limestone, varying in color from brown to gray with quite abundant chert nodules. There is no definite marker between the Escabrosa and Naco. As previously stated, the Escabrosa is given arbitrarily a thickness of 768 ft.

### *Naco Limestone*

*Distribution.*—This limestone is generally exposed on the top of the hills from Queen Hill to the east, and extends over the productive area from the Irish Mag shaft continuously east until covered by the later conglomerate. Good exposures of Naco are also found at the northermost range of hills just south of Government Draw, and on Escabrosa Ridge from the Modern mine, to the northwest, where it is finally covered by Quaternary wash. The best and least faulted section of the formation is found in the Naco Hills about 3 miles west of the town of Don Luis. Underground it is cut by all the shafts from the Irish Mag eastwards. Practically no drifts, however, have been driven in this horizon, except possibly at the upper levels of the Junction mine, and the 200-ft. level of the Lowell.

*Lithology.*—A thickness of 1,500 ft. has been measured in the Naco Hills. Assuming that the formation has been half stripped off we get a probable former thickness of at least 3,000 ft. Compared with the Escabrosa limestone, it is generally thinner bedded, and very much finer grained, and contains some distinctly shaly members, especially in the upper portions. Some coarse-grained beds do exist, however, which without fossil evidence would be impossible to distinguish from the Escabrosa. Abundant crinoids exist as in the Escabrosa, but large brachiopods and cephalopods are much more in evidence. The most common and characteristic Naco fossil is a large-sized member of the productus family.

### *Cretaceous Deposits*

*Glance Conglomerate.*—Resting on an extremely uneven erosion surface of Paleozoic beds, schist and granite, is deposited the Glance Conglomerate. This formation is exposed over almost the whole of the southern end of the range.

*Lithology.*—This conglomerate is made up of partially rounded frag-

ments of schist, porphyry, both fresh and sericitized, quartzite, and limestone. These fragments vary in size from ½ in. to 3 ft. diameter and are loosely compacted. Almost no cementing material exists. The conglomerate as a whole is stained a deep brown. The thickness varies from nothing up to 1,000 ft., depending on the old erosion surface. Where this old surface was fairly well worn down, as north of the Dividend Fault and on top of the Juniper Flat granite mass, the formation is a typical basal conglomerate with rounded pebbles cemented with sand.

*Age.*—The age of the formation is either late Jurassic or early Cretaceous.

*Morita, Mural and Cintura Formations.*—For a more detailed description of the Cretaceous beds above the Glance Conglomerate, Ransome's report may be referred to. As these beds occur outside the productive area, no additional study was made of them. The Morita consists of a series of buff and tawny sandstones, shales and calcareous sandstones, containing sparse fossil remains, and measuring 1,800 ft. in thickness. This is overlain conformably by 650 ft. of limestone with a 50 to 200-ft. member of pure limestone containing abundant Cretaceous fossils.

The Mural is followed by the Cintura formation, consisting of beds almost identical with those of the Morita, with a thickness of 1,800 ft., plus an unknown amount eroded off. These Cretaceous beds cover about two-thirds of the range, nearly the whole area northeast of Tombstone Canyon and Mule Gulch, and are finally buried beneath the Quaternary wash of Sulphur Springs Valley.

## B. Igneous Rocks of the District

The most important igneous rocks of the district, those of granitic composition, have been grouped together in the United States Geological Survey publications on this region, and have all been ascribed to the same magma and general period of eruption, although differences in composition, texture and field relations were noted. Notman[5] has already separated the granite and the porphyritic rocks by placing the first in the pre-Cambrian, and Boutwell has found a basic variety of porphyry underground and on surface, which corresponds probably to the post-Cretaceous dike described by Ransome from the Glance mine,[6] this latter being a rock easily distinguished from the other porphyry.

Careful search for field and petrographic evidence has yielded results which allow the following classification of the igneous rocks of the Warren district.

### *Pre-Cambrian Granite*

On Juniper Flats and along the north side of Tombstone Canyon, a mass of granite is exposed which forms the center of the Mule Moun-

[5] *Transactions of the Institute of Mining and Metallurgy*, vol. 22, pp. 550–562 (1913).
[6] *U. S. Geological Survey, Professional Paper* No. 21, p. 84 (1904).

tains. The rock is a pink or purplish-gray coarse-grained granite whose component minerals are quartz, orthoclase, microcline, a little biotite and plagioclase feldspar. Accessory minerals can be seen only under a microscope in thin section, such as apatite, magnetite, zircon and tourmaline. The orthoclase contains microperthitic intergrowths of albite, and dusty inclusions of iron oxides.

This rock corresponds exactly to that described by Ransome as the Juniper Flats Granite, an analysis of which is given in *Professional Paper* No. 21 *of the United States Geological Survey*, as follows:

| | Per Cent. | | Per Cent. |
|---|---|---|---|
| $SiO_2$ | 75.86 | $Na_2O$ | 3.60 |
| $Al_2O_3$ | 12.17 | $K_2O$ | 5.04 |
| $Fe_2O_3$ | 0.85 | $H_2O-$ | 0.27 |
| FeO | 0.36 | $H_2O+$ | 0.72 |
| MgO | None | $T_1O_2$ | 0.21 |
| CaO | 0.62 | | |
| | | | 99.70 |

There are on Juniper Flats some porphyritic alaskite dikes that seem to be directly related to the granite, but aside from these, all other porphyritic rocks here have not been ascribed pre-Cambrian age.

*Field Relations.*—The Juniper Flats granite has been found cutting only Pinal Schist, and while it has not been found directly overlain by Paleozoic sediments but covered directly by Cretaceous, pebbles of microcline and of microperthite granite have been found at the base of the Cambrian quartzite. This granite nowhere cuts the Paleozoic sediments, but it is itself extensively intruded by dikes and irregular masses of the later porphyries that do penetrate the sedimentaries. These observations, as well as the similarity of this granite to other pre-Cambrian granites in Arizona has led to the belief that the intrusion took place before the beginning of the Paleozoic.

In the area economically exploited in the district for copper, the Juniper Flats granite has not been found, and the only ore associated with the rock comes in fissure veins filled with quartz and some fluorite with pockets rich in gold.

### *Pre-Cretaceous Granite Porphyry*

The most important igneous rock in the district is the granite porphyry that is associated with most of the orebodies, and whose largest outcrop is the mass of Sacramento and Copper King Hills in Bisbee.

*Distribution.*—On surface this rock is found abundantly in the southwest, or older portion, of the Mule Mountains, but the distribution is by no means even. In the schist, granite, and Cambrian formations there are abundant masses of porphyry in the form of dikes and sills, while in the Carboniferous only a few well-marked dikes are found besides the main stock of Sacramento Hill.

In Juniper Flats and on Escabrosa Ridge there is abundant granite

porphyry, some of the dikes cutting granite, schist and Paleozoics in their course from the north to the south side of Tombstone Canyon. On one of the ridges close to the mouth of Tombstone Canyon, about opposite the old Modern mine there is a continuous sill of granite porphry several hundred feet thick in Carboniferous limestones. This sill is remarkable because it is cut by dikes of rhyolite, and because on an old erosion surface of the granite porphyry and Carboniferous limestones a considerable thickness of rhyolitic lavas has accumulated.

The character of these later dikes and flows is exactly the same as that of the volcanic plug with flow structure which is found in the Naco hills and is mentioned by Ransome in his report on the district. The relations in the locality at the mouth of Tombstone Canyon leave little doubt as to the great difference in age between the granite porphyry and the rhyolite, as the erosion surface on which the rhyolite flowed is probably post-Cretaceous.

In the mining area of the district the Sacramento Hill stock has penetrated between the Paleozoic sediments on the south side of the Dividend fault and the schist on the north. The outline of the stock is irregular, and extends about a mile from north to south and an equal distance along the fault. The only other porphyry outcrop of any size in the mining area is the Shattuck dike, which runs from above the Spray mine for more than a mile westward. Another important dike, the Sacramento, outcrops a few isolated places in the Naco limestone, southward from Sacramento Hill almost to the Briggs mine.

In the underground mine workings, the relative abundance of porphyry intrusions in the formations lower than the Carboniferous holds about the same as on surface. Besides the main mass of the Sacramento Hill intrusion there is another smaller, laccolithic mass of porphyry around the Lowell mine. This body is connected with Sacramento Hill and Sacramento dike by masses of porphyry, none of which reach the present surface, but which spread out mostly at the Carboniferous-Devonian horizon. This mass, with its accompanying contact breccia is shown in Plate 6.

The mode of intrusion of the granite porphyry and its relation to the structure of the older rocks will be touched upon later. It will be enough to state here, however, that the intrusions took place in one general period, but not all at once, nor were the conditions the same at the beginning and end of the eruptions, or the composition of the rock exactly the same. Considerable alteration was also started in the rocks before the last porphyry was injected.

*Lithology.*—Although there is abundant fresh granite porphyry in the district most of the rocks around the ore zones have been so altered by mineralizing solutions that it is difficult to find material which represents the original granite porphyry of the main masses associated directly with the ores. Fortunately, the alteration processes have been so varied

that they have sometimes left untouched some of the phenocrysts, at other times the base, so that from various sources a reconstruction can be made of what was for a short time the rock of Sacramento Hill and the Lowell mass. Of the dikes fresh rock is available.

The pre-Cretaceous granite porphyry is, when fresh, a light-gray rock weathering to a light-red color on surface. It has abundant phenocrysts, up to ½ in. in diameter of feldspar, quartz and biotite, the relative amounts of these being variable. The base is a fine-grained mass, almost glassy at times and composing from one-half to about eight-tenths of the rock. The microscope shows that the rock varies from a typical granite porphyry to a rather basic monzonite porphyry, sometimes in the same sill or dike. The main masses were probably of the more acid variety, with well-developed quartz phenocrysts that have embayments due to reabsorption. There are no noticeable inclusions in the quartz. In the more basic varieties of the porphyry, quartz phenocrysts become extremely scarce.

The feldspar phenocrysts are mostly orthoclase with albite and oligoclase increasing as the rock becomes more basic. The orthoclase is in large crystals when it predominates and in small ones when plagioclases are abundant. The only intergrowth of the orthoclase seen is in the form of a micropegmatite with quartz, and at the rim of some large crystals in the dikes where an aureole of quartz and orthoclase usually forms around a pure orthoclase of quartz center. The plagioclases are all between albite and oligoclase and sometimes are in greater abundance and size than the orthoclase.

Biotite is the only ferromagnesian mineral of any importance and comes in large well-formed crystals as well as fine shreds. In most of the porphyry it has been altered before any other minerals, and its traces cannot be distinguished megascopically, but in some of the more basic varieties, which are also chloritized, biotite is the most conspicuous phenocryst mineral and gives a decidedly dark color to the rock.

The base of the porphyry is usually a cryptocrystalline mass of quartz and feldspar, found sometimes with a micropegmatitic intergrowth, at other times with incipient spherulitic growth and a few times with a holocrystalline but extremely fine crystallization.

As accessories, there are: apatite in well-formed needles sometimes noticeably abundant, and generally in the biotites; very few grains of pyrite, and dusty undeterminable inclusions in the feldspars.

The foregoing description of the unaltered porphyry is rarely applicable to the rock encountered in the mining area, as the porphyry may be found in the form of a white mass of sericite and quartz, or a dense dark-green rock composed mostly of serpentine and penninite, with all variations between. These characteristics will be considered later under metamorphism and mineralization.

*Age.*—The granite porphyry distinctly cuts the Pinal Schist, granite

and all the Paleozoic sediments, while pebbles of the altered sericitized and oxidized porphyry are found at the base of the Cretaceous Glance Conglomerate. It is, therefore, certain that the age of intrusion is post-Carboniferous and pre-Cretaceous. That the porphyry intrusion did not take place all at once, is shown by dikes of the slightly more basic varieties cutting the contact breccias of the more acid ones. This difference in age is further emphasized by differences in alteration, but, as will be shown later, there is no reason to believe that there was any great interval of time between the porphyry injections, but that they all belong to the same general period.

### *Rhyolite*

This rock has not been distinctly recognized in the mining area of the Warren district, but due to its great similarity to the granite porphyry it may have been confused with the older rock, especially where both are found in dikes. The rhyolite from the mouth of Tombstone Canyon, whose field relations have already been given, is a gray to pink rock with decided flow structure in its surface forms. Phenocrysts are not abundant and are seen under the microscope to be entirely quartz and orthoclase, in an aphanitic or cryptocrystalline groundmass. Some of the dikes on Escabrosa ridge, and others northeastward from the mouth of Tombstone Canyon, have such a glassy groundmass, and so few phenocrysts of quartz and orthoclase, that they may very well be of the late rhyolite, especially as they are seen to cut other porphyritic dikes in places. The rhyolite of the Naco Hills described by Ransome is probably of the same age as that at the mouth of Tombstone Canyon.

*Age.*—Unfortunately no information is available at present for placing the age of the rhyolite any closer than post-granite porphyry. An erosion period separates the two, but whether it is the pre-Cretaceous or post-Cretaceous one cannot be definitely stated.

### *Other Dike Rocks*

*Hornblende Andesite.*—At a few scattered places on the surface, and in the Shattuck, Cole, Wade Hampton and Wolverine mines, there are dikes of a dark-green, fine-grained rock, which is seen under the microscope to be composed of abundant small phenocrysts of andesine and labradorite and a few of hornblende, in a microcrystalline base of feldspar and fine magnetite grains. This andesite cuts all the pre-Cretaceous formations, including the granite porphyry, but its relation to the ores is not clear. The rock is considerably altered, and chloritic minerals especially penninite, as well as some pyrite, have been developed. In a specimen from the Wolverine shaft there is also some sericitization, so it is possible that the andesite may antedate the last of the mineralization period.

*Diabase.*—Olivine and augite diabase in small masses and dikes have

been found to belong to two distinct periods. The earlier is folded in with the Pinal Schist and is pre-Cambrian, while the latest cuts the Carboniferous limestones and is entirely unaltered. There is no known exposure of them underground.

*Monzonite Porphyry.*—The dikes of this rock mentioned by Ransome as occurring around the Glance mine have not been studied during the preparation of this report, and no similar rock has been found in the mining area, unless it is the one classified as andesite porphyry, or the basic porphyry found by Boutwell.

## V. STRUCTURAL GEOLOGY

### General Structure of Mule Mountains

#### *Fault Blocks and Other Divisions*

In this report the structural geology will be confined almost entirely to the southwestern part of the range covered by pre-Cretaceous sediments, as these play the most important part in the relation to the ore-bodies of the district. Reference to the areal geology map, Plate 4, and sections made along its lines *A–A*, *B–B*, and *C–C* shown in Plates 5, 6, and 7, will help make clear the following description:

As previously stated, the range is roughly divided by Tombstone Canyon and Mule Gulch into two parts. To the northeast, and forming about two-thirds of the range, Cretaceous sediments cover the whole area, dipping gently to the northeast and north, and being very little faulted. Where cut by the above-mentioned canyons, they overlie a nearly level plain of schist and granite. This area will be alluded to as the Cretaceous Tract.

The area southwest of the two canyons is almost entirely made up of much faulted and intruded schists and Paleozoic sediments, covered at the southeastern end by Cretaceous sediments overlying a rough topography of Paleozoic beds. It is this area which is studied in detail in this report.

Referring to the accompanying sketch map (Plate 1) of the range, it is seen that in a general way the dips of the Paleozoic beds radiate around the Juniper Flat granite mass. At the northwestern end they dip to the north and northwest. At Mt. Martin and Mt. Ballard, the dip swings to the west and southwest. From Mt. Martin to the southeast, the dips change from south to due east. Each change in dip and strike, however, is generally marked by a fault, the beds seeming to withstand folding to a remarkable degree.

In our division of this area into fault blocks, the divisions given by Ransome[7] have been generally followed, with a few minor exceptions and additions.

Starting at the north is the Bisbee Extension Block with beds dipping to the northwest, and disappearing finally to the north under Quarter-

[7] *Loc. cit.*, p. 93.

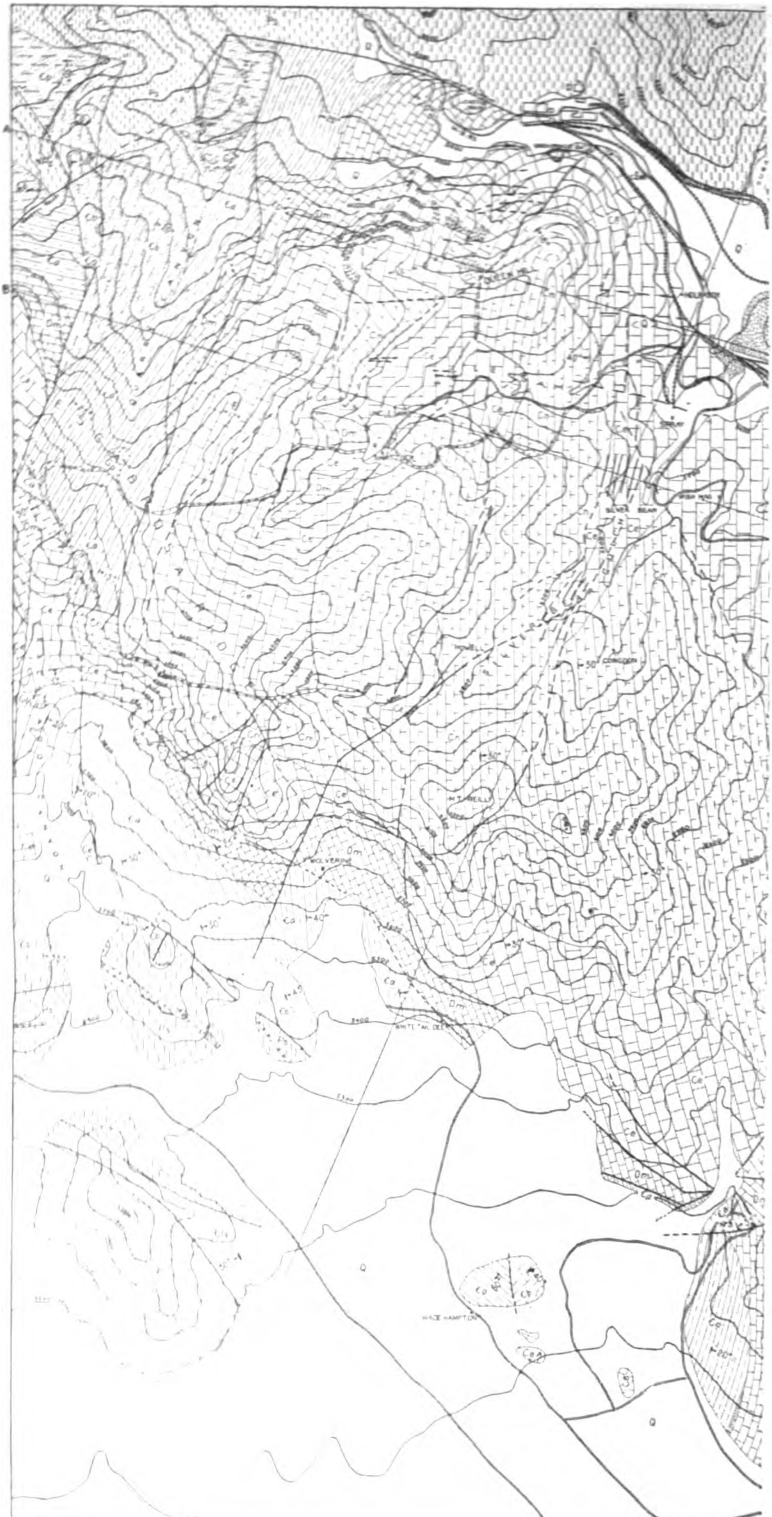
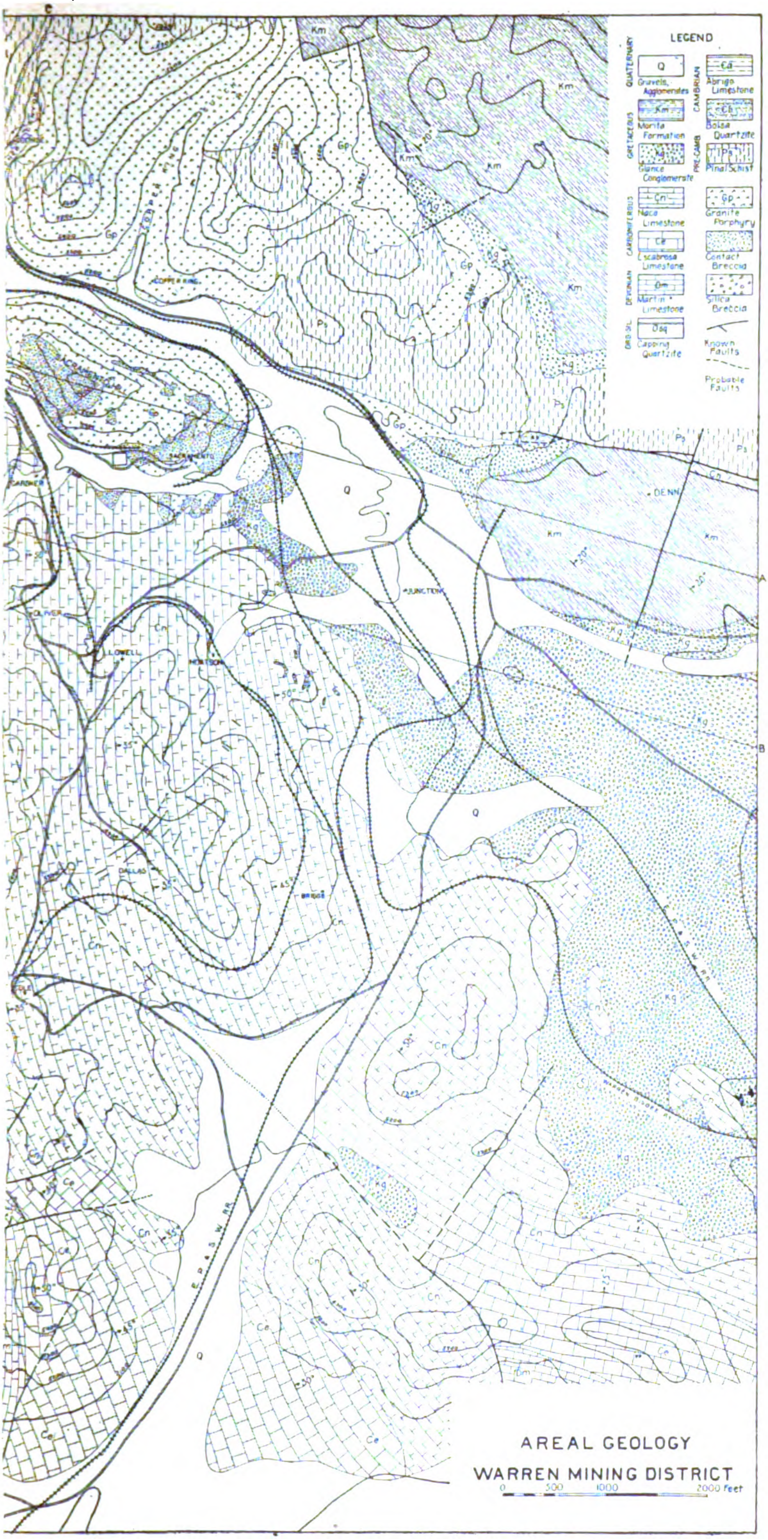


**PLATE 4.—AREAL GEOLOGY MAP WITH SECTION LINES A–A, B–B**




AND C–C RELATING TO THE SECTIONS SHOWN BY PLATES 5 TO 7.


**Plate 5.**

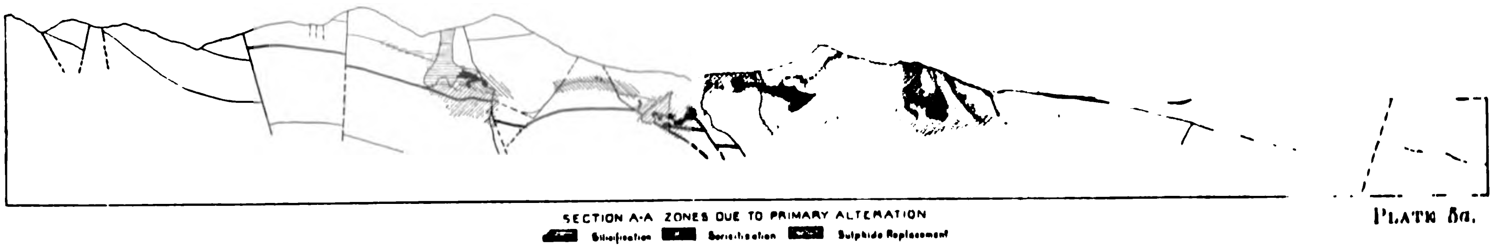



**Plate 5a.**

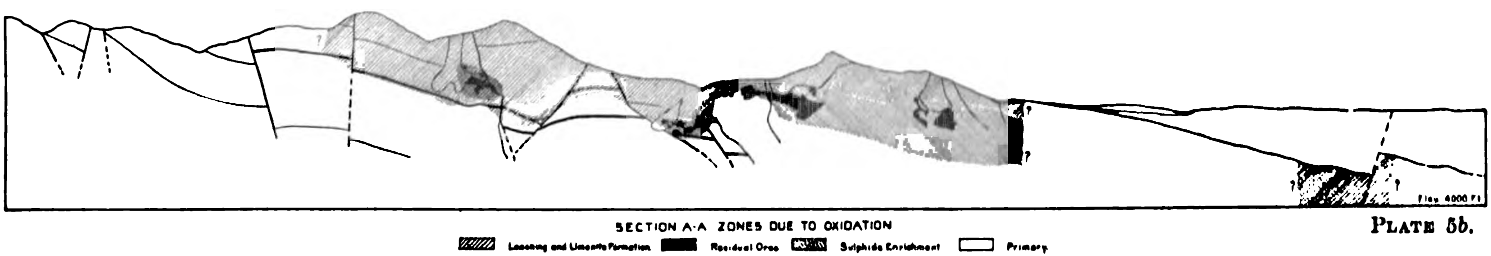



**Plate 5b.**

**Plates 5, 5a, 5b.—Geological Section Along Line A–A of Map, Plate 4.**

nary wash. This block is bounded on the south by the Modern stockwork of granite porphyry, cutting through the schist and Paleozoic beds and striking northeast and southwest.

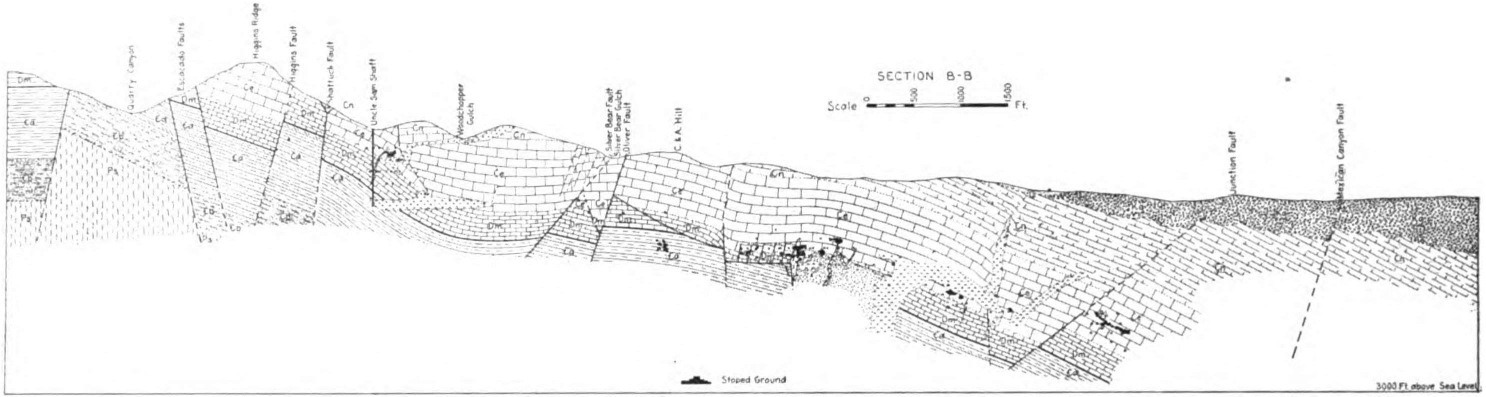
To the south of the Bisbee Extension Block are two blocks, the Escabrosa, extending from the Juniper Flat granite mass to the Escabrosa Ridge line of faults and porphyry intrusions, and the Bisbee West Block extending from Escabrosa Ridge to the southwest. The Escabrosa Block is up-thrown with respect to the Bisbee West Block, and consists almost entirely of schist and porphyry, until Mt. Martin is reached. Here we have the whole Paleozoic section exposed up to the middle of the Escabrosa limestone. The dips at Mt. Martin are southwest. The Bisbee West Block is covered at the northwest by Carboniferous beds dipping to the northwest and west. These are followed to the south by Cambrian beds, dipping to the west and southwest. The Escabrosa Block is bounded on the southeast by the Quarry Fault, beyond which are the Copper Queen and Don Luis Blocks. The Bisbee West Block is bounded on the south by the Abrigo Fault, to the south of which is the Naco Block; relatively very much down-thrown. To the northeast it is bounded by the Bisbee West Fault, to the north of which is the relatively up-thrown Don Luis Block.

The Copper Queen Block is down-thrown with respect to all the surrounding blocks. It is bounded on the north by the east and west Dividend Fault, which has a throw of at least 1,500 ft., throwing the Pinal Schist to the north against Paleozoic beds to the south. It forms a block with the Quarry Fault, a northeast to southwest fault which is the western boundary of the block. The Quarry Fault in turn, forms a block with the White-Tail Deer Fault which is nearly parallel to the Dividend Fault and throws Cambrian beds and Pinal Schist to the south against Devonian and Carboniferous beds to the north. The block is finally covered to the east by Cretaceous beds which are in turn covered by Quaternary wash. To the southeast, the block ends abruptly against the Gold Hill Overthrust Fault. The dips in the Copper Queen Block are generally to the east and northeast.

The Don Luis Block is a small up-thrown block between the Copper Queen and Bisbee West Blocks. The larger part of this block is covered by Quaternary wash, with numerous outcrops of schist and Cambrian beds. The dips in this block are variable but generally to the eastward.

In addition to these are four other blocks, three of which are evidently post-Cretaceous, and one probably so. These are the Tombstone Overthrust Block at the northwestern end of the range thrusting Naco limestone over Morita sandstone; the Naco Block bounded on the north by the Abrigo Fault; the Gold Hill Overthrust Block; and the Glance Overthrust Blocks at the southeastern end of the range.

The Cretaceous Tract, except where the sediments overlie the pre-




Legend As in Plate III

**PLATE 6.**

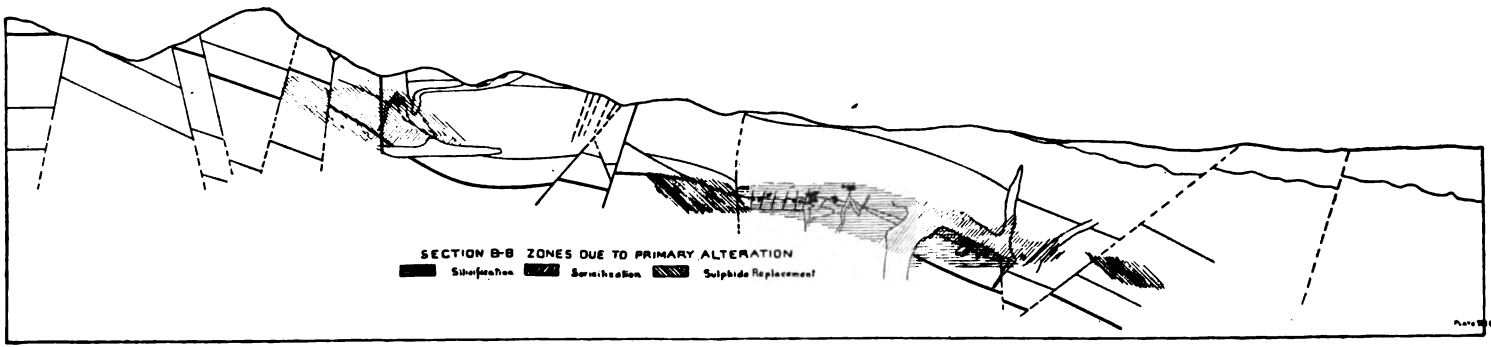
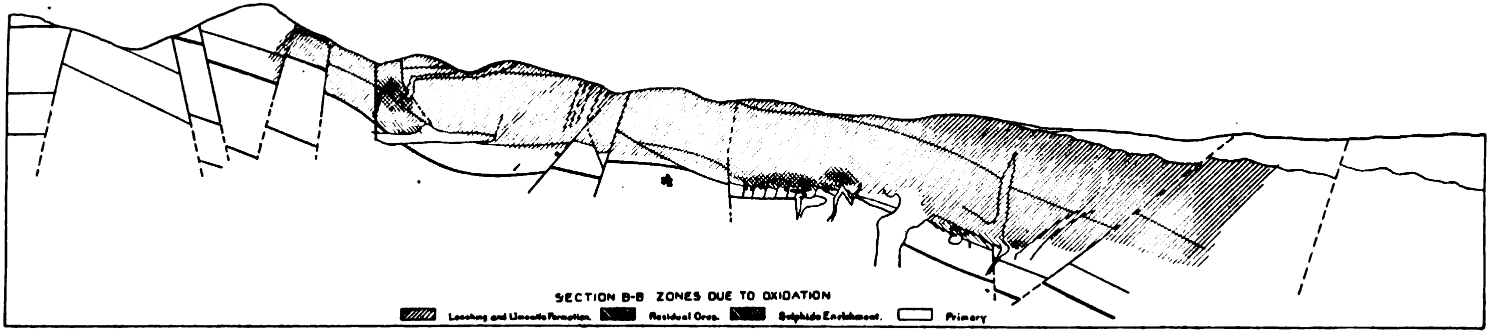



**PLATE 6a.**




PLATES 6, 6a, 6b.—GEOLOGICAL SECTION ALONG LINE B–B OF MAP, PLATE 4.

PLATE 6b.

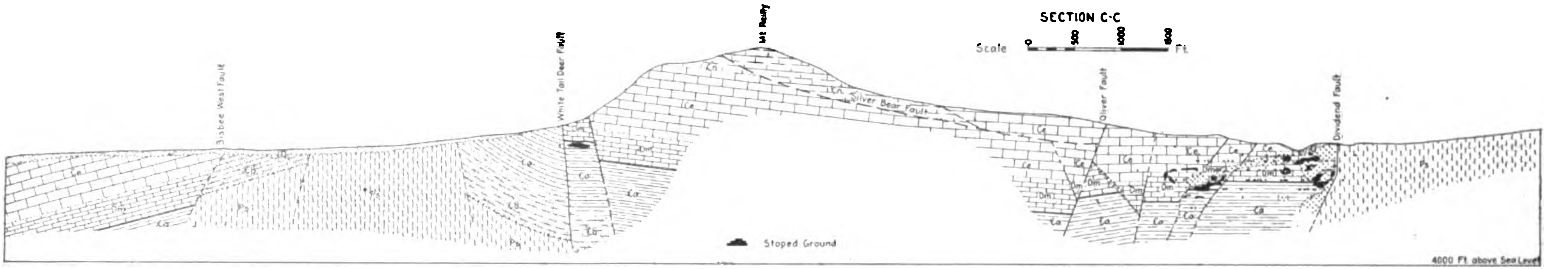



PLATE 7.

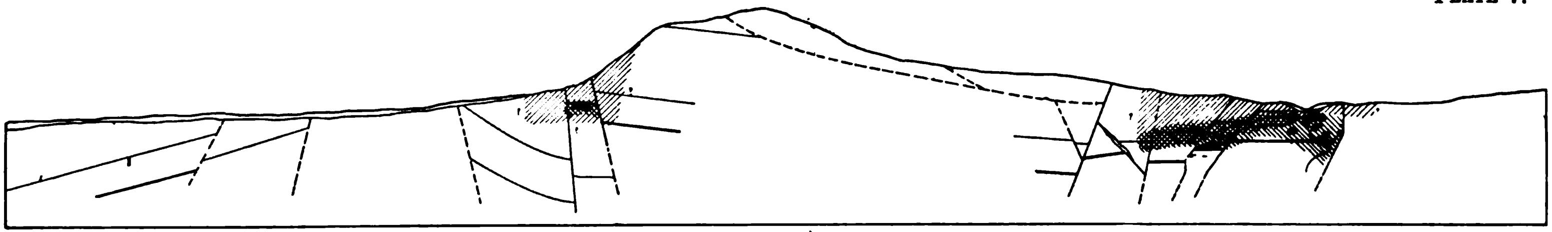



PLATE 7a.

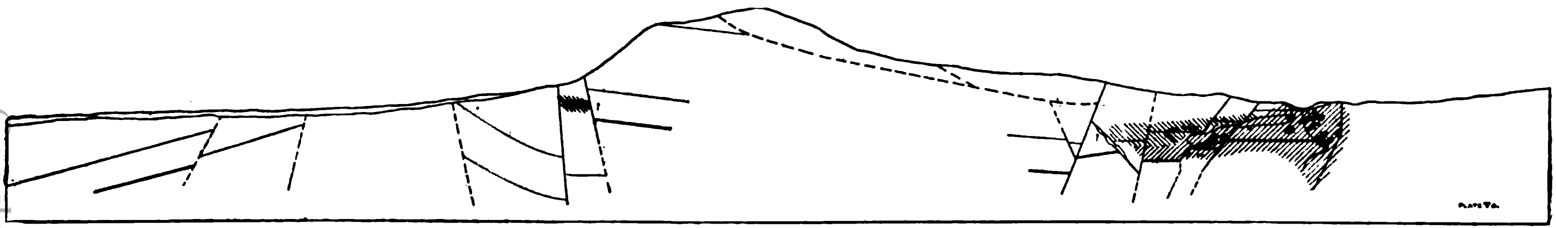



PLATE 7b.

PLATES 7, 7a, 7b.—GEOLOGICAL SECTION ALONG LINE C–C OF MAP, PLATE 4.

Cambrian granite mass, and the Sacramento Hill porphyry, is devoid of igneous rocks. No intrusion has been observed cutting the Cretaceous over this area.

*Igneous Rocks*

The pre-Cambrian Juniper Flat granite mass has the form of a batholith. The contacts with the schist are clean cut. It probably forced its way up very slowly along some ancient line of weakness. There is no evidence of its having engulfed the schist, as no horses of schist are found inside the area exposed. From the nature of the schist, however, it is difficult to tell how it came in.

In all the pre-Cretaceous blocks mentioned above, extensive intrusion has taken place of granite porphyry, followed in some instances by extrusive rhyolites. These intrusions and extrusions came in for the most part along faults already in existence, but spread out in the less resistant sediments as sills and irregular masses. Although coming in at more or less the same time, some dikes clearly cut the others. The general habit of the porphyry was to follow some fault for a certain distance, and then cut across to another fault. The whole process suggests a very rapid intrusion.

In the Bisbee Extension Block, at the northwestern end of Escabrosa Ridge, there is an occurrence of extrusive rhyolite overlying an erosion surface of Naco limestone and an intrusive porphry sill.

Cutting through the Sacramento Hill porphyry, north of Naco Road, are several small dikes of andesite. This same andesite occurs cutting Escabrosa limestones near the Cole shaft, and has been observed underground near the Shattuck, Wolverine and Wade-Hampton mines. At the Shattuck mine, it is undoubtedly later than the porphyry. This andesite in all observed cases has come in as very small dikes in already opened fissures. In only one case has an outcrop been observed in higher beds than the Martin.

In the post-Cretaceous blocks, no intrusions have been seen except in the Naco Block. Here, at the north end of the block, entering along the big Abrigo Fault, an extrusive rhyolite mass has come in. This mass is similar in all respects to the mass mentioned above in the Bisbee Extension Block. Except for this one case, the large faults of post-Cretaceous age have not been accompanied by igneous intrusion.

The most important, economically, of all these igneous rocks is the granite porphyry, as it alone seems to play an important rôle in the processes of ore deposition. By far the greater amount of porphyry in the district cuts through with clean contacts. The one exception to this rule is the mass of Sacramento Hill. The mode of intrusion of this particular mass is somewhat different from all the others, due to the accompaniment of mineralizing emanations. This will be more fully treated later.

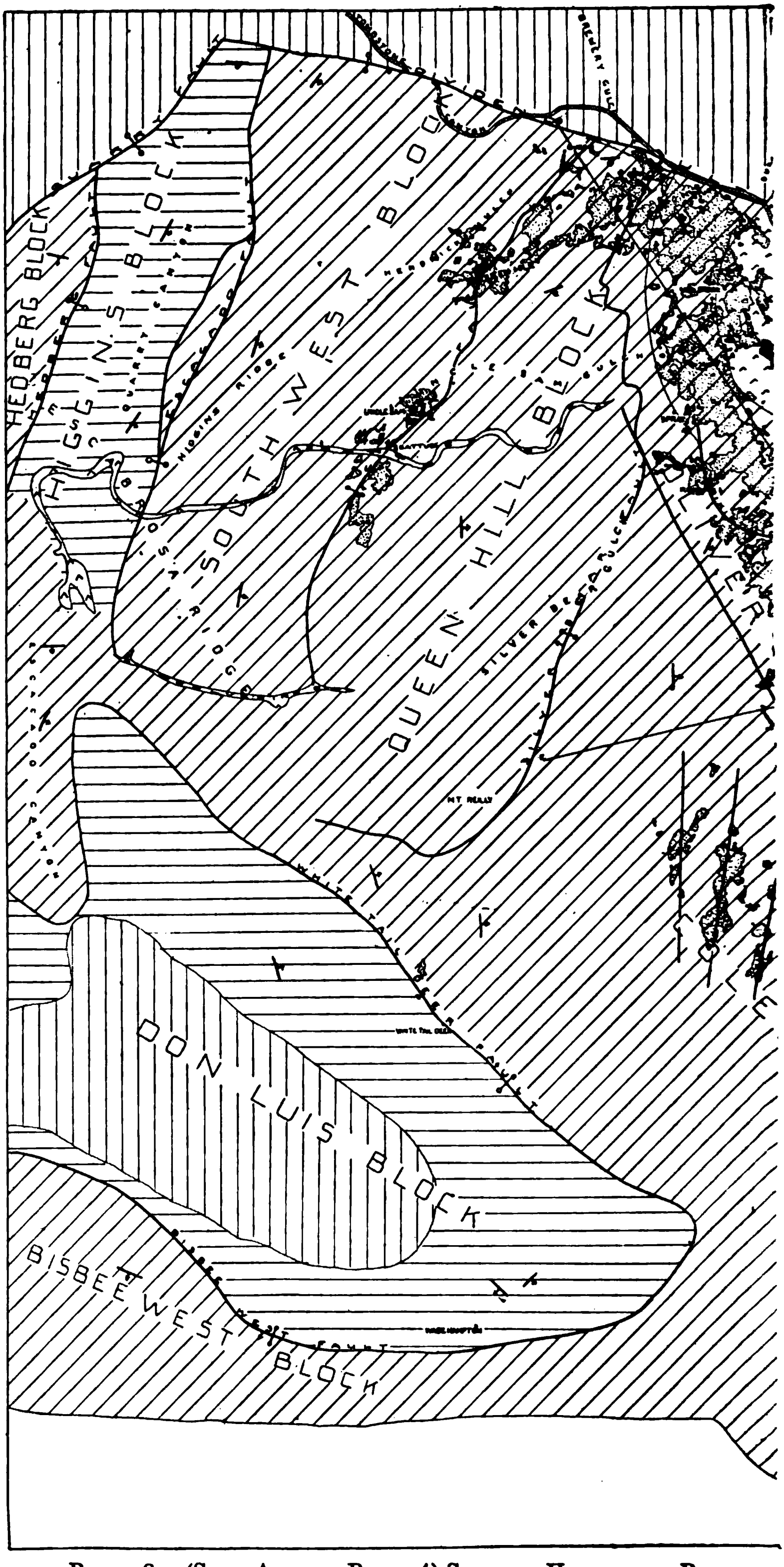
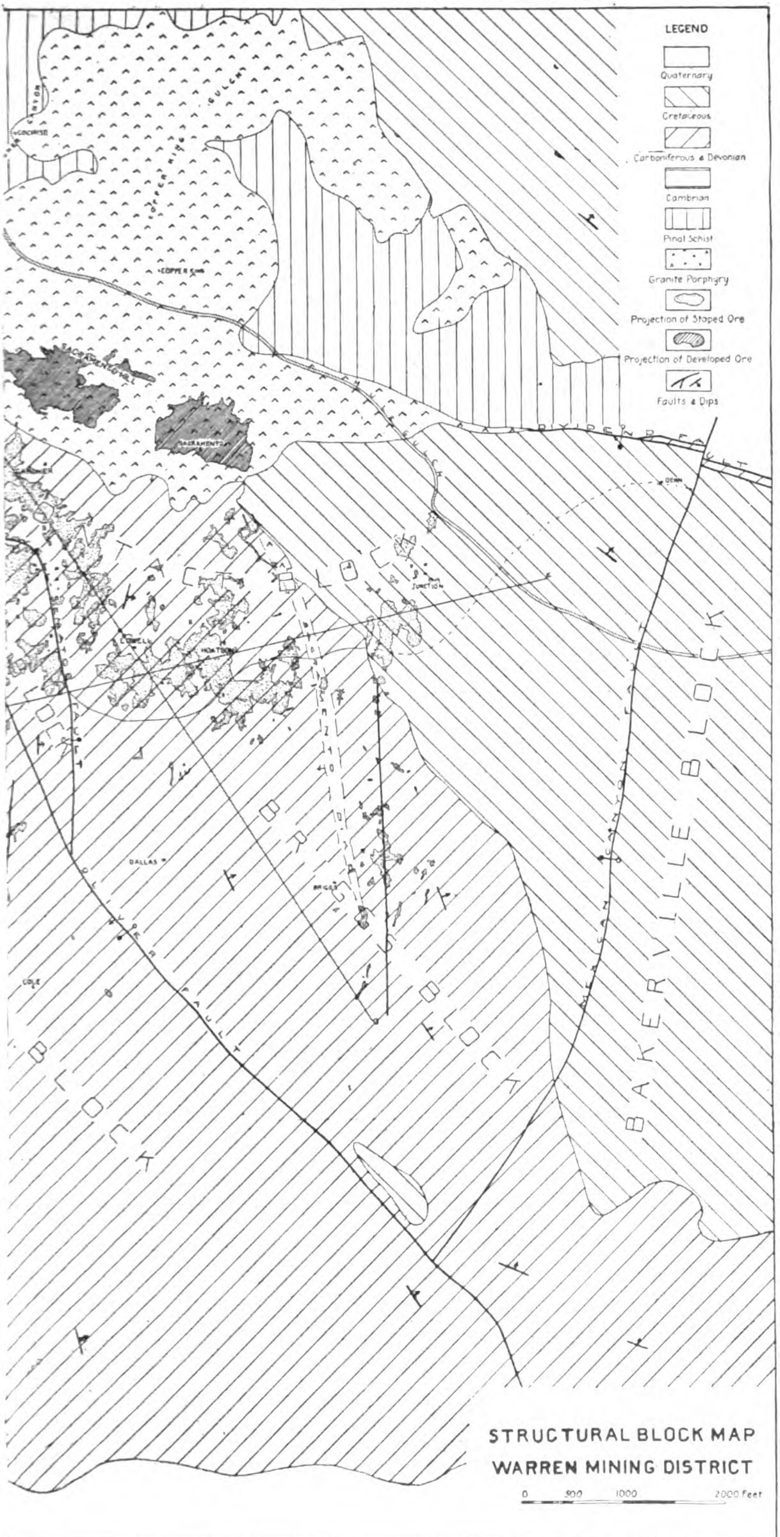



**Plate 8.—(Same Area as Plate 4) Showing Horizontal Projection**




**of Stopes and Section Lines D–D and E–E of Plates 9 and 10.**

## Copper Queen Block

### *Fault Blocks*

From an economic viewpoint, the Copper Queen and Don Luis Blocks are at the present time the only important ones in the range, and these two blocks, together with the mode of intrusion of the Sacramento Hill porphyry mass, will be more fully explained below.

The Copper Queen Block, whose details are best shown by Plate 8, lies in the heart of the range. It is a relatively down-thrown block, being bounded on the north by the Dividend Fault, a N. 75° W. fault dipping to the south from 60° to vertical. This fault has an estimated throw at the town of Bisbee of at least 1,500 ft. throwing Pinal Schist with a few outliers of Bolsa Quartzite on the north against Paleozoic sediments to the south. This fault apparently forms a block to the west with the Quarry Fault, a N.E.-S.W. fault which forms the western boundary of the area. To the south, the block is bounded by the White-Tail Deer Fault system parallel to the Dividend, throwing Bolsa quartzite and schist to the south against Abrigo and Martin limestones, to the north. To the east, the block is covered by Glance conglomerate and Morita sandstone, which are in turn covered by the Quaternary gravels of Sulphur Springs Valley. In the southeast, the block ends abruptly at the Gold Hill Overthrust Fault.

Starting somewhere near the Spray shaft, the block is cut by a N.W.-S.E. fault of considerable throw, the Oliver Fault, which dips about 60° S.W. This fault increases in magnitude toward the southeast and where it is finally covered by Quarternary wash, about halfway between Don Luis and the Country Club, it throws Abrigo limestone to the northeast against Naco limestone to the southwest, a throw of at least 1,500 ft. This fault is encountered in the Spray, Oliver and Cole workings. At the Spray the throw is about 200 ft.; at the Oliver it throws Abrigo limestone against Escabrosa with a throw of at least 500 ft. It finds topographic expression in the big draw between the Dallas and Cole shafts.

The Copper Queen Block, as seen by Plate 8, may be roughly divided into eight structural blocks. Starting from the west, we have first the Hedberg Block, relatively down-thrown, and with dips generally to the south. This is followed by the Higgins Block, up-thrown with dips to the east and southeast. Following this is the Southwest block, bounded on the east by the Czar fault system, and on the west by the Escacado fault system. This block is down-thrown relative to the Higgins Block but up-thrown in relation to the Queen Hill Block to the east. The dips here are nearly due east. The Queen Hill Block is a funnel-shaped block of Naco and Escabrosa limestone. It is well shown in section A-A, Plate 5. It is bounded on the west by the Czar fault system, and on the east by the Silver Bear Fault. The dips on the surface are irregular

but, in general, to the east. Underground, below where the two faults wedge out, the dips are consistently east. Surrounding Sacramento Hill, to the south and southwest is a dropped block, which we may call the Contact Block. It is bounded on the north and northeast by the Dividend Fault and Sacramento Hill, and on the south and southwest by the Dixie-Howell-Senator Faults. To the southwest of this Contact Block and up to the Oliver Fault is the Oliver Block of relatively upthrown beds. Southwest of the Oliver Fault is the Cole Block and northeast of the same fault is the Briggs Block which extends east to the post-Cretaceous Mexican Canyon Fault. The dips of all these blocks more or less surrounding Sacramento Hill are nearly due east. The Oliver Fault has tended, however, to swing the up-thrown side, especially where the throw becomes greatest, around to the northeast as is shown in the detailed geological map of the district, Plate 4.

To the east of all these blocks is an eighth block, the Bakerville Block, almost entirely covered by Glance Conglomerate. This block is bounded on the west by the Mexican Canyon Fault, striking about N. 15° to 30° east and dipping to the northwest. This fault is a late one and extends beyond the Dividend Fault, causing the prominent dislocations of the Mural Limestone seen north of Lowell and Warren.

As is shown in section B-B, Plate 6, the apparent dip of the beds in the whole block is very much less than the actual. This is due to minor faulting or slipping which has had the effect of dropping the beds very consistently to the west and raising them to the east. This is one of the typical structural features of the camp and seems to be caused by the extreme resistance of the beds to folding.

There are two ages of faulting, one pre-Cretaceous and the other post-Cretaceous. As has previously been shown, the porphyry in the district is pre-Cretaceous. The age of the faulting has therefore been determined in large part relative to the age of intrusion. The only large fault which is definitely post-Cretaceous is the Mexican Canyon Fault. As the fault apparently blocks with the Oliver Fault as shown in Plate 8, the Oliver is probably also post-Cretaceous. This is borne out by the character of the exposures underground. There, practically no mineralization accompanies it, and in Oliver ground it apparently cuts off an orebody, which as will be seen later, would make it post-Cretaceous. Both of these faults have the same effect, that is to raise the sediments to the east, and to drop them to the west. The probability is that the Junction Fault, shown in section B-B, Plate 6, is also post-Cretaceous, as it is reported to cut off certain parts of the Junction orebodies, and it is also possible that a large part of the north to south slipping in Lowell and Gardner ground is also post-Cretaceous. Here the throw of each individual fault is so slight, and the ground so highly metamorphosed, that it is not susceptible of proof one way or another.

All other faults, so far as observed, have had their major movements in pre-Cretaceous time and before the introduction of the porphyry. Some faults have had movements in both ages. The Dividend shows some movement at the present time, and enough movement occurred in post-Cretaceous time to throw out the Glance Conglomerate. Some post-intrusion slipping has also undoubtedly taken place along the Czar Fault.

### *Intrusions*

As has previously been shown, the block has been extensively intruded by granite porphyry. The largest and most important mass is that of Sacramento Hill, the other occurrences being more or less radial offshoots of dikes and sills from this central mass. It has also been shown that underground exposures below the Escabrosa horizon are much greater than surface exposures.

The path of the Sacramento Hill porphyry was the previously opened-up Dividend Fault. Underground development has shown that the porphyry acted as a plug, and that none of the later movements on the fault have taken place along the original line through the mass. The offshoots of the mass have generally followed major lines of weakness such as the Czar, and Silver Bear Faults. Some, however, such as the Shattuck dike, have cut across the country irrespective of fault lines.

The Sacramento Hill, and the connecting Lowell, mass differ from all the other porphyry occurrences in the range, in that they were accompanied by mineralizing emanations, which have completely altered them. These emanations, besides altering the porphyry itself, changed and broke up the surrounding sediments. In consequence of this, as the mass worked its way up, it dragged in pieces of schist, quartzite and limestone along its edges. The final result was a central core of metamorphosed porphyry, with a periphery of contact breccia, made up of highly silicified and altered fragments, sometimes rounded and sometimes angular, of schist, porphyry, quartzite and metamorphic limestones. This contact-breccia zone has a variable width, with a maximum of about 1,000 ft. It has fairly sharp contacts with the surrounding limestone, but tends to grade off into brecciated porphyry, and finally blocky unbroken porphyry, toward the center of the intrusive mass.

## Don Luis Block

The Don Luis Block is an elliptical-shaped block up-thrown relative to the Copper Queen. It is bounded on the north and east by the White-Tail Deer fault system, throwing Abrigo and Martin limestone against schist, Bolsa, quartzite, and some Abrigo limestone. To the south it is bounded by the Bisbee West Fault, which is covered by the Quaternary

wash toward the east and toward the west it is bounded by the Quarry Fault. The dips are variable, but for the most part are to the northeast and east.

### *Intrusions and Orebodies*

The block is intruded both by granite porphyry and by andesite, the latter, however, not outcropping. There are two ore occurrences in the block, one at the northern boundary, and rightfully in the Copper Queen Block, and one at the southeastern end. The first of these, the White-Tail Deer orebody, occurs as a replacement of Abrigo limestone, about 50 ft. from the base of the Martin. It is not associated with any known intrusion, but is close to the White-Tail Deer fault system. The second occurrence is that of the Wade-Hampton. Here the ore occurs as lead and copper ore in a fault separating Abrigo limestone from Bolsa quartzite. The ore here is apparently associated with a dike of andesite and occurs in the crushed Abrigo limestone on the fault.

## VI. GEOLOGIC HISTORY

The earliest geologic age is recorded in the district by the pre-Cambrian Pinal Schist. As has been previously stated, this was probably derived from arenaceous sediments.

Due to the extreme metamorphism, the geologic history subsequent to the deposition of these early sediments, and up to the deposition of the Cambrian sediments, is obscure. The sediments were probably subjected to extreme folding, intruded by diabase, and deeply buried. After the metamorphism was complete, the schist was intruded by a large mass of granite. From then on to Cambrian time, erosion worked on the whole complex, wearing it down to an even peneplain.

At the beginning of Cambrian time, a gentle subsidence took place, resulting in shore-line deposits of quartzites, the subsidence becoming more rapid toward the end of the period, in which time were deposited finer sands and some shales. The land surface toward the end of the Cambrian period became more distant, as evidenced by the deposition of the Abrigo shales, and shaly limestones.

At the end of Cambrian time, a gentle rise took place over an extensive area, bringing the flat sea bottom close to the surface, with land actually emerging probably toward the south. During the Ordovician and Silurian ages, this lowland surface of Cambrian beds was subjected to slow erosion simultaneously with the deposition in the district of an 8-ft. bed of quartzite. A slow subsidence then took place to the north of the district, allowing the thicker deposits of quartzite in the Yellowstone Range of 40 ft., and at Globe of 500 to 700 ft.

At the end of the Silurian, a sudden subsidence took place, resulting

in deeper-sea conditions and thus the Devonian age is represented by fairly pure dolomitic limestones.

The sea evidently continued to deepen all through Mississippian time with the deposition of the pure Escabrosa limestone. During Pennsylvanian time, subsidence did not keep pace with deposition and the sediments became increasingly more shaly, and cherty showing a nearer approach of land surface and shallower seas. At the end of the Pennsylvanian, and probably during the Permian age, a violent uplift took place, resulting in the doming of the sediments around the pre-Cambrian granite mass. The sediments were shattered by extensive faults, the major lines of weakness being northwest and southeast. The granite mass itself resisted this shattering to a great degree except around its edges. Immediately following the faulting, or partly accompanying it, granite porphyry was intruded, following generally the major lines of weakness, but also shooting sills and dikes into the less resistant sediments. Accompanying one of these porphyry intrusions, that of Sacramento Hill, were mineralizing emanations, which metamorphosed the surrounding sediments and the porphyry itself, and were the source of our present orebodies.

During Jurassic and Triassic time, the whole area was subjected to erosion, during which time most of the upper Carboniferous beds were stripped off, and at the crest of the dome, around the granite and schist of Juniper Flat, the old pre-Cambrian complex was laid bare. During this erosional period, the orebodies were subjected to oxidation, which would place the age of primary ore formation definitely at or near the age of the uplift.

At the end of this period, a remarkably sudden subsidence took place, before any of the land except the top of the dome had been leveled off, and the rough topography surrounding was filled up with the extremely coarse Glance Conglomerate. The subsidence was so sudden that the fragments making up this conglomerate were hardly worked over at all, it being made up of angular boulders, some as large as a yard in diameter, of schist, quartzite and limestone. The roughness of this old topography is well shown in the basin covered by conglomerate, in which the town of Warren is situated.

After the whole country was leveled off by the Glance Conglomerate, the subsidence became less rapid, and shore-line deposits of sandstones and sandy shales were laid down, followed by deeper-sea conditions, during which the pure Mural Limestone was deposited, showing abundant Cretaceous (Comanche) fossils. Shore-line deposits followed this again to the end of Cretaceous time.

During Tertiary time, a second uplift took place, along almost the same axis as the previous one. This also took the form of a dome around the old granite mass, and the Escabrosa Ridge porphyry intrusion. This uplift, however, was not as violent. It was accompanied by extensive

block faulting, with little disturbances of the beds within the blocks. The major faults took the form of overthrusts, the largest being the Tombstone and Gold Hill faults. Either accompanying the uplift or subsequent to it, some intrusion of monzonite porphyry took place, at the southern end of the range. Probably some of the extrusive rhyolite also belongs to this age of intrusion. Finally the whole range was subjected to a tilt to the northeast of about 15°.

From then on to the present time, erosion has been steadily at work, and has again laid bare the old pre-Cretaceous sediments along the crest of the dome, with Cretaceous beds exposed to the northeast. To the southwest, the Cretaceous has been all eroded, due to the final tilting. The only evidence left of the second doming is found at the southern end of the range, as shown by Plate 1. During this last erosion period the orebodies have been further subjected to oxidizing and enriching processes, so that we owe their condition as we find them today, to two widely separated periods, in each of which the original primary ores were worked over by surface waters.

## VII. METAMORPHISM AND MINERALIZATION

### General Metamorphism

In a region that has been subjected to as many geological changes as the Warren district, it was to be expected that the rocks would show many variations in texture and mineralogy from the unaltered types. The effects of regional metamorphism as well as dynamic are represented in the Pinal Schist as has been shown in describing the lithological characters of this formation.

The Cambrian formations have also been changed throughout their occurrence in the district by a process related to regional metamorphism in the wide scope of its action. The induration of the pure Bolsa quartzite has gone on until it is of glassy texture, but in the clayey, magnesian Abrigo limestone, besides the segregation of chert, the impurities have everywhere combined into such minerals as epidote, zoisite, chlorite, serpentine, and possibly albite. No such action is at all observed in the conformably overlying Martin limestone, which strongly suggests that the changes took place in the long period separating the middle Cambrian from the Devonian. But since field evidence excludes any great disturbance during the unrepresented period, it is probable that the general metamorphism in the Abrigo was caused by the deep burial under more than 4,000 ft. of overlying limestones after the impurities in the beds had had a chance to segregate and rearrange themselves due to the slow action of cold surface solutions during Ordovician-Silurian times. The necessary temperature conditions were probably obtained by the rise in iso-

therms due to deep burial and to the post-Carboniferous intrusive period when a general rise in temperature could effect changes in the more susceptible impure sediments.

Whether this tentative explanation is correct or not, the importance of recognizing the general distribution in the Abrigo of minerals which may be mistaken for indications of contact metamorphism cannot be overemphasized. This formation is of considerable economic importance and the underground search for ores cannot be aided by the same indications as in the higher limestones.

## Contact and Hydrothermal Metamorphism

That the granite-porphyry intrusion was the supreme factor in the ore deposition of the district cannot be doubted in view of the intimate association of the ores with the porphyry and the intense changes that have taken place in the rocks around the main intrusions.

It has been customary to separate the metamorphism of the rocks around a contact into contact metamorphism, caused by the igneous rock itself, and hydrothermal metamorphism, caused by the heated solutions emanating from the cooling magma. This distinction will not be made in the present article, because there has been nothing found by the authors to indicate that there is any genetic difference in the causes for changes in the rocks, outside of differences in intensity or quantity. The differences in *effects* are not to be confused with the causes, as the first are subject to the many variations of the rocks encountered, their physical and chemical differences, and their previous alteration. The metamorphosing agencies vary also with the distance from their source in the effect they may have on the same rock, but all changes observed so far have usually been graded and not susceptible to sharp definition.

In general, it may be stated that the two main centers of porphyritic intrusion are also the centers of metamorphism. Around Sacramento Hill the effects of metamorphism can be seen on the surface, but around the Lowell center the alteration did not extend through the covering of upper Carboniferous limestone. The effects around Sacramento Hill are more pronounced on the side of the calcareous sediments, which have been so much more altered than the schist of the north side. The porphyry on the south, as well as the covering remnants of contact breccia, are highly silicified and stand out as the crest of Sacramento Hill. The topographical depression around this hill is due to less silicified porphyry breaking through the contact breccia where this last turns from a more or less vertical to a horizontal body. Outside of this moat-like depression is a ring formed by the highly silicified contact breccia which grades into silicified metamorphic limestone, and this into limestone with a decreasing amount of metamorphic minerals until a belt of marbleized rock is

reached, from 200 to 1,000 ft. away from the porphyry. The zone of recrystallization also fades outward irregularly to the unaltered sediments.

On the north, or schist side of the contact, there is some silicification of the older rock, accompanied by considerable pyritization, but this very soon fades out into the unaltered schist.

Underground, in tracing out the zones of rock alteration due to the intrusion and its accompanying solutions, it has been found that they roughly correspond to the ones just mentioned as appearing on surface, with the following general modifications: The zones extend farther than the general surface contours along extensions of the porphyry mass, especially the zone in Lowell ground, and along lines of fracturing which may or may not also be lines of faulting. The extent of the zones is remarkably influenced by the formation they are traced in, the changes going farthest at the top of the Devonian and the top of the Cambrian. Local variations are found to break the arrangement of the zones of alteration where there are minor centers of strong action, such as close to the Shattuck dike in the Uncle Sam and Shattuck ground, in the Southwest mine, and in several other places.

*The Effect of Contact and Hydrothermal Metamorphism on the Formations*

Before taking up the effects of contact and hydrothermal alteration on the rocks of the district it is well to state that hydrous metamorphism or oxidation has obscured or complicated in a great measure the results of contact metamorphism, and in many cases made the exact separation of the process involved too difficult and useless to be undertaken in the course of economic work.

The changes due to circulation of meteoric waters will be taken up later. The mine workings have now gone down far enough in all zones to disclose the fact that there is a depth below which the rocks as well as the ores are different from those closer to the surface. Here there are none of the minerals evidently due to the process of oxidation, and mineralogical associations continue the same indefinitely downward, without the vertical variations of the upper portions. This is the criterion that has been used in separating the effects of primary and secondary processes in the rocks and in the ores. Though not found in this order in the mines, primary effects will be considered first.

As a whole the impression given by the rock alterations in the Warren district is one of abundance and persistence rather than great intensity. There are no high temperature minerals developed in any great amount, or if they were developed at some stage, they have now been replaced. Garnet, diopside, wollastonite, scapolite, and vesuvianite have been observed, but in very small amount, and never forming an important part of a formation. Tremolite, actinolite, and edenite are far more common.

The distinguishing minerals of alteration zones are quartz, sericite, chlorites, especially penninite, and the oxides of iron, magnetite and specularite, as well as the metallic sulphides of iron, copper, zinc and lead.

### *Metamorphism in Contact Breccia*

Around any of the granite porphyry that has been accompanied by mineralizing emanations, such as that of Sacramento Hill, the contact breccia is marked generally by an extreme amount of silica which replaces all other gangue and rock-forming minerals. Pyrite and sericite are next in abundance, with chloritic minerals in variable amounts and calcite practically unknown. Schist and quartzite fragments have naturally remained the least altered in this mass and are consequently recognized most easily. In this contact breccia, or at the edges, there are bodies of intergrown magnetite, hematite and pyrite, with associations of the best-formed garnet, wollastonite, and other contact minerals. Silicification usually decreases toward the outer edge of the breccia, sericite and chlorites increasing. In many thin sections studied, the prevailing impression gathered is that in most of the breccia silicification is the last process involved, following sericitization and chloritization.

There are, however, distinctly later porphyry dikes cutting the breccia, as has already been stated, which have also been accompanied by metamorphosing emanations, the result being a complication and confusion of the processes involved. Certain highly chloritized portions of the breccia, where penninite replaces quartz in a noticeable way, are the result of these conditions, which are economically important, since copper minerals are concentrated in very pure form by the superimposed alteration. There are also other observed forms of this repeated alteration depending on the variations of either the first or the last. But it is generally true that the second process never reaches the point of adding silica or even adding sericite to the previously changed rocks.

### *Metamorphism in the Contact Limestones*

The transition from breccia to limestone that can be recognized is apt to be very sudden in the Carboniferous horizons, and graded in those below. This is because the alteration of the lower limestones is more pronounced. The impure, and at the same time easily crushed, Devonian and Cambrian formations become a mass of sericite, penninite, calcite, and quartz, with some tremolite, garnet, diopside, wollastonite, and epidote, while the Escabrosa limestone usually has just a narrow fringe or a few bands of metamorphic minerals and the mass of the rock is simply recrystallized to marble. As extremes of alteration in this contact zone there are places where the limestone, especially the Devonian, is converted into a highly sericitic or chloritic mass, or into almost pure quartz.

There is some evidence in this zone also that silicification is the last process affecting the rocks under ordinary conditions.

Farther from the contacts or the centers of mineralization the changes visible in the limestone are a decrease in silica and sericite in all the rock, while these minerals may persist in breaks and joints. Recrystallization of calcite, but not of the impurities, is very common.

In this contact zone the accumulation of sulphides is marked by the increase in either sericite, silica, or chloritic minerals around the sulphides.

### *Metamorphism Outside of Contact Zones*

Sometimes rocks with considerable alteration are encountered far from any known porphyry, in which case it may be that solutions have traveled along fractures, or that they may have come with undiscovered porphyry. Usually, however, the copper ores have been found replacing the limestone very much farther away than any general alteration of the formations, in which case the transition from fairly unaltered rock to sulphides is sudden, but bears some relation to the local structure.

### *Metamorphism of the Porphyry*

The primary solutions emanating from the granite porphyry affect that porphyry in about the same way as they do the adjacent rocks. If the nearby rocks, whatever they may be, are silicified, the porphyry will be also. If sericitized or chloritized, the same processes will have affected the igneous rock. The main difference comes in the relative amount of sulphides which have formed during these changes. Of course, the same minerals will not form in limestone as in igneous rock, but an increase or decrease in silica or magnesia or iron will be parallel in both cases.

The main mass of Sacramento Hill is so highly altered that a thin section of this rock shows only a mat of sericite plates, with variable amounts of quartz, pyrite, and chlorite, without much being left to distinguish between quartz, feldspar and mica phenocrysts, and the base. Close to the breccia, silicification is common, accompanied by disseminated copper deposits.

In the dikes radiating from the hill, the changes in alteration of the porphyry are very clearly marked, as well as the relative ages of succession of the processes. Receding from the silicified portions, sericite increases and the outlines of phenocrysts, first of quartz, then of feldspar, begin to appear. Lastly appear the outlines of biotite, marked generally by successive laminæ of sericite and pyrite. Farther away penninite is found in increasing amount, the quartz phenocrysts being unattacked and some of the plagioclase feldspars scarcely altered. The chloritic minerals appear as remnants replaced by sericite.

When sericite has almost all disappeared, and there is no silicification, epidote, zoisite and calcite are sometimes found as alteration or meta-

morphic minerals in the porphyry. At this zone there are also found portions of the rock converted into almost pure penninite and edenite, with some serpentine, as if the ferromagnesian minerals lacking in the more sericitic rock had been driven to accumulate and replace other portions. Only the presence of some unaltered quartz phenocrysts and very resistant apatite crystals helps to determine the original rock, although transitions have been exposed in some of the workings.

Sulphides in some amount invariably accompany the contacts of porphyry that is altered to this degree.

Farther away from the centers of action slight chloritization of the base, of some feldspars and of micas, and eventually of the micas alone, is the only alteration noticeable. This fades out in a few observed cases to fresh rock, or rock affected only by meteoric waters. The contacts of this less altered porphyry have either just a little pyrite and sphalerite, or no sulphides at all, and even have had no appreciable effect on the immediately adjacent limestone when seen in thin section.

The alteration of the more basic dikes which cut the contact breccia is also similar to the one induced in the breccia, except that the earlier changes are not observed in these dikes.

## Summary

To summarize, it may be said that the weakest as well as the farthest reaching alteration in the porphyry is chloritization. Next comes epidotization, which is of minor importance. Sericite and finally quartz are the additional minerals formed with the increase in the strength as well, probably, as the temperature of the emanations. The minerals formed by the stronger solutions are formed by replacement of the earlier ones, in the ordinary case when metamorphism is advancing in intensity from the center. Superimposed alteration may be caused by later intrusives, or in certain instances by the decrease in intensity in the dying-out period of alteration.

Some unaltered dikes in Holbrook and Czar ground are found in zones where the rest of the rocks show intense sericitization, including some masses of porphyry. These dikes are associated also with some sulphide ores, the contacts being far more distinct than ordinarily. If it were not for the slight sericitization of these dikes close to the ore, while away from it they show no hydrothermal action, it would be possible to believe that they are the result of post-mineralization intrusions.

The Shattuck dike shows similar phenomena, as it is very fresh for long stretches, and then, close to the ore zones in Uncle Sam and Shattuck ground is almost as altered as any rock around Sacramento Hill, showing that the altering solutions did not accompany all the porphyry but found their way through special channels.

The following partial analyses of altered porphyries are typical:

| | I | II | III |
|---|---|---|---|
| Au, ounces | *Nil* | *Nil* | *Nil* |
| Ag, ounces | 0.04 | 0.08 | 0.04 |
| | Per Cent. | Per Cent. | Per Cent. |
| Cu | 0.24 | 0.11 | 0.11 |
| Pb | Tr. | Tr. | Tr. |
| $SiO_2$ | 55.9 | 76.6 | 73.3 |
| Fe | 9.8 | 5.6 | 5.7 |
| CaO | 1.8 | 3.1 | 0.5 |
| MgO | 0.5 | 3.2 | 2.8 |
| $Al_2O_3$ | 16.6 | 7.4 | 10.1 |
| S | 8.8 | 1.0 | 0.4 |
| Mn | 0.6 | 0.6 | 0.6 |
| Zn | Tr. | Tr. | Tr. |
| P | 0.08 | 0.10 | 0.09 |
| Totals | 94.32 | 97.71 | 93.60 |

I. Highly sericitized porphyry with pyrite, from Sacramento dike, close to Sacramento Hill.

II. Chloritized, partly sericitized and epidotized porphyry, from Sacramento dike; about 1,000 ft. away from the hill.

III. Slightly chloritized porphyry, Dallas mine.

The general zones of primary alteration are shown for the sections through the camp in Plates 5*A*, 6*A*, and 7*A*. Silicification and sericitization have been explained already. Chloritization is not marked because it is too irregular, and though important in the porphyry, the sediments in that zone are not generally affected. The zone of sulphide replacement was introduced, where the limestones surrounding sulphide replacements were affected so slightly immediately away from the orebodies that there can be said to be practically no metamorphism.

## VIII. THE ORES OF THE DISTRICT

### General Occurrences

The general occurrence of the orebodies can be best appreciated by reference to the horizontal and vertical projections of the stopes given in Plates 8 to 10.

In the horizontal projection (Plate 8), there is a well-marked crescent-shaped zone around Sacramento Hill, with an extension around the Lowell mass of granite porphyry. Along the Dividend and Czar zones the effects of major structural lines are well marked by the abundance of ore taken out. The Shattuck dike-Czar zone intersection has its group of orebodies, and extensions and outliers from the general zone follow both the Sacramento dike and the series of breaks in Briggs and Cole ground to the southward.

In the vertical projections (Plates 9 and 10) the inclination of the ore zone is shown; also in Plates 5 and 6 the stratigraphical position is seen to follow in general the dip of the limestones, at horizons varying from the top 300 ft. of the Cambrian to the lower 300 ft. of the Carboniferous.

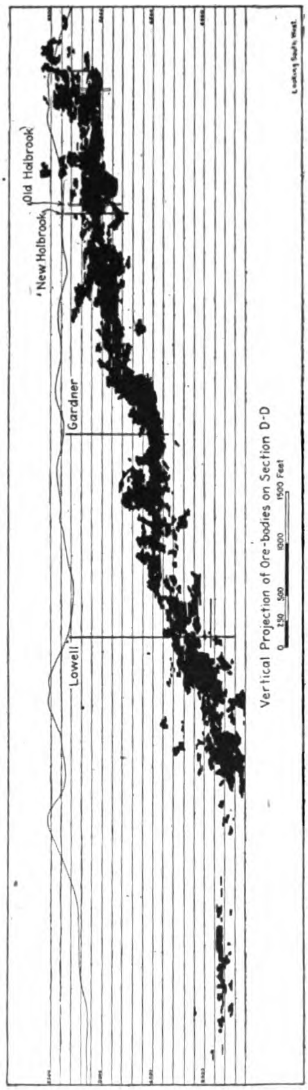



Plate 9, Section D-D.—Showing Inclination of Ore Zone Following Dip of Limestones.

## Classification of the Orebodies

As in the case of the metamorphism and alteration of the rocks, the authors have found no reason to separate the orebodies due to genetic differences. All the ores are believed to be due to one general period of

primary mineralization which followed closely the intrusion of the main porphyry masses. Differences in the ores are ascribed mainly to the character of the formation that they replace and to the amount of general

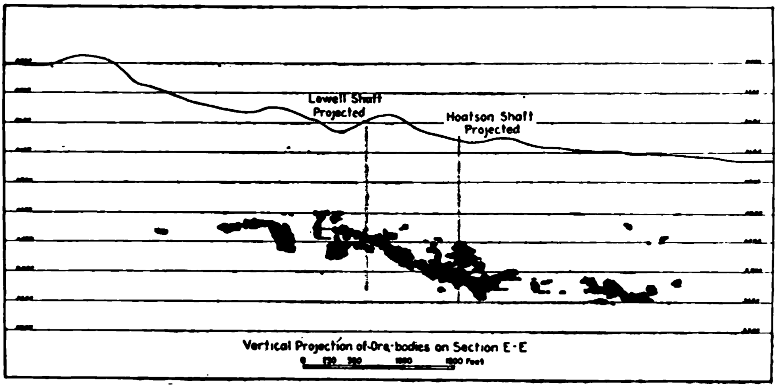



PLATE 10, SECTION E–E.—ORE ZONE BETWEEN LOWELL AND HOATSON SHAFTS FOLLOWING DIP OF LIMESTONE.

metamorphism in the surrounding rocks due to the distance from the centers of alteration. All the ores are also believed to be formed by metasomatic replacements of various rocks.

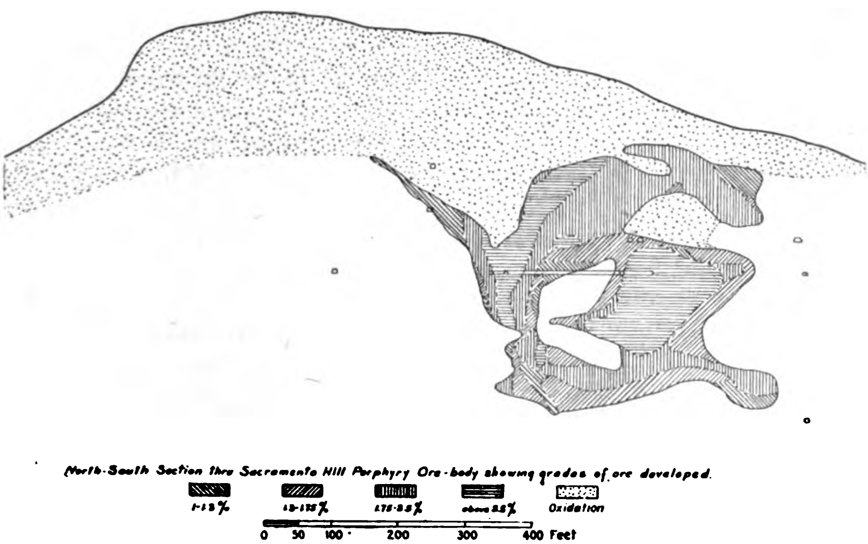



PLATE 11.—SACRAMENTO HILL OREBODY.

Four convenient subdivisions can be made, dependent on the formation replaced, as follows: ores in porphyry, in contact breccia, in contact metamorphic limestone, and in relatively unaltered limestone. In con-

sidering these ores, the primary mineralization will be taken up first and the effects of meteoric water circulation left for later discussion.

### *Ores in Porphyry*

To this type belongs the ore developed recently in Sacramento Hill. The ore occurs as a very irregular mass of chalcocite, pyrite and some bornite in brecciated, altered porphyry at the inner side of the contact breccia. The ore also penetrates the contact breccia, as shown in Plates 5 and 11, and connects with some typical bodies in this latter rock. The porphyry, in general, is sericitized, and somewhat silicified close to the

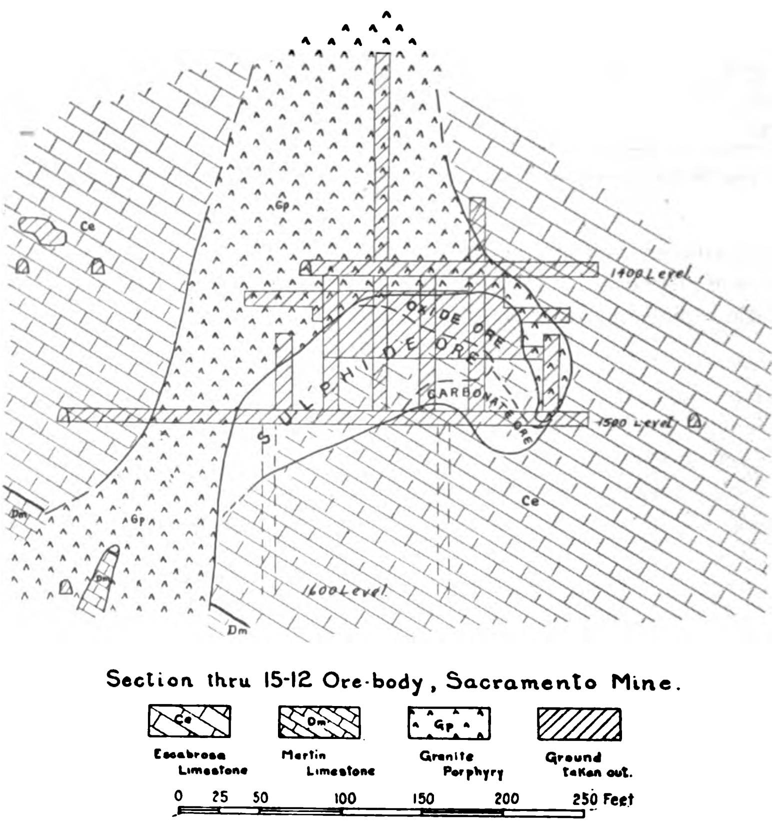



PLATE 12.—SHOWING CONTACT OREBODY ALONG SACRAMENTO DIKE AND SPECIAL FEATURES OF OXIDATION.

breccia, and in parts of the ore. The primary sulphide minerals are pyrite and bornite alone, this being the only type of ore in the camp in which chalcopyrite is not an important constituent. Partial analyses of this porphyry ore are given in Table 2.

Besides the ore in Sacramento Hill, ore occurs in porphyry in a

great many places in the contact-metamorphic zone, where the sulphides replace limestone and in a minor, but very distinct way, the adjacent porphyry. A noteworthy example of this is the orebody known as the Dividend Slice No. 1, where most of the porphyry is entirely converted into ore.

### *Ores in Contact Breccia*

This second type of ore is found from the Holbrook to the Lowell, Oliver and Sacramento mines. The bodies are extremely irregular in shape and size, and are not confined to any definite horizon, but are usually found in unenriched and unoxidized state. One of these bodies outcropped to the west of Sacramento Hill.

The breccia, as has been explained, is highly silicified, and the ores themselves may be in an extra siliceous portion, or in a chloritized zone due to one of the later dikes mentioned before. The value of the contact breccia as an ore zone has been realized only within the last few years, as the rock is perhaps the most difficult of all to prospect.

### *Ores in Contact-Metamorphic Zone*

To this class and the next belong a great majority of the ores heretofore mined in the district. The contact-metamorphic zone, marked by many alteration products in the limestone, but mainly by sericitization of both intrusive and intruded rocks, forms a block around Sacramento Hill and the associated extensions of the porphyry. The ores occur as irregular bodies intimately connected with dikes and sills which are direct offshoots of the main porphyry masses. The replacements extend from some distance in the porphyry, through some of the unrecognizable altered limestones, into the distinct bedding planes of less metamorphosed sediments.

Faulting and crushing previous to the introduction of the porphyry and ore solutions played an important part in determining the location and limiting the size of individual orebodies. Very little post-ore deposition faulting has occurred, so little in fact, that it is a negligible factor for most of the district.

The contact-metamorphic zone occurs almost entirely in Martin and in the first 200 ft. of Escabrosa limestone. Some orebodies, however, are found in the top 100 ft. of Abrigo as shown clearly in Plates 5, 6 and 7. To this zone belongs the ore found along the Dividend Fault, which has played such an important part in structure and mineralization of the district. The rich ore mined early from the Czar and Holbrook was taken out along this fault, and rich ore is still being mined in the Czar from along it.

To this type of orebody belong also a great part of the ores immediately adjacent to the Sacramento dike, Lowell sill, and to dikes in the

Junction and Oliver mines, though these bodies, being outside of the highly altered zone, generally grade into typical replacements of the unaltered limestones.

### *Ores in Relatively Unaltered Limestone*

To this subdivision belong the orebodies in the Southwest, Higgins, Wolverine, Uncle Sam, Shattuck, the south end of the Spray, Irish Mag and Oliver mines, all of the ore in the Cole and Briggs, and the major portion of the ores in the Sacramento, Hoatson and Junction mines. A brief description of each of these occurrences is given below.

The Southwest orebodies, which include the original ore discovery of the camp, occur as replacements of Martin and Escabrosa limestones along the Czar Fault. The original Queen orebody outcropped at the surface, at the big glory hole at the base of Queen Hill. It occurred in the foot-wall side of the Czar Fault, replacing Martin and Escabrosa limestones. Following the Czar Fault to the southwest is a continuous chain of orebodies in the hanging-wall side of the fault, in the same stratigraphic horizon, until the present Southwest workings are reached. At this point the orebodies again cut across to the foot-wall side, but keep in the same horizon. It is these foot-wall orebodies that are being mined at present. The Czar Fault is here broken up and forms blocks with east-to-west faults. In this crushed zone occurs a large mass of brecciated iron-stained silica, which outcrops on the surface on both sides of Hendrick's Gulch. The orebodies are found at and near the contacts of this mass, as replacements of Martin and Escabrosa beds, and penetrate the breccia itself for short distances. The ore is both of copper and of lead.

The ore early mined along the Czar Fault (and now being largely remined), was closely associated with porphyry dikes and sills, which came up along the fault. The present orebodies, however, are not in contact with porphyry. The silica breccia mass, however, in depth is connected with porphyry.

Following the Silver Bear Fault, which forms the eastern boundary of the Queen Hill Block, another series of orebodies occur, also as replacements of Martin and Escabrosa limestone. These orebodies are directly related to porphyry dikes and sills.

The Higgins, Wolverine, Shattuck and Uncle Sam orebodies occur close to and in contact with the Shattuck dike. The Higgins and Wolverine are the farthest west, and therefore farthest removed from Sacramento Hill. Here the dike, as exposed underground, occurs as a dike of unaltered porphyry, and the orebodies are found on both sides replacing Abrigo limestone about 100 ft. below the top. The Uncle Sam and Shattuck orebodies occur as replacements of Martin and Escabrosa limestones, very intimately associated with a large sill of porphyry into which the Shattuck dike makes in depth. At the Shattuck, another mass

of silica breccia occurs, more intimately associated with the porphyry than at the Southwest, with ore formed at the contacts, and penetrating into the mass. Here again the ore is both lead and copper as at the Southwest.

To the south and west of the Contact Block is a relatively up-thrown block, bounded by the Dixie-Howell-Senator Fault on the north and east, and by the Oliver Fault to the southwest, previously mentioned as the Oliver Block. In Irish Mag, Gardner, and Oliver ground, the block is much broken up by N.-S. fractures, and here there are orebodies in the foot-wall side of the Dixie-Howell-Senator Fault replacing Abrigo and Martin beds. In Spray, and in part of the Oliver ground where the block is less fractured, orebodies are found entirely as replacements of Abrigo limestone, from 100 to 300 ft. down from the top. In Spray ground no porphyry has yet been found anywhere near these orebodies. In Oliver ground, they are associated with small dikes.

In the Cole mine, orebodies occur in the Martin limestone and extend for a short distance into the top of the Abrigo, along N.-S. fractures. These orebodies occur in the hanging-wall side of the Oliver Fault. This fault, between the Cole and Oliver mines, has cut off one of these orebodies and is, therefore, post-ore deposition in age, one of the few breaks of its kind in the productive area of the camp. The Cole orebodies are not directly connected with porphyry.

The orebodies of the Briggs mine are similar to those of the Cole in structure, being related to a N.-S. fractured zone, not directly connected with porphyry. They are also replacements of Martin limestone.

The orebodies in the Hardscrabble claim of the Sacramento mine, and part of the Hoatson mine are directly associated with the Sacramento Dike. The orebodies exist as replacements of Escabrosa limestone, the horizon being about 200 ft. up from the base. The orebodies here are in direct contact with the dike on both sides, and make into it for short distances.

To summarize, the orebodies in the unaltered limestone occur for the most part at or near porphyry contacts, but they also occur away from porphyry in fractured country. In fractured country and where intrusions are numerous, the ore horizon tends to rise, and ore is found mostly occurring from the base of the Martin limestone to 300 ft. above the base of the Escabrosa. In unfractured, and relatively less intruded country, on the other hand, the horizon tends to drop and ore is found in the Abrigo limestone.

In the illustrations showing the zones of alteration (Plates 5*A*, 5*B*, 6*A*, 6*B*, 7*A*, and 7*B*) the orebodies in porphyry are seen to be between the sericitized and silicified portions of that formation. The ores in breccia are in a highly silicified zone. The ores in the contact-metamorphic limestones occur where the alteration is mainly sericitic.

Table 2 gives partial analyses of the orebodies along sections shown

by the illustrations and the rocks that they are found in, as well as the character of the ore dependent on oxidation.

TABLE 2.—*Partial Analyses of Orebodies along Sections of Plate 4*

Section A-A

| Rock Replaced | Location | Oxidation Features | Cu, Per Cent. | Fe, Per Cent. | S, Per Cent. | $SiO_2$, Per Cent. | $Al_2O_3$ Per Cent. | CaO, Per Cent. |
|---|---|---|---|---|---|---|---|---|
| Porphyry | East end Sac. Hill | Enriched | 2.76 | 9.3 | 8.0 | 59.6 | 13.6 | 1.2 |
| Porphyry | West end Sac. Hill | Enriched | 1.8 | 9.4 | 9.8 | 60.5 | 11.4 | 0.9 |
| Porphyry | West end Sac. Hill | Enriched | 3.7 | 10.8 | 11.8 | 58.1 | 10.4 | 0.8 |
| Porphyry | West end Sac. Hill | Enriched | 6.6 | 13.6 | 15.8 | 50.1 | 7.8 | 0.7 |
| Porphyry | West end Sac. Hill | Enriched | 7.8 | 17.0 | 20.1 | 44.3 | 6.0 | 0.7 |
| Porphyry | West end Sac. Hill | Enriched | 12.2 | 16.9 | 21.3 | 43.7 | 3.8 | 0.4 |
| Contact breccia | Breccia West of Sac. Hill. | Primary | 6.4 | 19.1 | 22.4 | 40.7 | 5.3 | 0.7 |
| Contact breccia | Breccia West of Sac. Hill. | Slightly enriched | 6.1 | 14.7 | 16.4 | 46.1 | 9.1 | 0.5 |
| Contact metamorphic limestone | Holbrook | Enriched | 8.8 | 22.7 | ? | 21.8 | 15.1 | 0.9 |
| Contact metamorphic limestone | Holbrook | Enriched | 4.4 | 24.4 | 25.3 | 13.0 | 15.1 | 1.0 |
| Contact metamorphic limestone (slightly altered) | Holbrook | Slightly enriched | 5.0 | 31.1 | 35.2 | 15.2 | 6.2 | |
| Escabrosa limestone | Holbrook | Residual carbonate | 6.3 | 22.3 | 0.2 | 10.1 | 5.3 | 14.8 |
| Devonian limestone | Southwest | Residual carbonate | 4.1 | 12.2 | 0.1 | 18.3 | 7.3 | 12.9 |

Section B-B

| Rock Replaced | Location | Oxidation Features | Cu, Per Cent. | Fe, Per Cent. | S, Per Cent. | $SiO_2$, Per Cent. | $Al_2O_3$ Per Cent. | CaO, Per Cent. |
|---|---|---|---|---|---|---|---|---|
| Escabrosa lime and porphyry | Uncle Sam Mine | Residual ore | 14.7 | 26.3 | ... | 20.6 | 7.5 | 2.6 |
| Devonian lime | Uncle Sam Mine | Residual ore | 6.5 | 22.5 | ... | 19.8 | 9.5 | 7.0 |
| Abrigo limestone | Gardner | Primary | 6.1 | 28.5 | 29.3 | 20.7 | 4.3 | 1.7 |
| Devonian limestone | Gardner | Primary | 4.7 | 32.5 | 36.0 | 16.1 | 6.1 | 3.1 |
| Escabrosa limestone | Gardner | Residual carbonate | 8.6 | 25.9 | ... | 20.0 | 5.1 | 3.0 |
| Metamorphosed Devonian | Gardner | Primary | 4.3 | 34.1 | 27.7 | 14.8 | 2.45 | 1.3 |
| Contact breccia | Gardner | Primary | 4.8 | 30.6 | 40.0 | 13.6 | 4.5 | 1.4 |
| Metamorphic limestone. | Sacramento | Enriched ore | 9.3 | 22.7 | 23.0 | 25.6 | 11.4 | 1.3 |

Section C-C

| Rock Replaced | Location | Oxidation Features | Cu, Per Cent. | Fe, Per Cent. | S, Per Cent. | $SiO_2$, Per Cent. | $Al_2O_3$ Per Cent. | CaO, Per Cent. |
|---|---|---|---|---|---|---|---|---|
| Abrigo limestone | White Tail Deer | Residual carbonate | 5.25 | 13.8 | ... | 45.9 | 5.2 | 2.6 |
| Partly altered Devonian | Holbrook | Residual carbonate | 8.1 | 13.2 | Tr. | 26.0 | 10.5 | 1.6 |
| Partly altered Escabrosa | Holbrook | Residual carbonate | 4.4 | 25.1 | Tr. | 19.4 | 9.5 | 1.5 |
| Sericitized metamorphic | Holbrook | Enriched sulphide | 5.5 | 21.1 | ? | 14.9 | 23.0 | 0.4 |
| Sericitized metamorphic lime and porphyry | Holbrook (Dividend). | Enriched sulphide | 6.6 | 17.5 | 12.6 | 17.8 | 23.2 | 0.3 |

## MINERALOGY OF THE ORES

As in the case of rock alteration, primary minerals in the ores have been changed greatly by the circulation of meteoric waters, and the criteria have already been given for determining the primary zone under

the caption, "The Effect of Contact and Hydrothermal Metamorphism on the Formations."

The original sulphides of copper deposited are comparatively few in number, chalcopyrite and bornite being the important ones economically. Tennantite is the only other copper-bearing mineral so far found in the primary zone, if exception be made of one or two unknown minerals seen with the microscope. Pyrite is such an important constituent of the ores, that it must be considered along with the copper minerals, and magnetite, hematite, sphalerite and galena are common enough to be considered part of the ores, especially the last two, since they are the source of oxidized ores of importance.

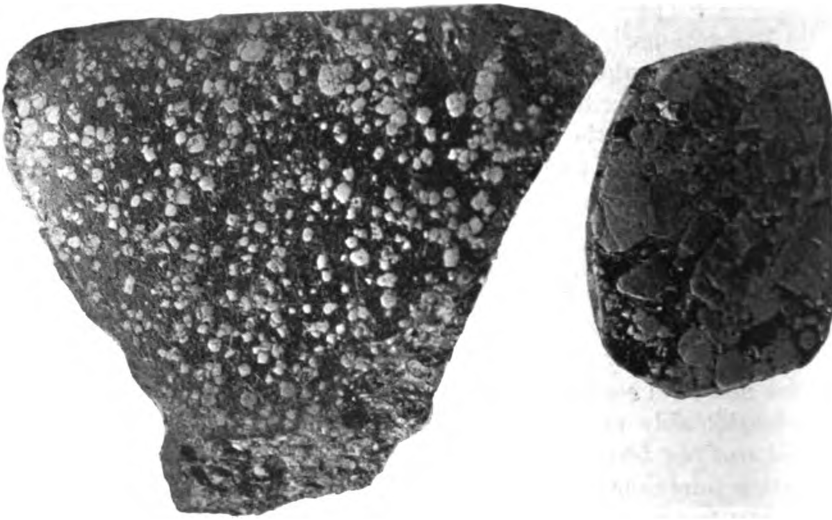
Chalcocite is very abundant in many forms in the zone of enrichment but has not been seen in the undoubted primary zone, though this has been considerably explored. It is therefore highly probable that this mineral was not formed by the ascending solutions in this district, and conflicting microscopic evidence in the ores above has been considered insufficient to offset the field evidence of the primary zone.

### *Pyrite*

Of the sulphides, pyrite is generally the first formed, and accompanies every one of the stages of alteration described under metamorphism. In the ores, very fine examples of the structures described by Graton and Murdoch[8] may be seen, grading from the perfect "porphyritic" relation between well-formed pyrite crystals and the other sulphides, to a mass of pyrite grains with little interstitial chalcopyrite or bornite.

As additional evidence for explaining the mode of formation of some of the sulphides the following observations will be given. Around any orebody, pyrite usually persists in the same general relations to the rock or gangue, that it has to the other sulphides in the ore, with the exception that in the ore the individual pyrite grains are broken up, penetrated or reabsorbed by the other sulphides in such forms as are illustrated in Plate 13, Fig. 2, and Plate 14. Thus, ore in porphyry has pyrite veinlets extending into the rock along joints and it will be found disseminated through the rock in irregular grains that usually started around a biotite. These same pyrite veinlets and disseminated grains will persist even when all the rock has been replaced by sulphides, as can be seen in Plate 13, Fig. 3. In limestone replacements, pyrite will extend out along bedding planes in layers that are continuous with pyrite bands in the copper ore. And pyrite crystals will be found to have the same "porphyritic" relation to slightly altered Abrigo limestone as they have to the rich ore in

[8] L. C. Graton and J. Murdoch: The Sulphide Ores of Copper, *Transactions*, vol. 45, pp. 26–93 (1913).



FIGS. 1 AND 2.—"PORPHYRITIC" STRUCTURE OF ORES IN ABRIGO LIMESTONE (LEFT) AND IN BRECCIA.

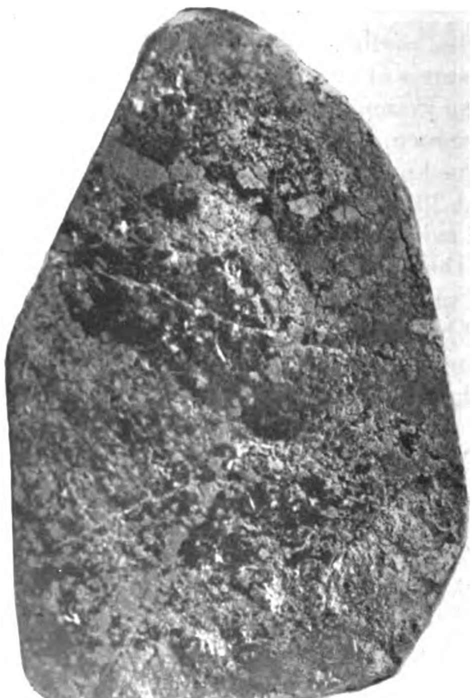


FIG. 3.—REPLACEMENT OF ALL ROCK BY SULPHIDES WITH PYRITE VEINLETS PERSISTING.

PLATE 13.

that formation, with the exception given above, that there is little breaking up of the pyrite (see Plate 13, Fig. 1).

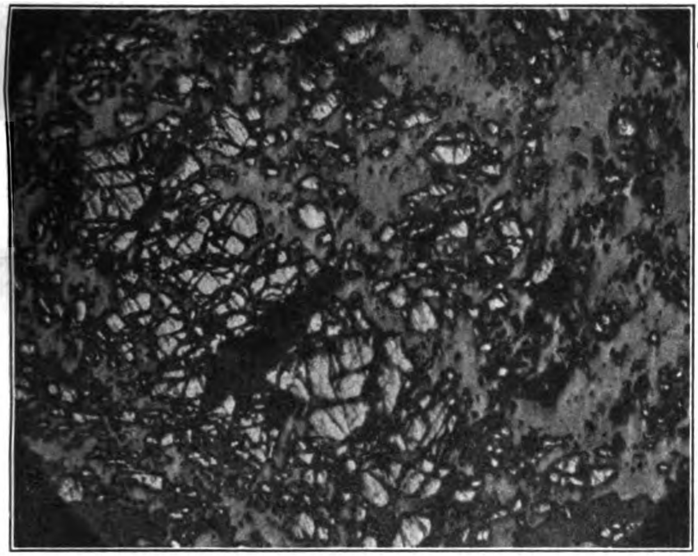


FIG. 1.

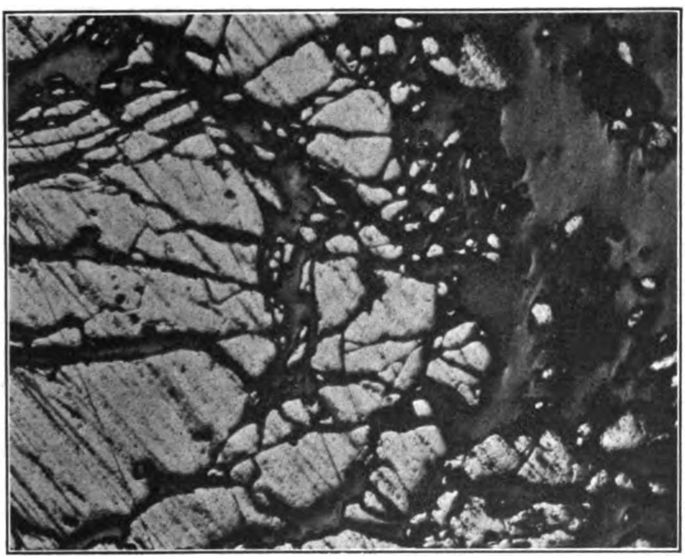


FIG. 2.

PLATE 14.—ORIGINAL PYRITE GRAINS BROKEN AND REABSORBED BY OTHER PRIMARY SULPHIDES.

This information, coupled with microscopic observations concerning the age of crystallization of the pyrite has led to the belief that in the ore

formation pyrite is the advance sulphide, formed partly by solutions that are not yet precipitating copper minerals. When these latter are formed, they take their place by replacement of the gangue and rock mostly, but also by slightly replacing the earlier formed pyrite. This may explain certain veinlet-like stringers of pyrite, which apparently cut the other sulphides, but which when examined in detail show that each grain of pyrite has been attacked by chalcopyrite or bornite.

*Age Relationship of Pyrite.*—In the porphyry ores, pyrite is found to have crystallized at about the same time as the sericitization occurred with some of the grains a little earlier. Pyrite deposition also extended into the period of silicification, but was undoubtedly also replaced by quartz. Bornite, the copper mineral, is found replacing the pyrite as already mentioned.

In the contact breccia, due to the many complicated interrelations found, the pyrite cannot be said to have formed at any definite stage. In general it holds true that pyrite is earlier than the rest of the sulphides, but distinct veins of it have been found through crushed pyrite, bornite and chalcopyrite ore. Magnetite, specularite, and sphalerite are of about the same age as the pyrite, and garnet, diopside and wollastonite slightly earlier.

In the contact-metamorphic limestone orebodies, pyrite and sericite again belong to the same period of formation, earlier than the copper sulphides, and slightly later than tremolite and other contact silicates. In this zone, pyrite veins are found cutting magnetite, specularite and earlier pyrite bodies.

In the orebodies that replace unaltered limestone the relations are the simplest, pyrite being closer in age to the copper, zinc, and lead sulphides than in any other place.

### *Chalcopyrite*

This mineral is undoubtedly the most abundant primary copper sulphide in the district, and is prevalent in all the ores, except in the disseminated porphyry ore in Sacramento Hill. This ore is distinguished mineralogically from all others in the district by having as undoubted primary sulphides, pyrite and bornite alone.

Chalcopyrite is generally associated with the more siliceous ores, except in porphyry, its period of formation being slightly later than that of sericitization, but contemporaneous with silicification. In cases where there is late chloritization, some remarkably pure bodies of chalcopyrite are formed, as is also the case in certain vein-like bodies extending far above the general ore horizon, where galena is also abundant.

The intergrowths of chalcopyrite and bornite show them to be of contemporaneous precipitation, with a slight lagging of the bornite formation.

### *Bornite*

This mineral is universally found in the primary ores, though not as abundantly as chalcopyrite, as has been stated.

Although bornite occurs undoubtedly in the richer primary ores, there has been no reason found for its formation in preference to chalcopyrite, as it comes in rich and lean ores, and in those with a great amount of pyrite, also when there is scarcely any pyrite. Fig. 1, in Plate 18, illustrates a case where bornite and pyrite replace the cement in a calcareous Devonian sandstone, where the amount of copper is not enough to give 0.2 per cent.

### *Tennantite*

This mineral is found usually in microscopic amounts only, in practically all the ores. Its age of crystallization is between that of pyrite and chalcopyrite. There may be some tetrahedrite, but as arsenic is more abundant than antimony in the specimens analysed, the mineral seen has been taken to be tennantite.

### *Sphalerite*

The sulphide of zinc is found in all the primary ores in minor quantities, with zonal inclusions of chalcopyrite and galena in very fine grains (Plate 17, Fig. 1). In a few places such as between the Southwest and Czar mines, in the Gardner, and in Briggs ground, sphalerite is found in rather large bodies with pyrite and chalcopyrite. It generally replaces Martin limestone, and apparently its general relations are no different from those of the copper ores.

### *Galena*

Like sphalerite, galena is associated with most of the primary ores, but is decidedly more abundant where the ores are in highly chloritized ground. The ground adjacent to the latest basic porphyry dikes, in contact breccia and in limestone, generally has more galena than the rest. An exception to this may have been the zones around the silica breccia masses of the Southwest and Shattuck mines, but unfortunately oxidation has obliterated thoroughly all primary mineralization. Lead carbonates and sulphates are here found in abundance.

Galena is found more abundantly also in the top zone of the ores, as a residual sulphide, so the general impression given by the occurrence of the lead sulphide is that it is more abundant in the last stages of mineralization and also that it was carried farther than the copper.

### *Other Primary Metallic Minerals*

Magnetite and specularite occur sometimes in large bodies that show a very perfect intergrowth of these oxides, as well as with pyrite, as has been mentioned. The occurrence of these bodies is limited to the contact-metamorphic zone, or is immediately on the contact of porphyry dikes. A few grains of pyrrhotite have been seen here, indicating the high temperature of their formation.

Specularite is also found associated with the orebodies in contact breccia that come with the basic, chloritizing porphyry dikes. The oxide is here a valuable indication of the proximity of ore, in a zone where all other features are baffling in their irregularity.

Silver and gold minerals have not been seen in the primary zone, with the possible exception of a few grains of polybasite found in galena. Gold and silver values are considerable in all the copper ores, and more so probably in the sulphides than in the oxides, but their amount could easily escape detection even under the microscope. Tennantite, as well as one or two brownish unknown minerals in the sulphides may be the precious-metal carriers.

## IX. OXIDATION AND ENRICHMENT

### General Features

The most striking feature of oxidation of the orebodies in the Warren district is the varying elevations at which the column, from oxidized to primary ores, repeats itself in going from the western to the eastern side of the mining area. At the extreme western end, the Higgins and Wolverine mines have primary ores in Abrigo limestone far above the level of the adjacent creek bottoms, and consequently far above the present ground-water level. In the Czar mine oxidation and enrichment go down to just under the 300 level, which is about 5,000 ft. above sea level. In the Holbrook division some primary ore outcropped at the west of Sacramento Hill, while some distance south, enrichment goes down below the 500 level. In the Gardner, primary ores are not found till the 9th level is reached, and in the the Lowell mine they are encountered, in some instances at the 10th level, but generally below the 13th, although the permanent water level was encountered here around the 10th level, or at an elevation of 4,325 ft.

In the Hoatson mine, oxidation features were found down to an elevation of about 3,800 ft. above sea level, and even deeper in part of the Sacramento and Junction mines, although the permanent water level was far above this before the mines were drained by opening up and pumping from the deeper levels of the Junction.

The oxidation features shown by the illustrations of the sections through the camp (Plates *5b*, *6b*, and *7b*), illustrate this general inclination to the southeast. It will be noticed that the oxidation follows in general the dip of the Paleozoic sediments, the average horizon reached being somewhere in the Devonian. There is a marked lagging of the oxidation at the eastern end, however, as here it does not reach the top Devonian. In this, the oxidation may be seen to correspond more closely to the pre-Cretaceous erosion surface.

It will be seen also that there are some sudden changes in the level of oxidation, as best shown in section A-A of Plate 5, and also that there are great variations in the extent of the zone of secondary sulphides. All these features have fairly definite relations to the geologic history and structure, to the nature of the formations, and to the primary alterations in them. The object of this chapter will be to explain what is known of those relations.

It will first be necessary to present the criteria for the subdivisions of the zone in which meteoric waters have circulated, in many directions, but generally downward. It has been stated before that the primary zone was marked by steady conditions and constant mineralogical associations. The deepest effects of the surface-water circulation are immediately and clearly marked in this district by veinlets, filled cavities and coatings of other minerals.

Just above the primary zone there is an extremely variable thickness where the prevailing secondary mineral is siderite with a little gypsum and an extremely small amount of chalcocite. Above this is usually a zone where halloysite and kaolin are most prominent among the secondary gangue minerals, and chalcocite is abundant as a replacement of primary sulphides. Next higher, limonite, and the staining due to iron oxides—gossan formation, in other words,—is the prevailing feature. Within this zone there are found the local bodies of residual ores, which have for some reason resisted the general leaching effect of strongly oxidizing waters. One or all the zones above the primary may be lacking at any point for reasons which will be explained later.

## Relations of Oxidation to Structure and Geological History

Notman[9] has already pointed out the relations between the attitude of the oxidation and present and past water levels. That the present water level does not account for the oxidation is proved by the occurrence of unoxidized ores above that level in formations that are in other places completely oxidized also above that water level. This is illustrated by the primary ores in Abrigo at the Higgins mine, and the completely oxidized Abrigo ores in the Wolverine and White Tail Deer mines.

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[9] *Transactions of the Institution of Mining and Metallurgy*, vol. 22, p. 561 (1913)

Further proof is had in the presence of gossans several hundred feet below the water level in the Lowell and Hoatson mines.

The general plane of the oxidation corresponds very closely to the known pre-Cretaceous erosion surface, and the tilt of the oxidation is, therefore, the present tilt of the Cretaceous sediments. If the ground-water level at the time of oxidation had been an inclined plane that followed the Devonian horizon, due to the permeability of the purer limestone above, there would not be abundant unoxidized orebodies in Escabrosa limestone in the Sacramento and Junction mines. And if the ground-water level had stood several hundred feet lower in those mines at some post-Cretaceous time and had risen, say, due to valley fillings, it is probable that the ground water, now directly over that rise, would stand horizontal, and would keep to the old plane, or even a lower one farther west, which is far from being the case.

There is field evidence of another form to show that the oxidation was pre-Cretaceous, and this is the presence of gossanized fragments of porphyry, as well as fresher ones at the base of the Glance Conglomerate. The exposed pre-Glance topography also shows hard gossan reefs around the southeast side of Sacramento Hill.

Additional weight is given to the idea of pre-Cretaceous oxidation by the geologic history of this region. A very long erosion and a consequently long oxidation period is recorded between Pennsylvanian and Cretaceous times by the disappearance of the paleozoic column off of the north side of the Dividend Fault. And while the granite porphyry intrusion may not have taken place till a good part was gone, the nature of the rock as well as of the later minerals formed presuppose a considerable covering of sediments which were removed by erosion before Cretaceous time.

It is, therefore, probable that the main part of oxidation and enrichment as found at present took place when the Paleozoic sediments were almost horizontal, the only tilt noticeable being slightly to the southeast.

There is very little evidence of superimposed oxidation since Cretaceous times, due very probably to the later ground-water level being higher than the first. Narrow, limonitic seams, and very slight enrichment on primary sulphides in the west end of the camp are the only visible effects of this oxidation. The post-ore faulting which is also post-oxidation in most cases, is not found very frequently in the thoroughly explored portion of the district, but will very probably be found more frequently when operations extend into other blocks.

## Relations of Oxidation to the Formations and to Metamorphism

The circulation of meteoric waters has been very much influenced by the material they traversed, and this was dependent upon the formation and the degree of alteration.

In the Carboniferous limestones the massive purer beds had a tendency to crack and break rather than to fold, and waters could dissolve out channels through which extensive oxidation could be carried down rapidly. The same applies to the purer lime beds in the Devonian and Cambrian formations.

The more impure beds, which are also more thinly laminated, had a tendency to crush and fold, and while as a whole they may be found more altered by meteoric waters, it is because they retarded and distributed the effects all through the rock. This is noticeable in the Martin and Abrigo formations especially, the Abrigo being by far the most impervious.

The ability of meteoric waters to circulate and oxidize quickly in the Escabrosa limestone has made possible the abundance of residual ores in this formation. These are mostly carbonates fixed in this form against total leaching by the continually circulating cold waters. In a great many cases, however, the residual ore is found in the form of sulphide, mostly chalcocite, and also, in rare instances, as the original chalcopyrite and bornite. These sulphides are called residual because they are far above the level of normal enriched sulphides, and they have limonitic gossans or carbonate and oxide ores under them. Usually they are very pure, rich sulphides of copper, and are in the primary zone of sulphide replacement where the adjacent rock is pure calcite and contains little or no pyrite. Occurrences like this are found in the limestones all the way to surface where the main ore zone may be a thousand feet below.

The same quick circulation through the Carboniferous limestones has usually brought the zone of complete oxidation directly against the primary ores, with a thin intervening zone of secondary sulphides, which, as will be shown later, is not one of enrichment, but of leaching of values. The slower circulation in the impurer Devonian and Cambrian produced fewer residual orebodies, but a deeper zone of enrichment.

In the porphyry and contact breccia the differences in the effect of secondary alteration are far more noticeable, as can be seen from Plates 5*b* and 6*b*. Both the porphyry and breccia retard the meteoric water circulation, the breccia almost entirely, and the porphyry making it slow enough so that sulphide enrichment is at its most efficient point.

A special case which illustrates the difference in the penetration of oxidation through different rocks is given in Plate 12, which is an east-to-west section through an orebody on the 1,500 level of the Sacramento mine, at present being extracted. This ore occurs along the Sacramento dike, here cutting Escabrosa limestone and having a projection downward to the east as shown. The ore replaces the limestone, which is not appreciably metamorphosed at a distance from the porphyry. The replacement is very perfect along the beds, as certain cherty bands are left untouched in the midst of the ore.

The oxidation feature that is noticeable is that the ore is almost unaltered close to the main dike, enriched farther away, converted into mixed oxide and sulphide under the thin portion of the porphyry, and entirely into carbonate next to the limestone. The water circulation here must have come partly from the east and partly from underneath through the limestone, as the porphyry ordinarily presented an effective barrier to circulation perpendicular to the beds before the tilting.

The oxidation features are even more noticeably related to the processes of primary ore alteration than they are to that of the formations.

Intense silicification such as is found in the contact breccia and in a great part of the contact metamorphic limestone of the Gardner mine made circulation so slow that in these places the primary ore zone remains at a level far above those of adjacent occurrences. Also, the oxidation is so complete where there is any meteoric circulation at all that the zone of leaching comes to within a few feet of the primary zone, with only a thin layer between where covellite is most abundantly found in the district. These relations are well shown in Plate 5, Section A–A just west of Sacramento Hill.

Sericitization produced rocks in which the circulation was slow, but in which the secondary waters had opportunity to remain acid, as nearly all the lime was removed in the primary alteration and pyrite was abundant all through the rocks. Under these conditions the depth of the zone of secondary enriched sulphides was greatest, and the enrichment most efficient. At the same time, in the rocks very much sericitized there are no residual carbonates found, due of course to the lack of lime. If, however, the ore formation extended into unaltered limestones above those sericitized, carbonates might be found higher, as is the case in the Uncle Sam mine. Native copper and oxides are developed to a great extent in a shallow zone between the gossans and enriched sulphides in aluminous ground due to primary sericitization.

The ores in Sacramento Hill porphyry, and in the zone of contact-metamorphic rocks around the main intrusives are the best examples of the coincidence of sericitization and deep sulphide enrichment, as can be seen in Plates 5*a*, 5*b*, 6*a* and 6*b*. The amount of enrichment even in these cases has not been exceedingly great, as the original ores were rich to begin with, and very pyritic bodies have remained poor, first because pyrite is replaced with so much more difficulty by chalcocite than bornite and chalcopyrite, and second, because pyritic masses are generally quite solid, more siliceous and impervious.

The ore developed in Sacramento Hill is extremely irregular in shape and grade and owes its irregularity more to primary mineralization than to secondary enrichment, because so far as has been seen, the relative amount of primary bornite left in parts at the top of the ore close to the oxidation line is about the same as at the deepest levels. A section show-

ing the ore grades is given in Plate 11, worked out from all information available. This body has not yet been mined extensively. The grades of ore can be seen to bear very little relation to the limonitic oxidation.

The orebodies classed as sulphide replacements of unaltered limestones are ordinarily found in their primary state or as residual carbonates. Oxidation is rapid through unaltered rocks, and waters usually carry enough lime to fix the copper as carbonate. In such a zone, carbonates are apt to be found to the very top of the mineralized zone, many times above hundreds of feet of gossanized ground.

TABLE 3.—*Showing Efficiency of Enrichment by Descending Waters*

| Orebody | State of Oxidation | Tons per Cu. Ft. | Pounds Cu per Ton | Pounds Cu per Cu. Ft. |
|---|---|---|---|---|
| | **Sericitised Contact Metamorphic Zone** | | | |
| 14-7 | Primary | 0.1143 | 126.2 | 14.42 |
| 13-13 | Enriched sulphide | 0.1008 | 189.8 | 19.10 |
| | **Sulphide Replacement Orebodies in Abrigo** | | | |
| 12-3-18 | Primary | 0.117 | 104.0 | 12.17 |
| 13-10 | Enriched | 0.0984 | 147.0 | 14.46 |
| White-Tail Deer | Residual carbonate | 0.0578 | 142.0 | 8.20 |
| | **Sulphide Replacement Orebodies in Martin and Escabrosa** | | | |
| 9-8-58 | Primary | 0.1255 | 130.0 | 16.31 |
| 8-17-11 | Enriched and oxide | 0.098 | 112.0 | 10.97 |
| 8-17-4 | Residual | 0.0609 | 113.0 | 6.88 |
| 8-16-5, 8-16-16 | Residual | 0.0609 | 137.0 | 8.34 |
| 8-16-18 | Residual | 0.0609 | 108.0 | 6.58 |
| | **Sulphide Replacement and Contact Bodies in Escabrosa** | | | |
| 1300 Junction | Primary | 0.1111 | 116.0 | 12.90 |
| 16-3-39 | Primary | 0.140 | 140.2 | 19.63 |
| 16-3-68 | Partly enriched | 0.1267 | 192.4 | 24.38 |
| 15-12 Sulphide | Secondary sulphide | 0.115 | 130.0 | 14.90 |
| 15-12 Oxide | Oxide | 0.08 | 224.0 | 17.92 |
| 16-3 Oxide | Oxide | 0.08 | 268.0 | 21.40 |
| 15-6-1, 15-6-9 | Residual carbonates | 0.065 | 170.0 | 11.05 |
| 15-13 | Residual carbonates | 0.065 | 188.0 | 12.20 |
| 14-20 | Residual carbonates | 0.065 | 114.0 | 7.41 |

Table 3 has been prepared to show approximately the efficiency (or lack of it) of enrichment due to descending meteoric waters.

Orebodies have been taken for analysis which were originally of similar composition, as far as knowledge of them goes, and from mining records kept for a period of several years, the contents of copper in pounds per cubic foot in place has been figured. This is of course a figure for the ore mined, and must differ somewhat from the actual content of all the ground in an orebody, depending on the method of extraction used. All errors have been considered, however, and the relative value of the figures is about correct. Unfortunately the ores in the breccia and silicified ground are all primary and no comparisons can be given, and all the ore in porphyry is enriched.

From these tables it will be seen that the copper content per unit of volume is increased materially in the sericitized contact zone and in the more aluminous Abrigo formation where the secondary sulphides are found. Chalcocite formation is accompanied by leaching out of values in the replacement bodies, and all the residual ores show dissipation of copper contents. The narrow oxide and native copper zone is generally the richest of all.

## Gossans

In regard to the gossans formed in the different zones of primary alteration, they can be said to extend downward for the greatest distances in the more permeable ore zones—this is true for those classed as sulphide replacements. These gossans are likely to contain residual ores anywhere. The thoroughly oxidized portions, where the formations were silicified or sericitized or chloritized are apt to be more completely leached out, and penetrate the ore zone least in highly quartzose ground. In composition the gossans of course reflect their source.

The only generalization that can be made about gossans derived from lean sulphides and from pyritic copper ores is as follows. The mixed sulphides of copper and iron are more readily attacked by oxidizing solutions than the straight iron sulphides, therefore producing stronger solutions. These stronger solutions have been found to have a segregating action on the components of the surrounding rocks, separating to some extent silica, alumina, and lime, and also depositing the iron oxides in purer form. The gossans from pyrite alone have been observed to leave the gangues and oxides of iron in about the same mixtures as in the original primary material.

These observations are of course susceptible of an infinite number of variations depending on the original composition of sulphides, of their gangues, and of the surrounding rocks, but in general they can be used to some extent in judging the kind of ground under an oxidized portion of an ore zone.

## Minerals of the Zones of Enrichment and Oxidation

### *Siderite*

Under the zone of secondary enrichment the presence of siderite has already been noted. This mineral occurs sometimes in veinlets replacing either sulphides or chloritic minerals or calcite. At other times siderite is found in a curious box-work honeycomb structure as if it had replaced laminated rock along certain planes and the intervening material had been later dissolved out. The ferrous carbonate has a distinct selective action in replacing a rock, as it usually leaves intact pyrite grains, which then appear contemporaneous with the siderite. Sphalerite and galena are also found more abundantly with the siderite than around in the unaltered formation, and this fact has pointed to the possibility that these two sulphides may have been here deposited by secondary solutions.

Copper minerals in the siderite zone are very faintly coated with chalcocite. Gypsum is also found in vugs, even deeper than siderite, but apparently in late circulation channels.

Siderite probably formed as an advance mineral, as oxidation progressed downward. This may be proven by the closely similar forms taken by limonite in some gossans above enrichment, and the presence of siderite cores in that limonite. The iron carbonate has played an important economic rôle in serving as a precipitant for rich cuprite ores in the Czar mine, where copper-laden solutions have encountered a flat bed in Devonian limestone previously replaced by siderite and kaolin, and have precipitated cuprite and native copper in considerable amount. Throughout the district cuprite crystals can be found on oxidized or partially oxidized siderite.

### *Chalcocite*

In the zone of sulphide enrichment chalcocite was the only secondary mineral of copper formed to any extent. It was usually formed by replacement of the bornite and chalcopyrite alone, leaving the pyrite untouched, or only slightly coated. The broken up, replaced pyrite grains in the primary ore remain in just about the same amount and with the same relations to chalcocite that they had to the primary minerals. When chalcocite and pyrite are the only remaining sulphides, it is difficult to believe that the iron mineral has not been abundantly attacked by secondary action. Pyrite has been replaced to a considerable extent, however, in the enriched ores of the contact-metamorphic type, by a sooty variety of chalcocite that possibly contains considerable covellite, since the latter has been seen coating pyrite grains in these ores.

In replacing the primary minerals chalcocite behaved in different ways depending on the general feature of the orebody. In the ore of

Sacramento Hill, pyrite is barely coated in the very sericitic portions of the porphyry, and is not affected appreciably in silicified rock. The selective action in replacing bornite is better marked here than anywhere


Fig. 1.

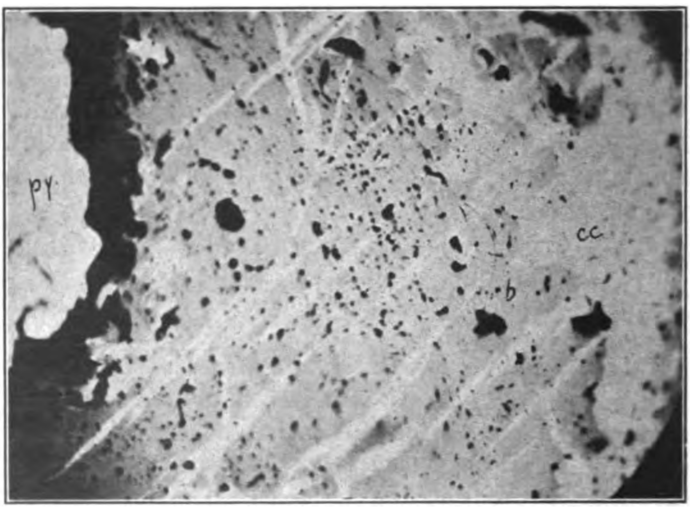


Fig. 2.

Plate 15.—Chalcocite, *cc*, Replacing Bornite, *b*. Pyrite, *py*.

else as is also the replacement along crystallographic lines of the bornite in the arrangement commonly known as "lattice structure." This form of alteration grades into the usual one of veinlets and irregular remnants, as is shown in Plate 15. The field occurrence of this "lattice

structure" alteration has been quite carefully followed in the district and found everywhere to coincide with slow or incipient enrichment in rich ores, and where the selective enrichment of bornite in place of chalcopyrite is well marked. This has led to the conclusion that it is a delicate process in which crystallographic weakness can be of some influence. Etching of the chalcocite thus formed shows isometric figures, probably due to included or dissolved cupric sulphide,[10] but there are also stringers in which the chalcocite has orthorhombic cleavage, and these stringers can be seen to form a network of veins. The proximity and association of this orthorhombic chalcocite to major cracks and circulation channels, where even limonite is formed, points to its being due probably to the working over and recrystallization of the copper sulphides first derived from bornite.

Chalcocite of the "sooty" variety is found most commonly in the enriched ores of the zone of contact-metamorphic limestones. Massive chalcocite is here rare, and then only at the top of the zone of enrichment, with considerable native copper present.

In the ores that replace limestone, the formation of chalcocite is accompanied by leaching and formation of abundant cavities in the ore, a process that is reflected in the decrease of values as previously shown. This same thing is true in ores that have a very siliceous gangue, such as those in contact breccia. Pyrite is here oxidized to limonite even before the copper minerals are replaced by chalcocite, as is shown in Plate 16.

In the Southwest mine there is an occurrence of chalcocite that is somewhat different from the usual. The Devonian-Escabrosa contact beds are replaced by silica, almost chalcedonic in character, accompanied by abundant fine flaky specularite. This siliceous mass connects up with the main silica breccia replacement in this mine.

The top Devonian beds under those silicified have been hardly altered at all, except for a very slight amount of quartz addition, some recrystallization of calcite in stringers, and the formation of very fine veinlets as well as disseminated grains about a millimeter in diameter of what was apparently bornite and chalcopyrite, now replaced almost entirely by chalcocite. The original copper minerals have been found partially replaced by chalcocite in the center of coarse calcite veins that protected them. Pyrite is very rarely found.

The formation of chalcocite has taken place in this instance under conditions which would hardly allow the circulation of acid waters. A considerable amount of carbonates has formed, but in general the sulphide is relatively abundant. The possibility of chalcocite formation with the acid generated in the replaced mineral itself is indicated in this orebody, though this is by no means perfectly clear.

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[10] Posnjak, Allen and Merwin: *Economic Geology*, vol. 10, No. 6.

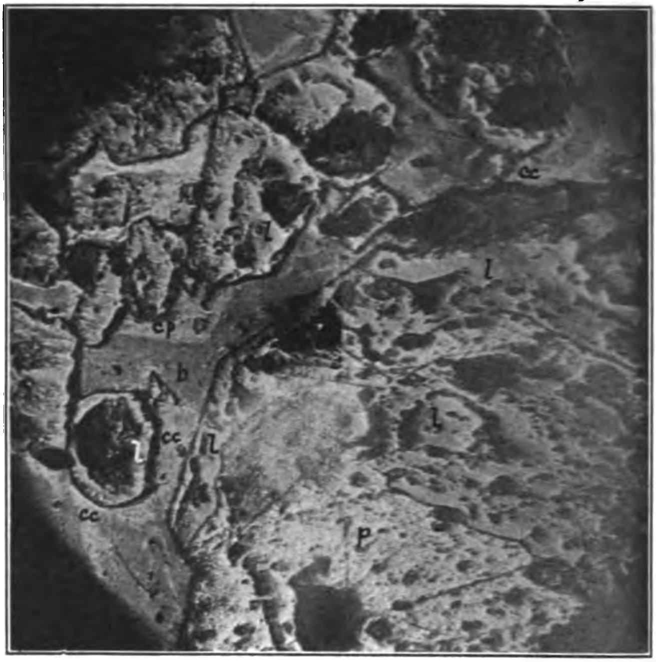



FIG. 1.

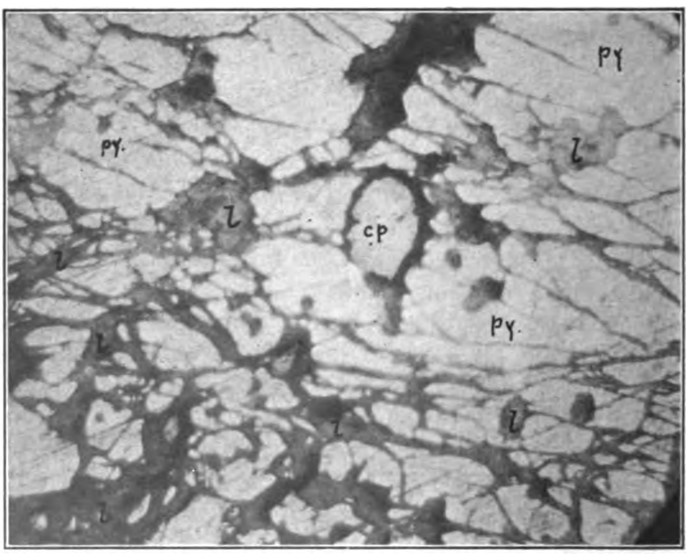



FIG. 2.

PLATE 16.—SHOWING OXIDATION OF PYRITE, *py*, TO LIMONITE, *l*, BEFORE REPLACEMENT OF COPPER MINERALS BY CHALCOCITE, *cc*.

### *Covellite*

Covellite is formed in very minor amounts especially in the narrow oxidized zone above the contact breccia ores. It replaces chalcopyrite in preference to bornite, and is found always in microscopic needles. The only other occurrences of this mineral are where the enriched zone touches highly silicified, not very sericitic ores.

Bornite and chalcopyrite have been found in many ores of the district, formed by secondary agencies, as transition products, but they are of no economic importance.

### *Gangue Minerals*

The gangue minerals that accompany the enriched sulphides are halloysite, kaolin, gibbsite, alunite, and serpentine with mixtures of quartz, calcite and the oxides of iron. Of the aluminous minerals, halloysite seems to be the most common, especially in the zone of highly altered limestones. Where the rock is sericitized so that the mica flakes are well formed, sericite persists unaltered through the process of enrichment and oxidation.

### *Oxide Ores and Native Copper*

In the oxide ores just above the secondary sulphides, cuprite, native copper, tennorite, melanochalcite and delafossite are found, but the first two are the only ones of economic importance. The association of cuprite and siderite has already been given. The zone of oxide has in many instances encroached entirely on the sulphides, leaving only a remnant here and there of the latter. Flat orebodies along the limestone bedding are especially susceptible to this process.

The minerals of the residual orebodies are mostly the carbonates malachite and azurite, with some chrysocolla, aurichalcite and brochantite. Besides these, there are the sulphides already mentioned.

### *Carbonates*

The association and the occurrence of the carbonates has been very well described by Dr. Douglas[11] and by Ransome[12] and no additional features can be given, except to repeat that these minerals are found all through the oxidized portion of the ores that replaced unaltered sediments, and are still a very important part of the ore reserves of the district.

### *Other Ores*

In recent years oxidized lead ores have become more and more important, especially from around highly siliceous replacements in Martin and

[11] *Transactions*, vol. 29, pp. 511–546 (1900).
[12] *Loc. cit.*, p. 125.

Escabrosa limestone. Most of this ore has come from the Southwest mine and occurs at the edge of the siliceous breccia, as cerussite with some

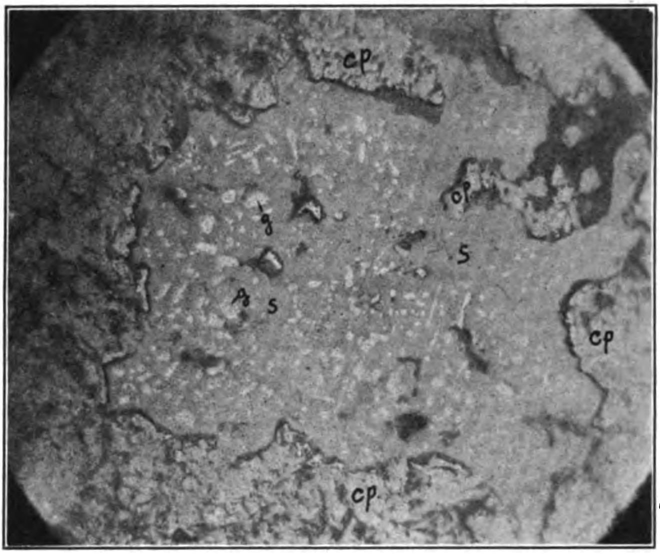



Fig. 1.—Sphalerite, *s*, and Galena, *g*, Included in Grains of Chalcopyrite, *cp*.

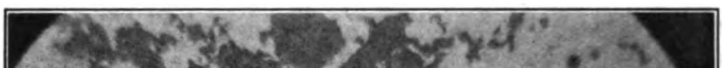



Fig. 2.—Chalcocite and Malachite. Alteration Resembling Intergrowth.
Plate 17.

anglesite and variable silver and gold values. The primary minerals from which this ore was derived are unfortunately not present now, but

it is very probable that the main one was galena, which, as has been before stated, is apt to come in vein-like masses above the general level of the copper ores. Lead ores have also been found with similar associations in the Gardner, Lowell and Briggs mines.

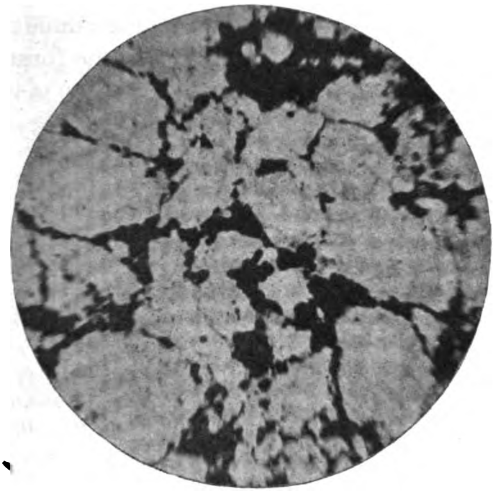


FIG. 1.—BORNITE AND PYRITE REPLACING THE CEMENT IN A CALCAREOUS SANDSTONE.

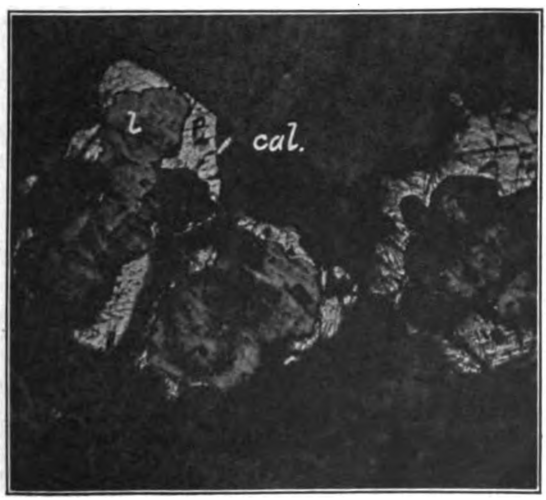



FIG. 2.—PYRITE REPLACED BY LIMONITE.
PLATE 18.

Vanadium, in the form of cuprodescloizite has been found in the Shattuck mine in considerable amount, and rarely in the Dallas and Sacramento. The occurrence may prove in future to be of economic importance.

Manganese ore, in the form of psilomelane and braunite, with little pyrolusite, has been mined lately from several places on the surface of the district. It is found in irregular bunches as a replacement of limestone beds in the Naco or Escabrosa, close to the contact breccia, or to highly silicified outcrops of the sediments. The origin of this manganese is not yet known, for although manganese is found quite abundantly with some of the carbonate ores underground, it is here in the form of soft earthy pyrolussite due apparently to the concentrating action of oxidizing water.

## Bibliography

The most important published works on the geology of the Warren district are the following:


JAMES DOUGLAS: The Copper Queen Mine. *Transactions*, vol. 29, pp. 511–546 (1900).

FREDERICK L. RANSOME: Geology and Ore Deposits of the Bisbee Quadrangle. *U. S. Geological Survey Professional Paper* No. 21 (1904).

*Bisbee Folio, U. S. Geological Survey*, No. 112 (1904).

ARTHUR NOTMAN: The Copper Queen Mines and Works, Part II. *Transactions of the Institution of Mining & Metallurgy*, vol. 22, pp. 550–562 (1913).

W. L. TOVOTE: Bisbee, a Geological Sketch. *Mining & Scientific Press*, vol. 102, pp. 203–208 (February, 1911).


Mention must also be made of the report on the mining geology of the Copper Queen property by John Mason Boutwell prepared in 1908–1909 for the use of the mines, one of the results of this work being the establishment of the present Geological Department.

## Acknowledgments


To the Copper Queen Consolidated Mining Co. for its support and encouragement in the preparation of this report, the greatest thanks are due.

Arthur Notman, Chief Geologist of the Company, has lent the utmost assistance, contributing from his store of knowledge gained by long and careful investigation in the district and in this part of the Southwest as a whole.

Former members of the Geological Department, and our colleagues of the Calumet & Arizona Mining Co., have also contributed materially to the information presented.

To Harvard University, and Prof. L. C. Graton in particular, we are indebted for being able to present the result of extensive petrographic and metallographic investigation, as 2,000 specimens from the Copper Queen geological collection were studied by one of the authors in the geological laboratories at Cambridge. Some of the photomicrographs were taken at the University of Arizona, for which thanks are especially due to Prof. C. H. Clapp.

Assistance has been received from the engineering department of the company in the way of maps and stope records, and from the chemical department in the form of the analyses that are given in the report. Thanks are due to the operating department of the mines for a great deal of information, also for coöperation in pointing out and solving most of the geological problems of the district.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Mine and Mill Plant of the Inspiration Consolidated Copper Co.

BY H. KENYON BURCH, B. S.,* MIAMI ARIZ.

(Arizona Meeting, September, 1916)

## CONTENTS



* Chief Engineer, Inspiration Consolidated Copper Co.

## INTRODUCTION

The Inspiration Consolidated Copper Co.'s plant at Miami, Ariz., was designed and built to make possible the profitable working of a low-grade, finely disseminated copper deposit containing 100,000,000 tons of ore averaging 1.64 per cent. in copper.

From the beginning it was evident that the plant could not be kept integral but that a break would have to be made somewhere in the flow-sheet, removing at least the concentrator to a site more suitable than any available near the mine. It was finally decided, after considering numerous arrangements, to do the coarse crushing at the mine, to store the crushed ore in a bin from which it could be loaded into railroad cars and to haul it to the concentrator, an excellent site for which was available about 1¾ miles from the mine.

The original intention was to equip a plant to treat 7,500 tons of ore per day, but through the acquisition and proving up of additional ore reserves, the introduction of the Ohio caving system, and the excellent results obtained in the test mill (which made it possible to treat a lower-grade ore than had been thought possible) it was evident that a plant of much greater capacity should be supplied. It was, therefore, decided to treat approximately 15,000 tons of ore per day, the duties of the four main divisions to be as follows:

| Division | Operating Time in Hours | Capacity in Tons per Hour | Daily Capacity in Tons | Available Capacity in Tons |
|---|---|---|---|---|
| Hoisting plant...... | 15 | 1,000 | 15,000 | |
| Crushing plant..... | 15 | 1,000 | 15,000 | |
| Storage bin........ | .... | ...... | ...... | 25,000 |
| Concentrator....... | 24 | 625 | 15,000 | |

It was realized from the beginning that with such an enormous tonnage to treat it would be well worth the time and cost to carefully work out a flow-sheet. Accordingly, a gravity test mill was erected and placed in operation near the Joe Bush shaft in November, 1910, and its operation was continued until August, 1911. Soon after this, flotation began to attract considerable attention in this country and realizing the possibilities that might arise through its systematic investigation, it was deemed advisable to go into the process in detail. A 50-ton Minerals Separation machine was first erected, which after a few months thoroughly demonstrated that flotation was applicable to the concentration of the Inspiration ores, but in order to better determine its proper place in the flow-sheet, a 600-ton test mill was designed and erected in 1913 on the benches of the concentrator site, the grading for which was at this time completed. Numerous flow-sheets were experimented with, the final result being the

one now in use in the concentrator. A discussion of certain phases of the results accomplished in the test mill will be given in another section, mention being made of it here simply for preserving the proper sequence.

That the large-scale test-mill method for working out flow-sheets for large plants is the only logical method, is evidenced by the fact that in nearly every stage of the treatment either an entirely new machine has been adopted or a new application has been made of a standard machine; the result in each case being either increased efficiency or a more economical arrangement. The first mill was to have a capacity of 7,500 tons per 24 hr. This design covered an area of approximately 350,000 sq. ft., or a little over 8 acres. The mill now in use covers an area of approximately 125,000 sq. ft., or a little less than 3 acres, but has a capacity of 15,000 tons. The recovery in the two types of plants on favorable ores, that is, ores not carrying over 10 points of oxide, may be closely estimated at 70 per cent. for the first, and 85 per cent. for the second.

In order to account for the long period required for the design and construction of the plant, it may be interesting to note that six complete designs for the concentrator were executed, the idea being to keep this work abreast of the developments brought out by the test mill. Design No. 2 was completed and a contract entered into for structural-steel requirements. A portion of the steel had been fabricated when flotation developments pointed to the fact that wet gravity concentration could be greatly improved upon. At this point all work on the steel contract was stopped and that portion of the contract pertaining to the concentrator was cancelled. Although facts relative to flotation continued to develop, in July, 1913, the steel design for the present concentrator building was completed. This building was no sooner erected than very marked changes in grinding machinery began to develop, the ultimate result of which was another altogether new arrangement for the entire mill. As it stands today there are but three single pieces of machinery in the mill building occupying the places originally intended for them, these being the three electric cranes now in use. Considering these changes it is remarkable that such a good arrangement was found possible.

## PLANT SITES

The sites for the plants as finally decided upon permitted of an arrangement entirely adequate to meet the proposed requirements, but the following important features may be noted:

The mine plant site is contiguous to the orebodies but at a safe distance from ground to be caved; it occupies a position as regards elevation well suited to a favorable hoisting arrangement and allows of a downgrade to the concentrator of 0.3 per cent. The shortest line from the shafts to the orebodies approximately bisects them, making possible a

direct underground haulage system; the site occupies a remote corner of the property and does not lie in the general trend of the mineralized zone. The mill site has a slope favorable to the type of mill erected and is of sufficient area to allow of an economical arrangement of the various units of the plant. There is ample storage space for tailings and an excellent reservoir site available for the storage of water.

## TYPE OF CONSTRUCTION

As the life of the property was estimated to be 17 years, or longer, depending upon new developments in the treatment of lower-grade ores, it was decided to make the various structures of a semi-permanent char-

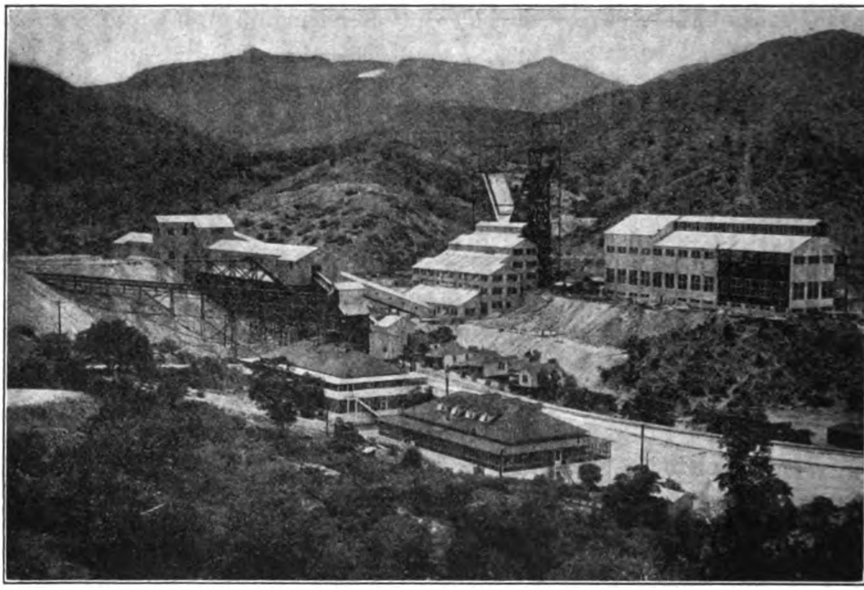


Fig. 1.—Mine Plant of Inspiration Consolidated Copper Co., Miami, Ariz. Looking West. Showing Compressor House, Main Shafts, Coarse-Crushing Plant and Storage Bins.

acter. The buildings are of steel with corrugated steel coverings, except the concentrator which has a four-ply composition roof. All windows as well as skylights are of rubber glass. The floors throughout are of concrete, the retaining walls of reinforced concrete, and machinery foundations of massive concrete with little or no reinforcing. Reinforced concrete was used wherever applicable.

## MINE PLANT

Under this general heading will be considered that portion of the plant with its adjuncts which takes the ore from the mine cars, hoists it to

surface, reduces it to a size suitable for mill treatment and places it in storage, ready for transportation to the concentrator.

The relation between the orebodies and the mine plant can be described as follows: Conceive of a chain of orebodies the lateral extensions of which form a parabola with its vertex pointing to the north and having a length of approximately 9,000 ft. Place the main haulage ways on either side of the axis, 60 ft. apart, and open them to surface through two three-compartment shafts, 102 ft. centers, and about 750 ft. from the nearest approach of the orebodies. Group the mine plant symmetrically about the axis between the shafts and the orebodies and a general idea of their relation is obtained.

## Dual Arrangement

As a result of the preliminary studies, the dual arrangement of the mine plant was evolved, the primary consideration being the necessity of handling 1,000 tons of ore per hour, which in itself precluded the use of a single shaft. Two shafts would also insure continuity of service. It was then considered advisable to follow this idea a step further and make the whole mine plant duplicate in its arrangement, which would give reasonable assurance against total shutdowns and also permit of a better load factor.

### *Underground Haulage*

Underground haulage, for transporting crude ore from the stope chutes to the loading stations at the main shafts, is confined to two levels, the 600 or main haulage level, and the 400 level. The main haulage level is, in turn, made up of two distinct systems, each serving its main shaft. All haulage is to be controlled by a block signal system.

Two types of motive power for underground haulage were considered—electric and compressed air. On account of less danger to life due to the elimination of bare conductors, the compressed-air locomotive was chosen rather than the electric. The difference in the efficiency of the two systems is probably a small percentage of the total cost. "Safety First" was therefore the deciding factor.

The locomotives are of the two-stage, four-wheeled type, and have a weight on the drivers of 10 tons. The initial cylinder air pressure is 250 lb., and the charging pressure 800 lb. per square inch. The haulage capacity per charge was estimated at 50 ton-miles. The locomotives haul 25 cars, each of 5 tons capacity, on a 30-in. gage track over a 0.4 per cent. grade in favor of the load, and will negotiate a curve of 38 ft. radius.

All ore tapped from the stope chutes is broken to pass a 12-in. grizzly so that no further attention has to be given to it after it is once in the cars. At the shafts the cars, weighing 14,000 lb. each loaded, are dumped, five at a time, by motor-operated tipples, one for each shaft on the 600

or main haulage level, with a third on the 400 level. All hoisting is done from the main haulage level.

### *Tipples*

The tipples have an overall length of 56 ft., and make a complete revolution in 15 sec. The driving shaft is connected through suitable gearing to a 35-hp. motor which runs continuously in one direction, the starting and stopping of the tipple being accomplished by means of a friction clutch located on the intermediate shaft. Each tipple is provided with an automatic stop which brings it into the proper position for running on and off the cars, this stop being released by means of a foot lever on the operator's platform. The two main tipples are operated from a centrally located platform.

### *Underground Pockets*

The tipples on the 600 level dump directly into two main underground pockets (Fig. 2), which serve as storage for the four automatic measuring and loading devices that are placed beneath them. The capacity of each pocket is 1,600 tons, or 800 tons for each loader, which, with full pockets, will run the crushing plant about 3 hr. They are of reinforced-concrete construction throughout and are lined with 2-in. planks. The upper pocket has a capacity of 500 tons and is connected to the lower pockets by an inclined chute.

### *Automatic Measuring and Loading Devices*

To carry out the automatic feature of the hoisting equipment, it was desirable to make the loading of the skips automatic. There being nothing on the market which could be used for this purpose, it was necessary to work up complete designs in accordance with original ideas.

In a few words the device (illustrated in Figs. 3*a* and 3*b*) can be described as follows:

Ore from one of the 1,600-ton underground pockets rests on a roll feeder which is actuated by a pawl and ratchet wheel, and is fed into a 12-ton measuring hopper. This hopper is suspended between the legs of a U-shaped scale beam with fulcrums at the two ends. Through a link at the middle of the yoke the proportional weight of the hopper and ore is transferred to the main scale beam which carries a counterweight of 1,370 lb. Motion obtained from the counterweight scale beam when the hopper becomes filled is utilized to operate a system of levers and lift the pawl from the ratchet wheel, thus stopping the roll feeders. The hopper is now ready to be emptied.

As the returning empty skip settles on the chairs, it strikes an arm

projecting out into the shaft. This arm is connected by a system of links to the shaft on which the yoke of the gate forming the bottom of the hopper is hinged. The motion imparted to the system by the skip trips the gate yoke, the weight of the ore in the hopper causes the gate to drop

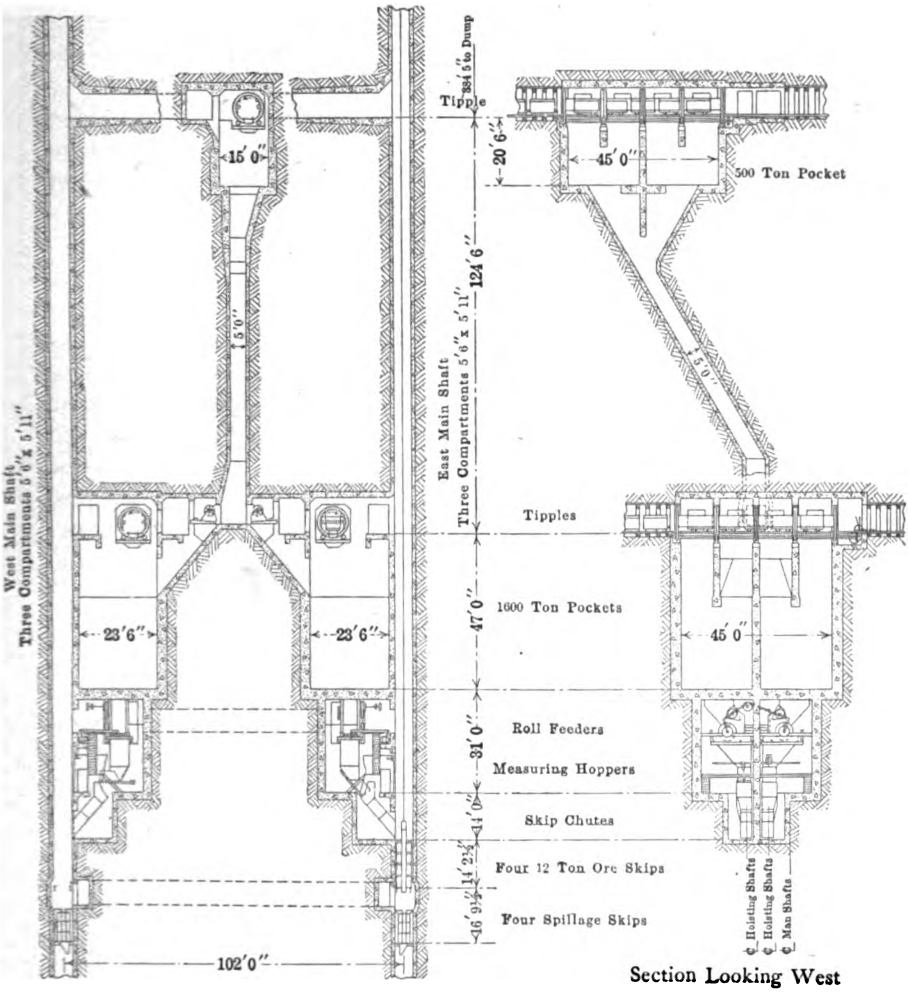
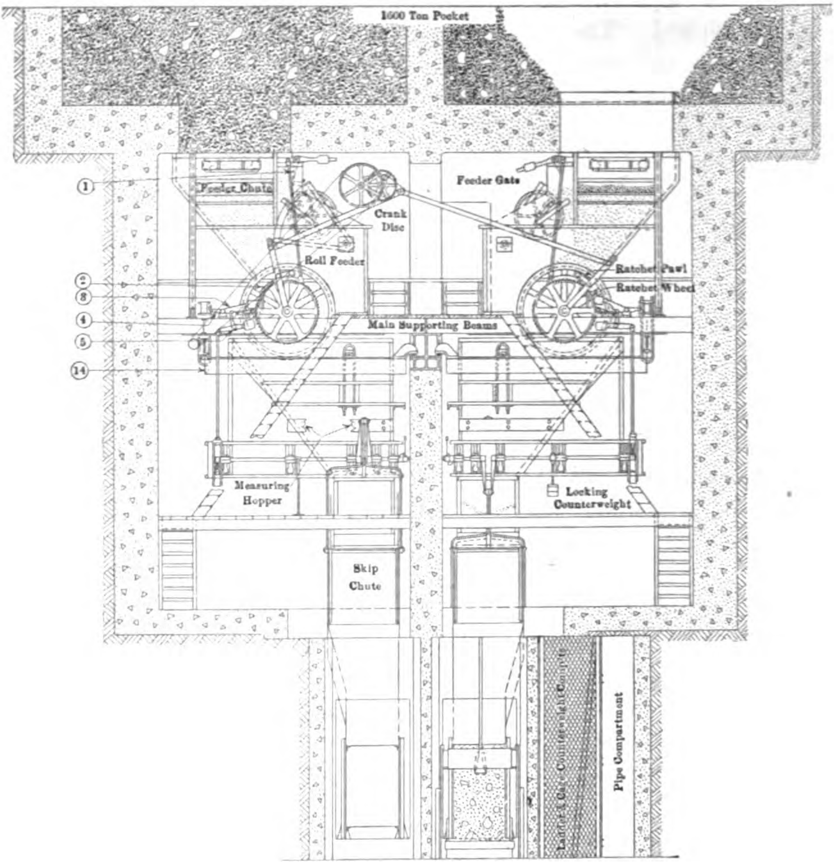



FIG. 2.—ARRANGEMENT OF UNDERGROUND POCKETS.

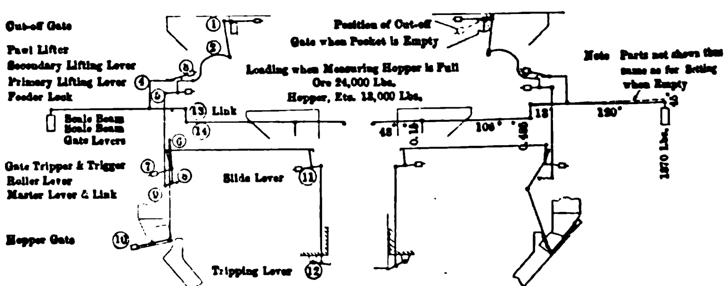
into the skip chute, and the hopper is emptied into the skip. As soon as it is empty, a counterweight on the gate causes it to close, all motions are reversed and the measuring hopper is again filled.

A cut-off gate prevents the roll feeder from becoming bare in case the pocket should be emptied. A 5-ft. layer of ore thus protects the feeder




Measuring Hopper Full    Measuring Hopper being Emptied

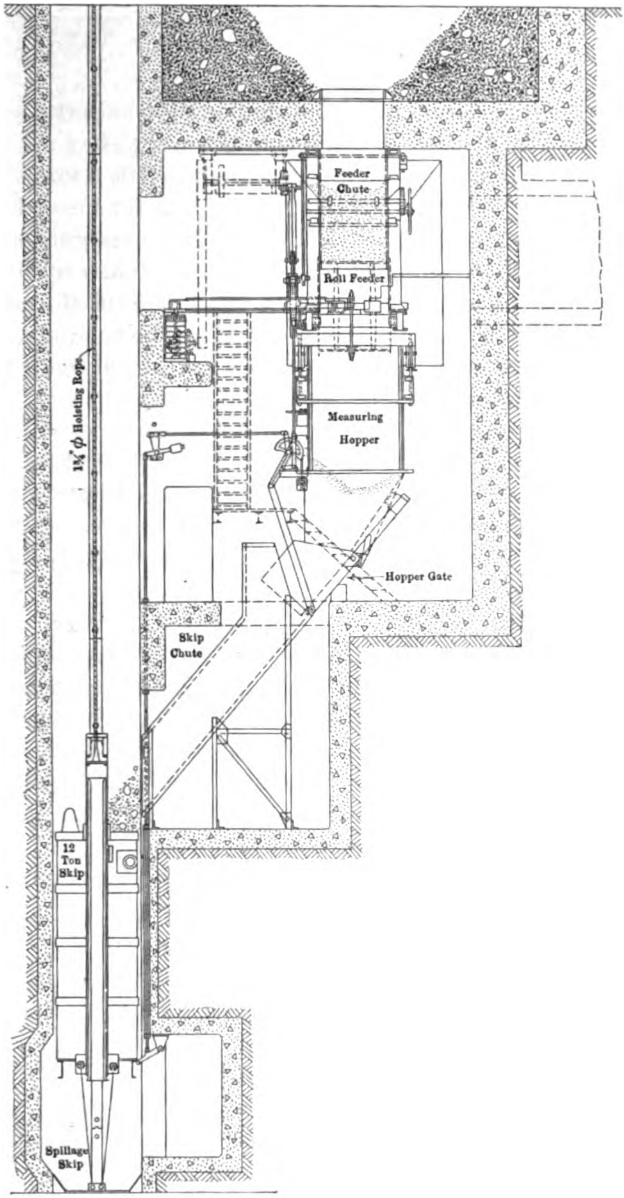
Sectional Front Elevation




Setting when Empty or being Filled    Setting when Full ----- when being Emptied ——

Line Diagram showing Principle of Operation

FIG. 3a.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.




**Sectional Side Elevation**
**Measuring Hopper being Emptied Feed Cut-off**

**Fig. 3b.—Automatic Ore Measuring and Loading Devices.**

and does away with excessive impacts due to the dumping of the ore from the tipple 50 ft. above. A 5-hp. motor furnishes the required power for the roll feeders of each pair of loaders.

### *Shafts*

Taken individually the shafts do not deviate materially from usual designs, being of reinforced concrete throughout. They are of the same size, each being made up of three compartments 5 ft. 6 in. by 5 ft. 11 in., the two south compartments of either shaft being for the ore skips. The north compartment in the West Main serves the man elevator and the corresponding compartment in the East Main is taken up by pipe lines, ladderways and the elevator counterweight. All platforms, supports, and ladders in this compartment are of steel. The conduits carrying the power and lighting circuits underground are in the end walls, being brought to junction boxes every 100 ft.

### *Ore and Spillage Skips*

The ore skips are built according to special designs. They have a nominal capacity of 12 tons and weigh 17,000 lb. each. The overall length is 21 ft., the skip proper being 14 ft. deep. The body is of 3/8-in. steel plate with liners to take up concentrated wear.

In order to keep the sumps at the bottom of the shafts clean and to facilitate the handling of ore which does find its way below the skips, spillage skips have been installed (Fig. 2). These rest on chairs 17 ft. below the ore-skip chairs and by means of hinged aprons completely close off the shaft. When one is filled it is attached to the bottom of the ore skip and hoisted to the top of the underground pocket, where by opening a hopper bottom it is discharged. They have a capacity of 9 tons each.

### *Automatic Hoists*

At the time it was decided to sink two shafts the conditions seemed to favor the use of two entirely independent hoists, located symmetrically on either side of the main headframes. It was next proposed to put the two hoists under the same roof but to operate them independently of each other. Then on account of the advent of several new conditions it was deemed advisable to make the operation of the hoists automatic.

While the hoists are entirely automatic in their operation, either can be manually operated independently of the other if so desired. No attempt will be made to cover the automatic features of the installation as these are presented in detail in another paper.[1] Suffice it to say here

[1] H. Kenyon Burch and M. A. Whiting: Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co., *Bulletin* No. 111, pp. 583 to 596. (March, 1916.)

that the problem as presented has been solved in a most efficient manner, sufficient proof being the entire satisfaction that the equipment has thus far given.

*Man Elevator*

When the question of handling men came up, several possibilities as regards point of distribution presented themselves. They could be handled through an existing tunnel, hoisting or lowering to the working levels through an inclined shaft, or entirely through an inclined shaft, or through one of the main shafts. Whichever scheme proved the most feasible, the idea was to have the men do as little walking as possible, ample provision to be made for transferring them from change house to working places. It was finally decided that distribution from the main shafts was the most logical and the shafts were equipped accordingly.

The next consideration was the type of hoists to be used. Standard mine practice favored a separate hoist, manually operated and working against a counterweight. This type made necessary the services of two attendants, one on the cage and one at the hoist, but in working out probable operating costs the idea was conceived of eliminating the hoist attendant by an application of the elevator principle. Studies along this line were at once begun, but no great amount of enthusiasm among the manufacturers could be aroused on account of the unusual conditions. Equipment was wanted to handle a counterweighted, double-deck cage, weighing 7,100 lb. and loads of 7,500 lb., the maximum speed of the cage to be 800 ft. per minute. The fact that the hoist and counterweight would have to be removed from the elevator shaft also presented a new feature in the design. It may be said, however, that placing the hoist in the headframes directly above the shaft, as is the usual practice in office building design, was at one time considered.

The equipment as installed is operated entirely from the cage, the hoist, motor-generator set, and control apparatus being in the hoist house about 220 ft. away. The motor driving the direct-current generator on the motor-generator set has a rating of 190 hp., the generator developing 130 kw. The hoist is driven by a 158-hp. direct-current motor. It is provided with a safety brake, so arranged that when the hoist is stopped the brake is automatically applied to hold the cage. This brake is actuated by spring pressure, is constantly in service except when electrically released during normal operation of the hoist and is, therefore, instantly applied in case the current supply is interrupted from any cause. Additional safety devices are provided which take care of overwind, overload on motor, slack cable, etc. Two oil-cushioned buffers are provided at the bottom of the shaft, being so designed as to bring the loaded cage to a gradual stop in case the cage from any cause, while running at normal speed, should not stop at the lower terminal. The cage is equipped with

a telephone which keeps the attendant in touch with the hoist house, also with an annunciator system covering the several levels.

Each level and the collar are provided with two-story stations that make it possible to load both decks without shifting. The cage accommodates 36 to 40 men besides the attendant. The surface station, which is in the headframe, is connected with the change house by a covered runway. From the underground stations the men are transferred to the various working places in cars accommodating 12 men each and drawn by the haulage locomotives.

The flow-sheet in Fig. 7 should be referred to in connection with the following description of the surface plant of the Inspiration company.

## Crushing Plant

The loaded ore skips assume a dumping position at a point about midway between the shaft collars and the center of the sheaves, leaving about 25 ft. for possible overwinding. The sheaves are 125 ft. above the shaft collars. They are 12 ft. in diameter and are pressed on 8½-in. shafts; 8-ft. sheaves on 6-in. shafts are used for the man-elevator ropes. The dumping tracks are of usual design.

### *Crude-ore Bin*

By means of chutes leading away from one compartment of each shaft, an even distribution of the ore is effected throughout the length of the 2,000-ton crude-ore bin. This bin is of rather unusual design and very nearly self-cleaning. The bottom slopes down 45° each way from a hip center, toward the vertical back and front, a section resembling an inverted capital M. Gates are placed under the bin on the slope toward the back as well as on the front of the bin. This arrangement of bottom and gates not only allows the bin to be emptied but the ore is prevented from packing or hanging up by drawing off alternately from the rear and front gates. The gates are 48 in. wide and are operated by rack and pinion from runways.

For each of the four units of the crushing plant two gates, one under the bin and one in front, discharge on an apron feeder that travels across and under the bin, delivering the ore upon a 3-in. bar grizzly feeding a No. 8 gyratory crusher set to crush to 4-in. cubes (Fig. 4).

The apron feeders have a width of 48 in., a length of 25 ft. 6 in. between centers and travel at 7 ft. per minute; 20-in. belt conveyors traveling at 7 ft. per minute and driven from the feeder sprockets are placed beneath the feeders to take care of drippings. These discharge into chutes that join the undersize from the grizzlies. Wood scraps are removed from the ore at the discharge end of the apron feeders and also

from the inclined conveyors leading to the disk crushers. These scraps are carried away by two 20-in. belt conveyors traveling at 75 ft. per minute, running across the building and dumping outside.

One of the four similar crushing units will be described in the following paragraph. Fig. 4 shows a cross-section of the crushing plant.

## *Description of Crushing Unit*

The discharge from the gyratory and the undersize from the grizzly are conveyed by a 30-in. inclined belt conveyor to two 48-in. disk crushers,

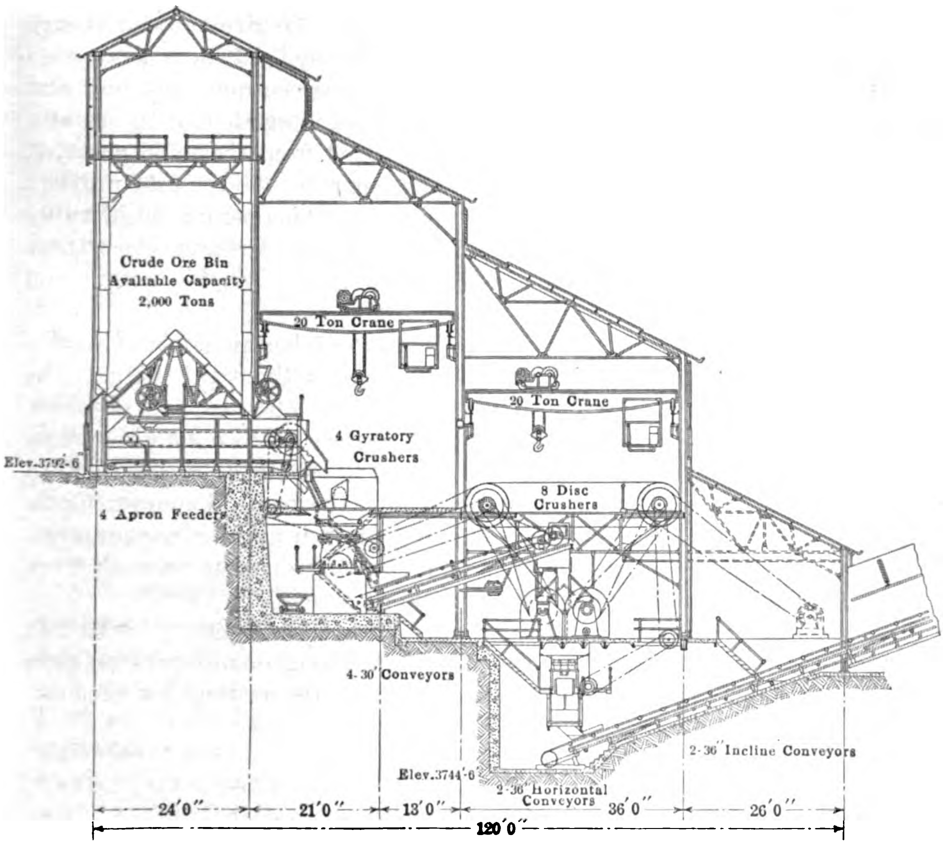



FIG. 4—SECTION OF COARSE-CRUSHING PLANT.

the ore first passing over a 1½-in. bar grizzly. This conveyor has a magnetic head pulley for removing tramp steel. The 48-in. disk crushers are set with a maximum opening of 2 in., their discharge and the undersize from the 1½-in. grizzlies joining on two 36-in. horizontal cross belts which in turn discharge at the center of the building on the two main

inclined belts, also 36 in. in width. These belts are 300 ft. long, travel at 350 ft. per minute and are equipped with hand-adjusted tail-pulley take-ups and also with weighted tension carriages. They discharge on four 24-in. horizontal conveyors over the storage bin, each of which has an automatic tripper which distributes the ore uniformly throughout the bin. Weightometers are placed on the two main inclined belts, thus giving a record of the ore handled.

## Storage Bin

The purpose of the storage bin in connection with a mine plant is identical with that of a flywheel on an engine: "To give out and absorb energy when variation in the load occurs suddenly." It does away with the necessity of maintaining a nice balance between mine and mill and allows either, for a short time, to cease running altogether, or to run at a lessened capacity. Where the mill is removed from the mine plant, it also furnishes an economical means of transferring the ore into railroad cars. Besides this it allows of a hoisting and crushing period considerably less than full-time running, a very important consideration, since there are always repairs to be made in shafts, to loaders, tipples, hoists, and crushing machinery.

The Inspiration storage bin is double tracked, has an available capacity of 25,000 tons and will load a train of 14 cars without switching. The base is of reinforced concrete, and is provided with four expansion joints. The bin proper and conveyor housing are of steel. The main dimensions of the bin are as follows: Overall length 465 ft.; width 40 ft.; depth 40 ft. The bin is divided into three compartments, the two end compartments being for the storage of special ores if any such ever be encountered. They are comparatively small, accommodating one car on each track and having an available capacity of 1,700 tons each.

The distributing belts are driven by two 50-hp. motors located at either end of the bin. The ore is drawn off through hand-operated gates of the swinging cut-off type, there being three gate openings for each car.

## Sampling Plant

The heads sampler is placed at the point where the main inclined belts discharge on the horizontal cross belts (Fig. 7). This sampler consists of a rectangular cast-iron bucket, 9 by 30 in., on the end of a revolving arm 10 ft. in length. The bucket runs on a circular track 12 ft. 4 in. in diameter and is driven through a wormwheel mounted on a vertical shaft, making a revolution every 40 sec. The bucket has a swinging cut-off type of gate for a bottom. In its travel it cuts the two main ore streams, taking out $\frac{1}{20}$, and at a point 180° from the ore stream strikes a tripper

which causes the gate to open and the sample to be discharged into a 10-ton hopper. From the hopper the sample is fed by a roll feeder and started on its way through the sampling plant, which is located against the center of the storage bin opposite the main conveyors. Here the sample is first passed through a 24-in. disk crusher. A $\frac{1}{5}$ cut is then made with a Snyder sampler and this cut is delivered to a set of 27 by 14-in. rolls. The roll product is cut by a second Snyder sampler, this $\frac{1}{5}$ cut going to a mixing drum that equalizes the flow. The resulting stream is delivered to a duplex Vezin sampler making two $\frac{1}{10}$ cuts. These samples are duplicates and weigh 110 lb. each, representing a day's run. The rejects from the three samplers go to an elevator that discharges into the storage bin.

## Motors and Other Equipment

The four units of the crushing plant are driven by four 200-hp. motors located in two dustproof rooms, symmetrically arranged on either side of the plant. Except for some minor reductions, belt transmission is used throughout.

Two 20-ton electric cranes are installed in the plant, one to serve the gyratories, the other the disk crushers, motors, etc. These are in turn connected to the yard track by transfer cars and a 20-ton electric trolley hoist, running at right angles outside of the building. Each 200-hp. motor is served by a 6-ton crawl so arranged as to allow of crane handling after leaving the motor room. A repair car and track near the gyratories permits of easy handling of their driving mechanisms, or bottom plates.

## Compressor Equipment

Compressed air is used for two distinct purposes: For underground haulage, and for mine drills. For the first, 1,000-lb. air is used and for the second, 100-lb. The haulage system is served by two compressors, each having a capacity of 1,125 cu. ft. of free air per minute at 107 r.p.m. These are of the duplex, four-stage, power-driven type, the cylinders being cross-compounded, water-jacketed, and having intercoolers between the stages. These compressors have an automatic regulating arrangement whereby the 1,000-lb. discharge line is cut out when a predetermined high-pressure point has been reached, and the four cylinders converted into a double-compound compressor, discharging directly into the 100-lb. main. Each compressor is driven by a 430-hp. self-starting, synchronous motor.

The low-pressure equipment consists of two 100-lb. compressors having capacities of 3,000 and 7,270 cu. ft. of free air per minute. Both compressors are of the parallel, two-stage type, all cylinders being water-jacketed with intercoolers between the two stages. The 7,270-ft. machine

is equipped throughout with Hoerbiger-Roegler plate valves, which on account of the very short movement and exceedingly light construction permit of a high compression efficiency. The cylinders of this machine are 46-in. and 28 by 36-in. stroke. It is driven by a self-starting synchronous motor of 1,150 hp. The 3,000-ft. machine is driven by a 500-hp. self-starting, synchronous motor at 107 r.p.m.

Two 100-kw., 125-volt, motor-generator sets arranged to operate in parallel serve as exciters to the compressor motors. Cooling water for the jackets and intercoolers is obtained from a circulating system consisting of a duplicate installation of 750-gal. triplex pumps and a spray cooling pond.

The compressors and hoisting equipment are housed in a spacious and well-lighted building which has a 25-ton electric crane running its entire length connecting at one end with a well into which railroad cars can be run and unloaded. A 22-panel, remote-control switchboard, together with the low-tension (2,200-volt) bus equipment, occupy a room along one side of the building, the high-tension transformer station being outside near one end of the building. Only two attendants per shift are required, one being an oiler. Without the automatic features for hoisting ore and the elevator for handling men, at least four would be required.

## CONCENTRATOR

The considerations which led to the type of plant finally decided upon will first be given; this will be followed by a description of the plant. The concentrator includes the concentrates filter plant. Fig. 5 shows the concentrator with its adjuncts.

### 600-Ton Test Mill

The underlying principle that pervaded the work in this mill was to carry on all experiments with the idea constantly in mind that the particular method or machine under test might eventually be installed in the new mill.

The test mill was put up primarily to try out flotation on a large scale in order to determine its proper application to Inspiration ores. Since the flotation process, as applied to the concentration of copper ores, was an innovation, it was logical to suppose that in its last analysis the flow-sheet might differ altogether from wet concentration methods. This did not prove to be the case, but the assumption was made and it was for this reason that the test mill was made so flexible.

The test mill was started February, 1914. The concentrates sold have paid the expenses of erection and operation, a remarkable record for a test plant. It has treated from 600 to 1,000 tons per day, the amount depending upon the types of machines being tested.

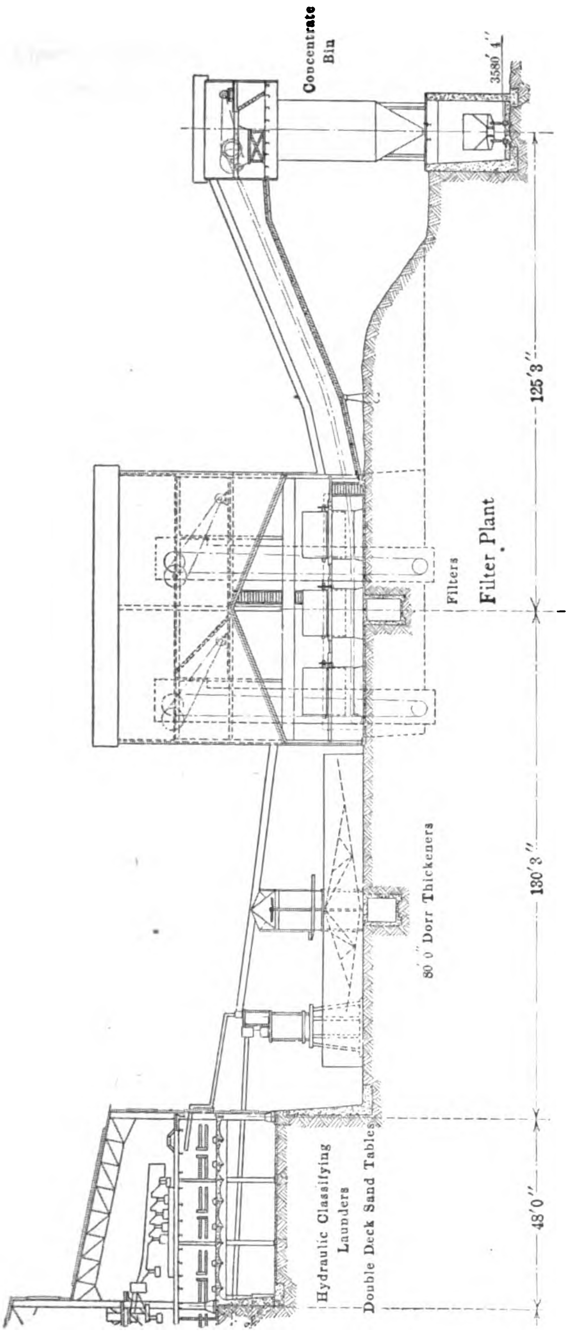



FIG. 5.—SECTION OF CONCENTRATOR AND FILTER PLANT.

The building is of timber, covered with corrugated steel. This type of building was chosen because it would facilitate changes or additions, was the cheapest and quickest form of construction, and would still have considerable value when dismantled. All machinery was placed on concrete foundations.

The ore for the test mill was taken from a development shaft and reduced to 3-in. cubes in a No. 8 gyratory crusher located nearby, the crushed ore being loaded into cars which emptied into a 180-ton bin at the head of the test mill. The following description gives the way in which the mill was first run, the various additions and changes being noted in sequence.

Two 30-in. apron feeders delivered ore from the bin to an inclined 20-in. belt conveyor, which was equipped with a recording weighing machine and a magnetic head pulley. This conveyor delivered the ore to a 36-in. Symons horizontal disk crusher and a Symons 48-in. fine reduction disk, or vertical disk crusher. These two machines delivered to a second 20-in. inclined conveyor. If so desired the two crushers could be bypassed, the delivery then being made direct to the second conveyor. This permitted the crushers below the second conveyor to be tried either as primary or secondary machines. The second conveyor delivered the ore to another 48-in. fine reduction disk and a Symons 48-in. roller mill. Water could be added either above or below these crushers. The ore then went to four Hardinge mills, the feed to which was regulated within suitable limits by a mechanical distributor. Three of these mills were 8 ft. in diameter with barrels 36 in., 44 in. and 72 in. long, all being direct-driven by induction motors. The fourth mill was 10 ft. in diameter, had a barrel length of 28 in. and was belt-driven. All of these machines were pebble mills with either silex or El Oro linings.

A drag classifier followed each mill, the whole floor being so arranged that each mill could be placed in a closed circuit with its drag or two mills and their drags run in tandem, that is, the first mill discharged into a drag, the sand from which was delivered into the second mill which in turn discharged into the second drag. The sand from the second drag was returned to join the sand from the first drag to form the feed for the second mill. The overflow from the drag was delivered either to eight sand tables by two distributors, or sent to other distributors which fed eight slime tables at the lower end of the mill.

Between the sand and the slime tables were placed the flotation machines which could be fed directly from the drag overflow or by any table product. The original installation consisted of two Minerals Separation eight-compartment flotation machines, one of 50-ton and the other of 600-ton capacity. By means of elevators any flotation product could be retreated on the slime tables or in the small flotation machine.

The concentrates from the tables and flotation machines went to a

filter plant containing an Oliver and a Trent filter. The dried concentrates were loaded into 1-ton cars, weighed and dumped into a railroad bin from which they were taken to the smelter.

Mechanical samplers were installed at important points and hand samples taken where necessary.

The crushing and grinding experiments were not only extensive but proved most interesting, the results being quite unexpected. The 36-in. disk crusher represented the 48-in. disk crushers now installed in the crushing plant following the gyratories.

As secondary or intermediate crushers, that is, machines between the 36-in. Symons disk and the Hardinge mills the following were tried out: The Symons 48-in. fine reduction disk; the Symons 48-in. roller mill; the Symons 56 by 48-in. ring mill; the Bradley 66-in. centrifugal roller mill; the Overstrom centrifugal crusher; the Allis-Chalmers No. 4 hammer mill; and the Marcy ball mill.

The final results indicated that the Symons fine reduction disk, with a little redesigning, would be an efficient machine for the limited field of crushing from 4-in. cubes to 4-mesh. It was found that this could be done with a single pass, no oversize resulting. The machine would not work well wet and it was found necessary to remove all fines from the feed to the machine, otherwise a packing would occur. Consumption of steel wearing parts appeared low, as also did power consumption, although no definite results as to these two points were obtained.

The Marcy mill also proved to be a most excellent intermediate crusher, handling as high as 800 tons of 3-in. feed and under to pass an 8-mesh screen without return of oversize.

To compete with the four Hardinge mills a 6 by 20-ft. Chalmers & Williams tube mill and an 8 by 5-ft. ball mill were installed. The latter was a Marcy mill using 5-in. diameter steel balls. It was placed in a closed circuit, as were the pebble mills, and gave good results. It was next tried on a feed from the 36-in. disk crusher and then on a feed direct from the gyratory. These experiments definitely demonstrated that only three reductions were necessary: A gyratory, followed by a disk crusher and ball mills in a closed circuit. This is the arrangement installed in the present plant.

Several different classifiers were tried in the ball-mill circuit, a specially designed duplex Dorr classifier being adopted.

The flotation machines and the tests conducted on them are described in another paper,[2] but it is interesting to note here that the following machines were tried out: Minerals Separation, Hoover and Hebbard types; the Callow flotation cells; the Cole-Bergman; and the Inspiration machine.

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[2] Rudolph Gahl: History of the Flotation Process at Inspiration, *Bulletin* No. 117, September, 1916.

The Inspiration machine was devised during the operation of the tes mill.

The concentrator is equipped as follows: 4 units with the Callov machines; 13 units with the Inspiration machines; and 1 unit with th Minerals Separation machines.

The drag classifier as originally tried out proved to be too small fo the tonnage required, the drag now in the concentrator having bee developed from the small one.

The tables tested were the Deister Machine Co.'s double-deck slimers double-deck sand tables, and four-deck slimer; the Deister Concentratin Co.'s single-deck slimer, double-deck sand table and No. 4 sand machine and the Wilfley No. 6 table with decks arranged for both sand and slime The Deister Machine Co.'s double-deck sand table was adopted, 19 being installed.

This sketch of the test mill does not bring out the many small thing experimented with, such as shaking screens, Caldecott cones, samplers feeders, slopes of launders and conveyors, or the necessary efforts t work out improvements on the new machines.

## The 15,000-Ton Concentrator

The ore comes from the mine plant in 14-car trains, and is dumped into the concentrator bins, which have a capacity of 12,000 tons, or 67 tons per unit (Figs. 5 and 6). These bins are of the suspension type, ar double tracked, and extend the full length of the mill, 300 ft.

The mill is made up of 18 units, similar in equipment except for the differences in the flotation department. Eighteen 30-in. apron feeders, located on the center line of the bin, feed the ore to 20-in. inclined conveyor belts traveling at 150 ft. per minute. Weightometers are installed on these belts, which record the amount of ore fed to each unit. The conveyors discharge into hoppers where a split is made, half of the feed going to the north ball mills and half to the south mills, the transfer being made through launders with cast-iron liners and having a slope of 45° (Fig. 7).

### *Grinding Section*

*Floor Arrangement.*—One of the novel features relative to the layout of this plant is the arrangement of the grinding floor. With due regard for the individual capabilities of the machines used, it is nevertheless a fact that the arrangement has been important in bringing about the excellent results obtained. The following tabulation will serve to show that the method, besides being efficient in the work it is doing, is conservative of floor space as well:

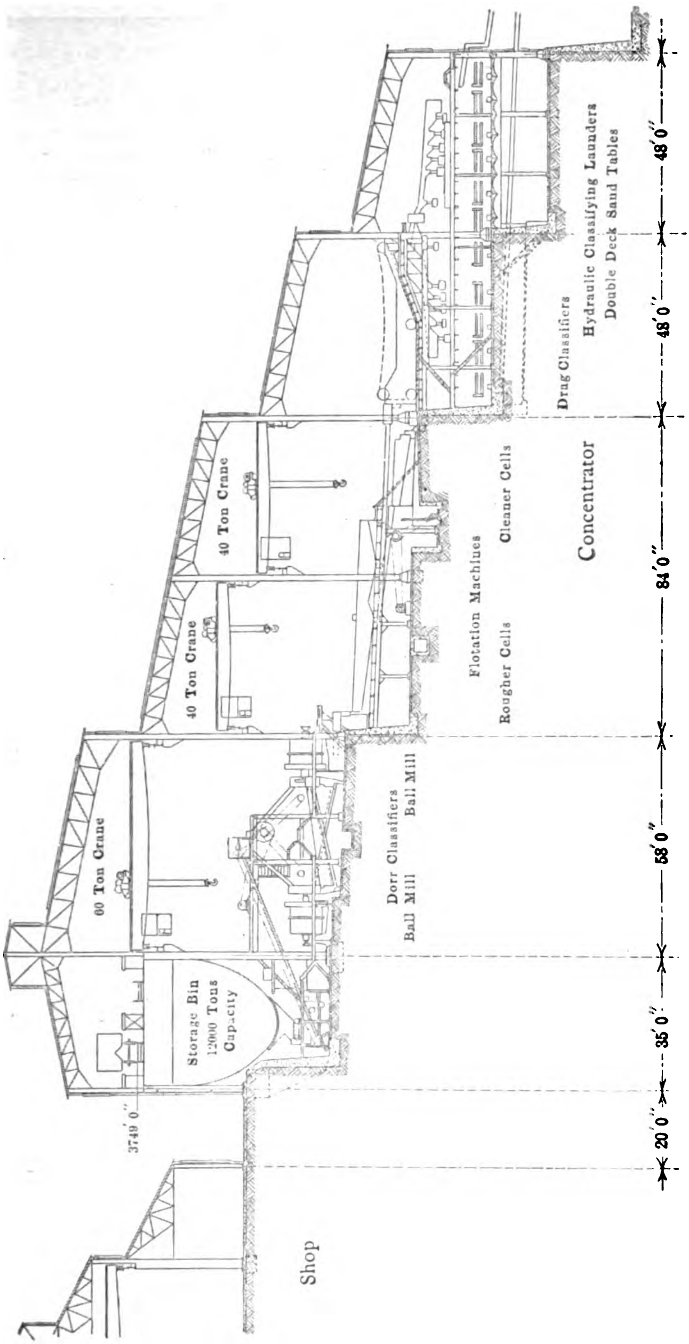


FIG. 6.—SECTION OF CONCENTRATOR.

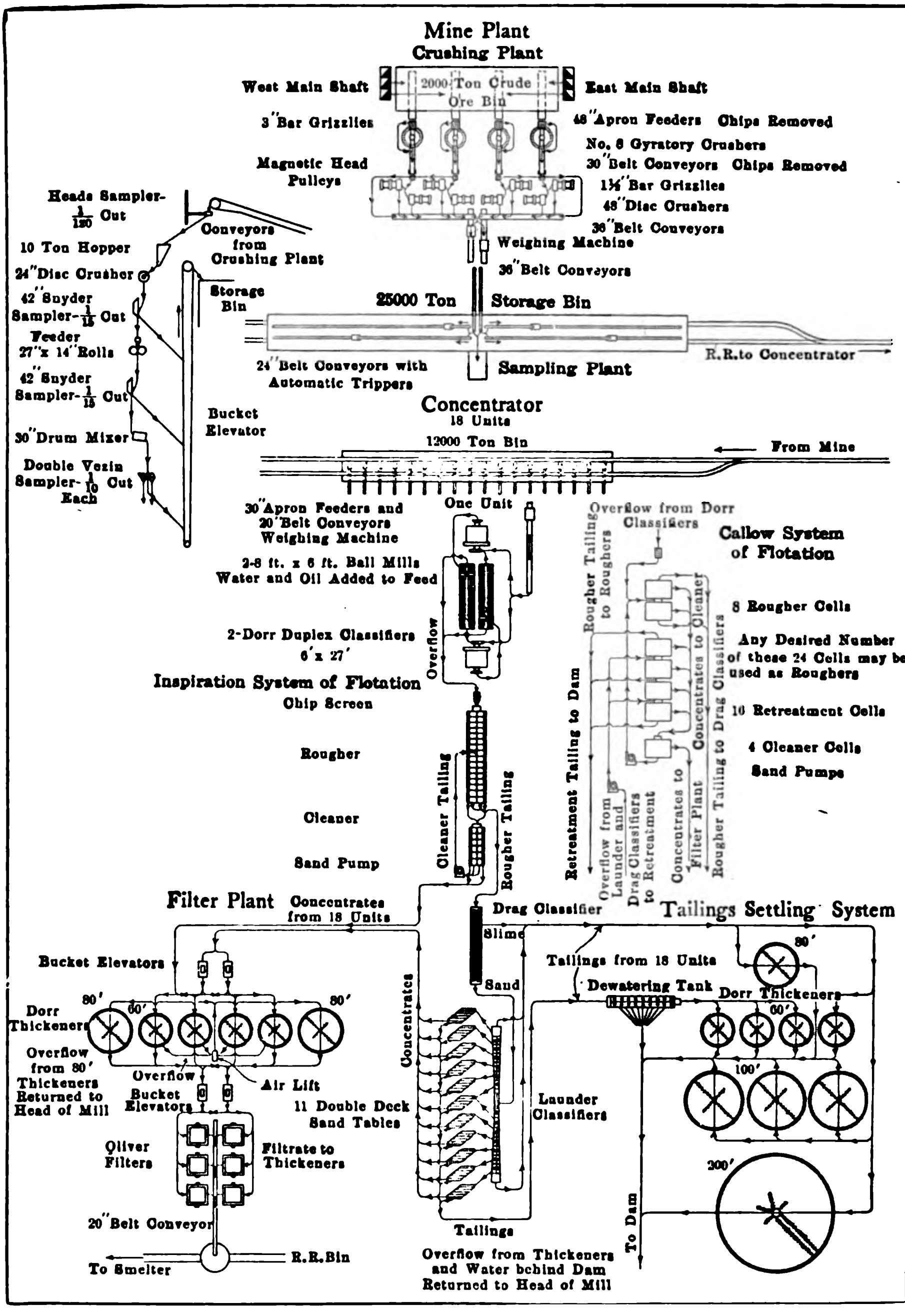


FIG 7.—FLOW SHEET OF INSPIRATION PLANT.

Capacity of installation, 15,000 tons per day from 2-in. disk crusher opening to 1½ per cent. on 48-mesh.

Area of grinding floor, exclusive of motor platforms, 300 by 66 ft., or 19,800 sq. ft.

Floor space per ton capacity, 1.32 sq. ft.

The scheme can be briefly described as a parallel-series arrangement; two circuits—each with a Marcy ball mill and a Dorr classifier in series—are arranged in parallel, the oversize from each classifier being returned to the mill opposite for regrinding. The mills are symmetrically arranged in two rows extending the length of the mill with their feed ends facing each other, just enough room being left between them to accommodate the Dorr classifiers (Figs. 10 and 11).

The Dorr classifiers act as elevators and since all return is handled by them, the whole department is confined to a single level. There is a main runway between the mills above the classifiers, from which branch runways are taken off to lead to the motors, between mills, etc. This arrangement gives the attendant an excellent opportunity to watch his machines and to reach them when necessary. The conveyors bringing the feed to the mills are terminated in hoppers on a platform high enough above the mills to allow of a 45° slope for the launders leading to the feed boxes. The motors for driving the conveyors, classifiers, etc., together with their control apparatus, are located on this platform, as are also the control panels for the ball-mill motors. The switchboard attendant thus has a commanding view of the whole grinding floor. The floor is served by a 60-ton electric traveling crane which transfers the mills to and from the repair floors located at either end of the building. An inclined skip-way along one side of the building serves the whole concentrator, the cranes on each floor making direct connection with it.

*Marcy Ball Mills.*—The feed for each ball mill consists of ore from the bin, together with the sand from the discharge of the mill opposite, the final product of the system being the overflow of the classifiers. The feed to the ball mills has a consistency of about 1 to 2, and the overflow from the classifiers about 3 to 1, the latter being maintained by an automatic device which adds water as required in the discharge boxes of the ball mills.

The ball mills are 8 ft. in diameter and 6 ft. long. They are direct-driven through herring-bone gears by 225-hp. induction motors. Our experiments with the ball mill in the test plant led us to the conclusion that we could crush 400 tons of the coarse-crushing plant feed with each mill in 24 hr. to 2 per cent. on 48-mesh screen and 60 per cent. minus 200. Now that the Inspiration mill is running with all sections, it has been shown that our test figures were correct. Not only have we been able to crush 400 tons per mill, but with 18 sections running the average daily dry tonnage for the month of May was 15,358. We have a test

section (of two mills) that showed for a short period a tonnage rate of 1,000 tons per 24 hr. While this is by no means the average, yet it indicates what may be done in the future.

*Performance of Crushing Machines.*—In view of the character of the ore and the fact that much of it is tough and more difficult to crush than the ordinary porphyry ores, we consider the power consumption for our crushing plants very satisfactory. The power consumption at the coarse-crushing plant for the month of May, taking mine-run ore, crushing it in gyratories and disk crushers, and conveying to the railroad bins, was 0.329 kw.-hr. per ton. At the mill, the average power consumed per ton during the month of May for the ball mills, including power for feeders, conveyors and classifiers, was 10.4 kw.-hr. Therefore, the total power consumed per ton for the month of May on ore taken from the mine bins, crushed and delivered to the flotation machines was 10.729. We have had one section of the ball mills in operation for a period of 10 days that has given a power consumption of 8.53 kw.-hr. per ton. If we add to this figure the power used at the coarse-crushing plant and of the conveyors, we would have 8.88 kw.-hr. per ton of mine-run feed delivered to the flotation machines. The above does not include the power required for transporting the ore by steam road from the mine to the mill.

The crushing and delivery of the ore for concentration is a matter of great importance, and one that absorbs a large portion of the operating costs in the treatment of ores. Too much attention cannot be given it. We are now conducting many experiments in connection with our crushing problem, including different types of steel for wearing parts and modifications that influence our tonnage and cost of production. We have equipped ball mills without the grates and the results of the mills so equipped have shown us that ball mills equipped with grates have advantages in increased tonnage and efficiency. Our experiments with the different shapes of pebble mills, our subsequent experiments with the ball mills, and finally our monthly operation, lead us to the conclusion that a good choice was made in the selection of our present grinding machinery.

### *Flotation Section*

The overflow from the Dorr classifiers goes to the flotation machines, of which there are three types now in operation. Generally speaking, the flotation machines make a clean concentrate which goes to the filter plant, a middling which goes to drag classifiers and a tailing which goes to waste. As the subject of flotation is to be taken up in detail in another paper, just enough will be given here to bring out the part the process plays in the general scheme. The flow-sheets for the two processes differing somewhat, each will be considered separately.

*Flow-Sheet with Callow Flotation Cells.*—A unit installation of these machines consists of a total of 28 cells which for sake of compactness are grouped into sets of four cells each, individual cells being 3 ft. 3½ in. by 10 ft. 2 in. Of this total eight are roughers, 16 are for retreatment and four are cleaners, the arrangement of the group being such that all feeds and products except two are handled by gravity. These two are elevated by sand pumps. The Callow cells, as installed in this plant, make two products, a concentrate and a tailing.

The overflow from the ball-mill classifiers first undergoes a roughing treatment in the eight roughers. Here the two products are a concentrate which goes to the four cleaners and a tailing which goes to a large, specially designed drag classifier, a detailed description of which will be given later. The concentrate from the cleaners is a finished product and goes to the filter plant. The tailing from these cells is pumped back into the rougher cells for a second passage through the system. The tailing from the rougher cells is classified in the drag into two products, a sand and a slime overflow, the sand being still further prepared for table concentration in hydraulic classifiers. The tables make two products, the concentrates joining the flotation concentrates in the filter plant, and the tailing going to waste. Further treatment of a middling product from these tables is contemplated and studies have been made along this line, but as yet nothing definite has been reached. The overflows from both the drag and the hydraulic classifiers are united and pumped back to the 16 re-treatment cells, the concentrate from which goes to the cleaners, and the tailing to waste. Throughout the mill adequate provision has been made for sampling.

*Flow-Sheet with the Inspiration Flotation Machines.*—Except for a special arrangement in one unit, the only difference between this and the Callow flow-sheet is in the flotation department.

From a metallurgical standpoint the difference here is not radical. In a few words the essential difference can be stated as follows:

The Inspiration system makes use of but two machines per unit, a rougher and a cleaner placed end to end (Fig. 9). These are each designed to allow a free passage of the feed from the upper or feed end, to the tailing or discharge end, during which passage the feed is subjected to the frothing action of several compartments. Because of the large number of compartments (16 for the rougher), it has been found that a tailing can be made, the slime of which does not require re-treatment as is the case in the Callow system. The sand contained in the rougher tailing is subjected to table concentration. The tailing from the cleaners is pumped back to the roughers for a second treatment.

Several mechanical features have been introduced which are expected to keep the cost of operation at a minimum. The arrangement is simple,

FIG. 8.—BALL MILLS AND FLOTATION MACHINES.

FIG. 9.—INSPIRATION FLOTATION MACHINES.

requires a minimum of floor space and is easily attended. Its large capacity is an important feature.

*Flotation Air Supply.*—The flotation department requires a large volume of low-pressure air. This is supplied by four single-stage centrifugal compressors occupying one end of the flotation floor. The rating of these compressors is as follows:

Capacity, 23,000 cu. ft. of inlet air per minute.

Discharge pressure, 5.75 lb. per square inch.

Revolutions per minute of impellers, 3,850.

Horsepower of driving motor, 720.

The step-up to the impellers is made through Alquist gears. One unit of the four is held in reserve.

### *Concentrates Filter Plant*

Both the flotation and table concentrates receive the same treatment. From the concentrator they are either elevated or flow by gravity into the five 60-ft. and three 80-ft. Dorr tanks, located on either side of the elevator house. Here the moisture content is reduced to about 50 per cent. The thickened product is drawn off through the bottoms into launders and conveyed through tunnels to bucket elevators, which raise it to the top of the elevator house where it is distributed to six Oliver filters (Fig. 5).

The overflow from the Dorr tanks is practically clear and flows to the return-water sump. The filters are standard Oliver filters, the drums being 11 ft. 6 in. in diameter and having a length of 12 ft. After leaving the filters the concentrates have a moisture content of about 17 per cent. The arrangement is such that a single 20-in. conveyor belt traveling at 100 ft. per minute takes the concentrates away from the filters and delivers them to a railroad bin of the tank type. Here they are loaded into hopper-bottomed steel cars of 60 tons capacity. A 350-ft. train shed protects the loaded cars from heavy rains and winds.

### *Drag Classifiers*

The drag classifier used on the flotation tailings for separating sand from slime is the result of experimental work in the test mill. It is worthy of a detailed description not alone on account of its ability as a classifier but also because of its large capacity. Except for the concentrates taken out by the flotation cells, each machine handles the entire tonnage of one unit, or 800 tons per day.

The classifier consists essentially of two 18-in. belts, running parallel to each other and so arranged that the return belts run entirely clear of the settling surface, the rough classification being carried on sufficiently

far below the surface to cause no disturbing currents. This is made possible by the use of four pulleys, one at the discharge end, two (one vertically above the other) at the feed end, and one at the center near the bottom, where the slope begins. The distance between head and tail pulleys is 39 ft. The direction of travel of the return belts is horizontal,

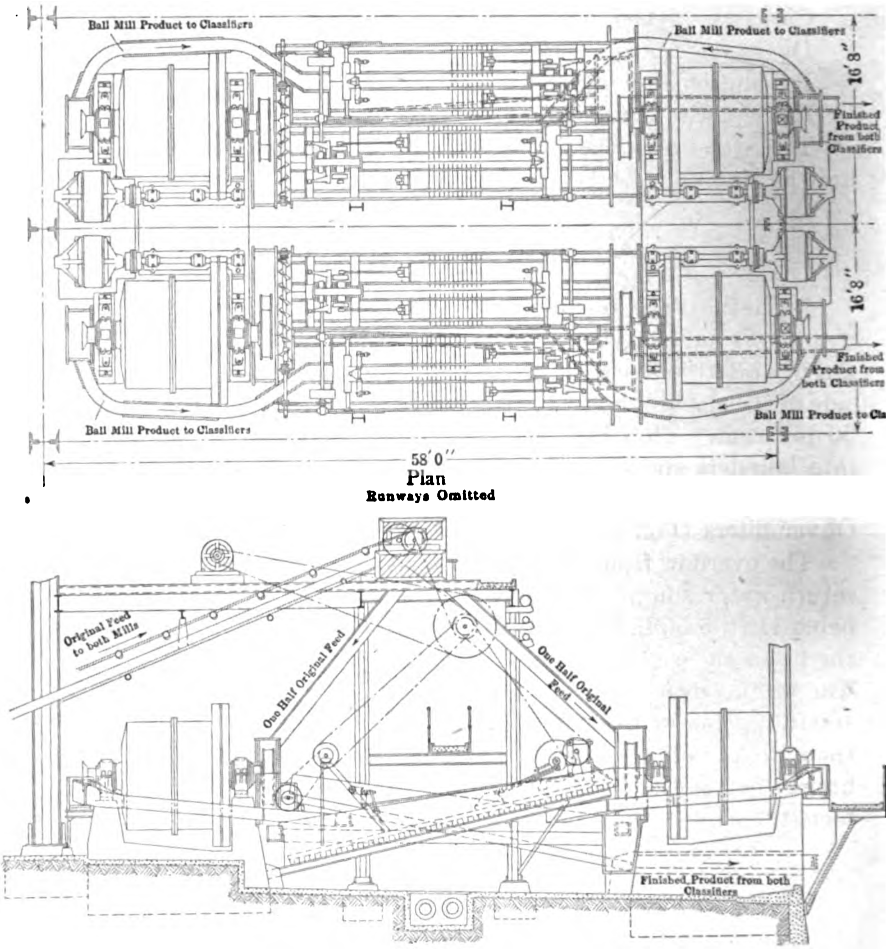


FIG. 10.—BALL MILL AND CLASSIFIER ARRANGEMENT.

but on passing over the upper tail pulley it becomes vertical. The belt thus enters the pulp perpendicular to its surface. At the bottom of the tank its direction is again changed to the horizontal by the lower tail pulley. It travels thus to the center of the tank where a break is made; the belt with its load passes under another pulley and starts up a 20° slope toward the discharge end. All pulleys are 32 in. in diameter.

Two adjustable overflow launders on either side of the tank having a combined lip length of 52 ft., carry off the slime. The effective settling area of the tank is 25 ft. by 4 ft., or 100 sq. ft. The overflow is 5 ft. above the bottom horizontal belt. The two sets of submerged pulleys are mounted on shafts running in water-tight bearings supported by the sides of the tank.

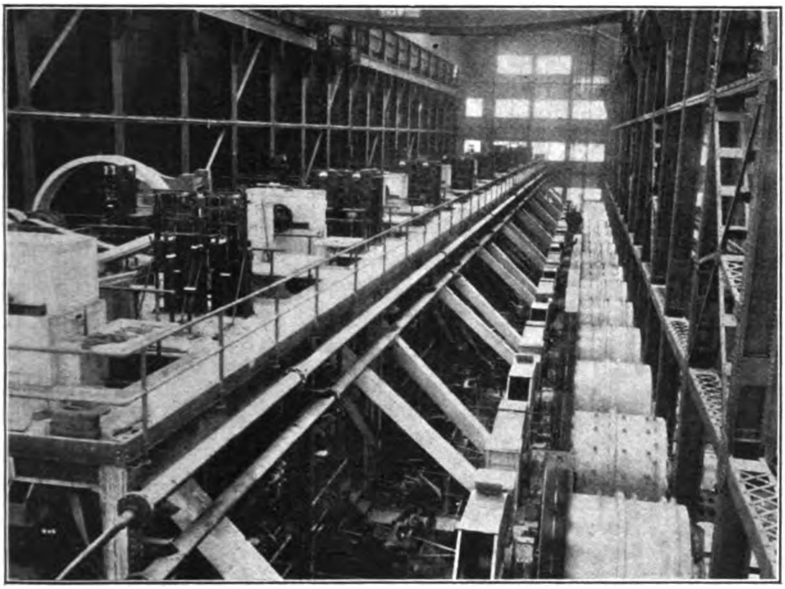

FIG. 11.—BALL MILLS AND MOTOR PLATFORMS.

*Tailing Settling System*

The conservation of the water supply is imperative, and it was for this reason that the extensive tailing-settling and return-water system was installed. The water is reclaimed at the concentrator and at the tailing dams. The equipment at the concentrator consists of three 100-ft. and one 200-ft. Dorr thickeners, and a large dewatering box 17 ft. by 109 ft. The overflow from the 80-ft. concentrate tanks is also turned into the return system. The dewatering box handles only the table tailing, the overflow from it being further settled in three of the 60-ft. Dorr thickeners. The overflow from all tanks goes to a 210,000-gal. concrete sump from which it is pumped to a 60-ft. diameter steel tank of like capacity on the shop level above the concentrator.

The return-water pump equipment consists of four vertical triplex pumps direct-driven through gearing by 100-hp. synchronous motors. These pumps each have a capacity of 2,000 gal. per minute and work against a head of 113 ft.

The water reclaimed from behind the dam (Fig. 12) is at present being handled by a 3,000-gal. two-stage centrifugal pump, so arranged that it can be moved up a skidway as the water level of the pond behind the dam rises.

200-*Ft. Dorr Thickener*.—The 200-ft. Dorr thickener is larger than any single unit ever installed and is, therefore, interesting from a mechanical standpoint. The thickener has not yet been erected but at this time the designs are complete. It consists essentially of a reinforced-concrete tank 200 ft. in diameter with the bottom sloping toward the center and having a depth at the outside of 7 ft. 3 in. The feed is delivered at the center of the tank through a launder supported by a 107-ft. steel truss. At the center of the tank is a steel plate pivot, 18 ft. high, upon which is

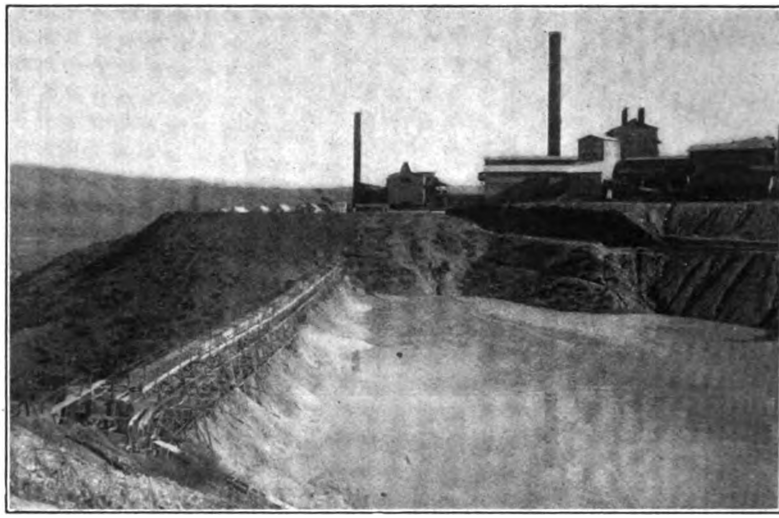

FIG. 12.—TAILINGS DAM.

mounted the drive drum. To this drum are fastened four short rakes (two 24 ft., and two 35 ft.) and the driving truss, which in turn carries two long rakes, 10 ft. apart and 100 ft. in length. The slope of the bottom of the tank varies, it being 2$\frac{3}{16}$ in. per foot near the center, 1$\frac{3}{4}$ in. per foot for the short rakes and the inner 16 ft. of the long rakes, and the remainder 1 in. per foot. The driving truss is completely submerged except for the outer end, which rests on a circular track on the tank wall. The driving mechanism is placed at the outer end and consists of a 5-hp. motor which is connected through gearing to a single 24-in. plain-tread driver. Two similar wheels are placed 10 ft. on either side of the driver to distribute the weight. The weight of the driving truss provides the necessary traction. The thickened product is drawn off through an

annular opening at the center, and the overflow at two diametrically opposite points on the periphery of the tank.

## GENERAL EQUIPMENT AND SUPPLIES

### Water Supply

Various possibilities were presented for supplying water to this plant. To insure a supply, sufficient ground and water rights were early purchased at Wheatfields, about 12 miles from the mill site and 1,000 ft. lower in elevation. Before developing this supply, however, it was decided to prospect on the flat below the tailing storage site. Wells were sunk at various points until the best location had been determined, which is 2½ miles from the mill, 430 ft. lower and at the junction of two fair-sized drainage channels, receiving their supply from the Pinal Mountains about 10 miles away and 4,000 ft. higher. Here six wells have been sunk and 24-in. multi-stage turbine well pumps installed. These deliver through wooden pipe lines to a common steel sump tank having a capacity of 235,000 gal. Each well pump is belt-driven by a 150-hp. vertical motor and delivers a maximum of 1,200 gal. per minute. Near the sump tank is located the pumping station which contains six 1,200-gal. pumps delivering into a common 20-in. pipe line. These pumps are horizontal, duplex, double-acting and are direct-driven through herringbone gears by 300-hp. synchronous motors taking current at 6,600 volts. Power for the pumping station and wells is supplied by the Inspiration-International power house. A 10-ton crane serves all parts of the building.

The 20-in. pipe line is 14,600 ft. long with a rise of 520 ft., and delivers water to the storage reservoir near the concentrator. From this reservoir water is delivered to all parts of the property. It is located about 80 ft. above the concentrator, 1,200 ft. away and has a capacity of 3,000,000 gal. An oval excavation about 20 ft. deep was made in the top of a hill and the sloping sides and bottom lined with concrete. Two 14-in. pipe lines deliver the water to the concentrator.

### Power Supply

Electric power is obtained from two sources. The Reclamation Service of the United States Government, from its hydro-electric plant at the Roosevelt Dam, 40 miles away, furnishes energy at 40,000 volts, 25 cycles, three-phase.

In conjunction with the International Smelting Co., the Inspiration company built a power house that utilizes the waste heat from the reverberatories under one set of boilers. Another set of boilers, oil-fired, is also used when the power demand is greater than can be supplied by the waste-heat boilers alone. This power house contains three 7,500-kva.

steam-turbine-driven generators, delivering energy at 6,600 volts, 25 cycles, three-phase; also three 15-lb. reciprocating blowing engines, each having a capacity of 15,000 cu. ft. of free air per minute. The air is for the smelter use exclusively, while the electric energy is for the mining, smelting and ore-dressing plants.

The distributing system is so laid out that any division of the plant can take power from either source. When available, the Reclamation power is used for the greater part of the Inspiration load. Each unit of the plant has its transformer station for stepping down the transmission line voltage.

Motors of 50 hp. or greater are run on 2,200 volts, smaller motors on 440 volts, and lighting circuits on 110 volts. Synchronous motors are used as required to maintain a satisfactory power factor.

## Transformer Stations

There are two separate outdoor transformer stations, one at the concentrator and one at the mine plant. These stations are of rather unusual design, being skeleton-steel structures. The supports are of pipe poles and the trusses between which the busses are stretched are angle lattice construction.

As the power from the two sources is received at different voltages, each station requires two sets of transformers. The Reclamation energy is delivered at 40,000 volts and the power house at 6,600 volts, both being stepped down to 2,200 volts for distribution about the plant.

Both high-tension transmission lines are in duplicate circuits. The incoming lines pass over electrolytic lightning arresters to electrically operated remote-controlled circuit-breakers, then through electrically operated remote-controlled automatic circuit-breakers to the transformers. Any one transformer may be cut in or out as desired, or all at one station can work in parallel. At the concentrator there are eight outdoor-type, 2,000-kva., oil-insulated, water-cooled, three-phase transformers, four on the 40,000-volt line and four on the 6,600-volt line. At the mine there are four similar transformers, two for each incoming line.

The 2,200-volt circuits are led from the transformers to the distributing stations, the one at the mine being located in the compressor and hoist house and the one at the concentrator in a separate building. Each distributing station consists of a concrete cell structure, carrying two sets of 2,200-volt busbars, one for each power line. This cell structure also carries the disconnecting switches, meter transformers and oil switches as required by the remote-control feature of the switchboard. In front of the cells is the switchboard with separate panels for the control of each incoming high-tension line, each set of transformers and each outgoing 2,200-volt circuit. Wattmeters, frequency meters, and power-

factor meters, all graphic, and integrating watt-hour meters are installed for each power supply, and ammeters, watt-hour meters and graphic wattmeters for each outgoing circuit. Two-throw oil switches, controlled from the switchboard, make it possible to run any circuit on either power, or to disconnect it from both.

The transformer and distributing stations, besides presenting an attractive appearance, are unusually complete, convenient for the attendant and provide the greatest possible safety.

## Shops and Railroad Facilities

All shop work, except that for the mine, is done at the concentrator shops where a complete equipment is installed. The equipment comprises about 30 different machine tools, all with individual motor drives, a 40-ton electric crane, traveling the entire length of the building, the necessary facilities for locomotive repairs, several welding outfits for various classes of work and a forge shop equipped with both oil and coal forges. The warehouse and electric shop and the locomotive roundhouse occupy opposite ends of the shop building. A standard-gage track through the center of the building permits cars to be run either into the shop or through the shops into the warehouse.

The shop level is 22 ft. above the upper floor of the concentrator. Communication between the two buildings is provided by the inclined skipway which occupies one end of the concentrator. This connects with the shops by a well through which material can be transferred either into the shops from the mill for repairs, or into the mill from railroad cars.

The railroad facilities comprise the main line and several spurs, the total length being about 10 miles. All equipment is standard gage. The lines connect the several units of the surface plant, except the pumping plant and the Live Oak Division. These are reached via the Arizona Eastern. On account of the topographical conditions, it was necessary to use switchbacks to make the grade between the Arizona Eastern on the flat and the concentrator, about 400 ft. higher. The switchbacks provide for handling six cars. The extremes of grade and curvature on this line are 4 per cent. and 12° respectively. The line which connects the mine and the mill has a grade of 0.3 per cent. in favor of the load.

The six-wheel, side-tank type of locomotive is used. Two are used on the mine line. These have a weight on the drivers when loaded of 112,000 lb. A third locomotive is used on the low line, which has a corresponding weight of 200,000 lb.; the latter being equipped with a superheater and Walschaerts' valve gears, oil burners being used on all. The ore cars are of the Ingoldsby patent-dump type, have a capacity of 60 tons and a train length of 33 ft. They are of all-steel construction and weigh about 22 tons each.

Electric haulage on the mine division was at one time considered, but for sake of uniformity on the several lines the idea was abandoned.

## CONCLUSION

The salient features of the mine plant can be summarized as follows: The dual arrangement of the entire plant; the automatic loading and hoisting of ore; the application of the elevator principle for handling men; and the reduction of the crude ore for mill treatment in two simple stages.

The final product of the plant is ore in storage ready for mill treatment. It carries roughly 1¾ per cent. copper, mainly sulphides, and has been crushed to pass a 2-in. opening of the disk crushers. The average moisture content is 2½ per cent.

To the mine summary can be added the following covering the remainder of the plant: A present mill recovery on sulphides of 90 per cent., or 85 per cent. on the total content; finished grinding in one machine; and flotation followed by table concentration.

Although the Inspiration plant is not quite the largest, it does embody what is today the latest practice in the treatment of low-grade chalcocite ores. In certain departments considerable pioneering has been done, which has resulted in marked economies, advantage of which will no doubt be taken where similar conditions make them applicable.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Ore-Drawing Tests and the Resulting Mining Method of Inspiration Consolidated Copper Co.

BY GEORGE R. LEHMAN, B. A., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

THE Inspiration Consolidated Copper Co. had an orebody at Miami, Ariz., of close to 100,000,000 tons of low-grade copper ore, and the method of mining this ore most profitably was of great importance. The selection of a method was practically limited to a "top-slicing," a "shrinkage-stope" or a "caving" method. The "top-slicing" method would give a high extraction of the developed ore, at a high mining cost, and the "shrinkage-stope" or "caving" methods a lower extraction of the developed ore, at a lower mining cost.

## Ore-Drawing Tests

To decide on the method to be used it was necessary to know about what extraction of the ore could be obtained by the "shrinkage-stope" or "caving" methods. As no definite information regarding the question was available at the time, C. E. Mills, the general manager, thought that some information could be obtained by experimenting in drawing ore, covered with capping, from an experimental box. Some work had already been done along these lines by W. C. Browning for the Inspiration Copper Co. and by myself for the Live Oak Development Co.

The idea was to represent as nearly as possible, in the experimental box, the conditions within an area of caved stopes of caved ore all ready for drawing. That is, the area was supposed to represent a semibroken mass with the capping above it in the same condition and ready to follow the ore downward as it was drawn out of the chutes under the ore. To represent the above condition, in the experimental box, crushed ore from the mine was placed in it to a given height, and red barren capping from the mine was placed on top of the ore.

### *The Experimental Box*

The experimental box shown in Fig. 1 was made of wood and glass. The glass was used for the sides so that the drawing action of the outside row of chutes could be seen and noted, the latter being cut in the center by the glass sides. The length of the box was 30 in., width 20 in. and the

height 25 in., and built to a scale of 1 in. = 5 ft., the box represented an area of 150 by 100 ft. and a height of 125 ft. within the mine. The box, as shown in Fig. 1, was divided in the center and only half was used for each test. The bottom was bored with ½-in. holes, representing chutes, 1¼ in. center to center, thus indicating chutes of 2½ ft. diameter spaced on 6¼-ft. centers according to the scale. Thus the ore could be represented as drawn at either 6¼, 8⅞, or 12½-ft. chute intervals. The holes were fitted with wooden plugs through which wire nails were driven, so

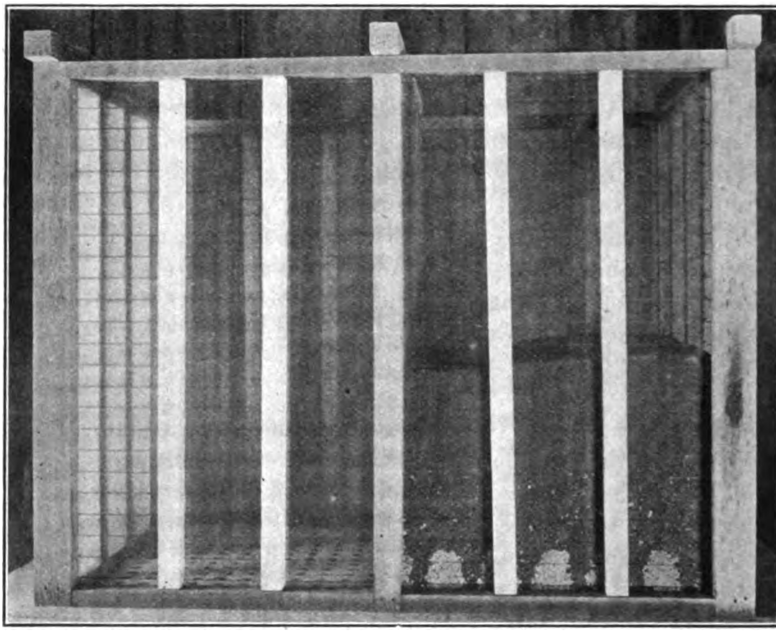

Fig. 1.—Experimental Box Filled with Crushed Ore and Capping; Drawn through Holes in the Bottom Representing Ore Chutes. Scale, 1 In. of Box =5 Ft. of Mine.

that when the plugs were in place the heads of the nails projected just the least bit above the inner bottom of the box. The nail was used to clear the chute when it "hung up."

### *Conditions of Tests*

In all, 16 tests or experiments were made in a period of 6 months beginning in the latter part of the year 1913. In all the tests the rock representing capping was nearly barren, containing only from 0.03 per cent. to 0.08 per cent. copper. The color of the capping was a dark red and that of the ore a light gray, thus giving a sharp and definite line between them, as clearly seen in Fig. 1.

In the first two experiments, weights and assays were not used. The

ore in the box and products drawn from the chutes were measured by volume. They were run to get an idea, in a short time, of what could be done. In all the other tests the ore was weighed, sampled and assayed for copper, as were the chute products drawn during the test.

Tests Nos. 3 and 4 were made with fine and coarse ore mixed, the capping being a similar mixture of coarse and fine material. The conditions of test No. 3 represented 35 ft. of ore and 35 ft. of "capping" and those of test No. 4 represented 70 ft. of ore and 55 ft. of capping. Both tests were drawn at the closest chute interval, 6¼ ft. In both tests ore was drawn from one chute equivalent to about 10 tons, on the scale adopted, and then 10 tons from the next chute, and so on until 10 tons had been drawn from each chute. The drawing of all the chutes was then repeated, 10 tons at a time, until capping appeared in each chute. The total chute-drawings product was then weighed, sampled, assayed and called the clean-ore product. Drawing was then continued as before until each chute showed that it was running one-quarter capping. To determine this point, a sample mixture was kept in a test-tube and the chute product compared with it. This product was then weighed, sampled and assayed. Again the drawing continued until each chute showed about one-half capping. This point was also determined by comparing with a sample mixture and this product also weighed, sampled and assayed. The tests were concluded by drawing 10 tons from each chute in rotation, weighing, sampling and assaying the product. The assay value for the combined products of the tests was calculated, as the portions removed for assay could not be returned. Details of test No. 3 are given in Table 1.

TABLE 1.—*Details of Test No.* 3

Weights and assays—35 ft. of ore and 35 ft. of capping.
Ore and capping—coarse and fine mixed.
Largest boulder—1 ft. diameter.
Smallest particle—powder.
Drawn at 6¼-ft. centers—10 tons at one drawing.

Ore in box 127.50 lb.................. 1.96 per cent. copper
Capping.............................. 0.08 per cent. copper

| Products | Pounds | Per Cent. | Per Cent. Cu |
|---|---|---|---|
| (1) Drawn clean to capping................ | 81.00 | 63.52 | 1.96 |
| (2) Drawn clean to ¼ capping............ | 8.50 | 6.67 | 1.36 |
| (3) Drawn clean to ½ capping............ | 8.50 | 6.67 | 1.10 |
| (4) Drawn 10 tons each chute.............. | 11.00 | 8.62 | 0.79 |
| Totals and average...................... | 109.00 | 85.48 | 1.74 |

TABLE 1.—*Details of Test No. 3 (Continued)*

| Products | Per Cent. Total | Grade, Per Cent. Copper | Product | | Per Cent. Total Cu Recovered |
|---|---|---|---|---|---|
| | | | Per Cent. Ore | Per Cent. Capping | |
| Clean ore.......(1) | 63.52 | 1.96 | 63.52 | ..... | 63.52 |
| Ore and capping (2) | 6.67 | 1.36 | 4.54 | 2.13 | 4.54 |
| Ore............(3) | 6.67 | 1.10 | 3.62 | 3.05 | 3.62 |
| | 76.86 | 1.83 | 71.68 | 5.18 | 71.68 |

Estimate Applied to above Figures to Find Final Product of 1 Per Cent. Copper and Applied to Ores Below

| Grade, Per Cent. Copper | Extraction, Per Cent. of Total | Grade of Product, Per Cent. Copper | Per Cent. of Total | | Per Cent. of Product | |
|---|---|---|---|---|---|---|
| | | | Ore | Capping | Ore | Capping |
| 2.00 | 79.500 | 1.842 | 72.96 | 6.54 | 91.77 | 8.23 |
| 1.90 | 78.340 | 1.763 | 72.44 | 5.90 | 92.47 | 7.53 |
| 1.80 | 77.180 | 1.681 | 71.84 | 5:34 | 93.08 | 6.92 |
| 1.70 | 75.490 | 1.605 | 71.07 | 4.42 | 94.14 | 5.86 |
| 1.60 | 73.490 | 1.529 | 70.06 | 3.43 | 95.33 | 4.67 |
| 1.50 | 71.490 | 1.449 | 68.92 | 2.57 | 96.41 | 3.59 |
| 1.40 | 69.290 | 1.367 | 67.54 | 1.75 | 97.48 | 2.52 |
| 1.30 | 67.290 | 1.283 | 66.35 | 0.94 | 98.61 | 1.39 |
| 1.20 | 65.290 | 1.195 | 65.02 | 0.27 | 99.58 | 0.42 |
| Above Applied to Mining—25 Per Cent. by Development | | | | | | |
| 2.00 | 84.625 | 1.889 | 79.81 | 4.81 | 94.31 | 5.69 |
| 1.90 | 83.755 | 1.804 | 79.41 | 4.35 | 94.81 | 5.19 |
| 1.80 | 82.885 | 1.717 | 78.95 | 3.93 | 95.26 | 4.74 |
| 1.70 | 81.617 | 1.634 | 78.35 | 3.26 | 96.00 | 4.00 |
| 1.60 | 80.117 | 1.551 | 77.59 | 2.53 | 96.84 | 3.16 |
| 1.50 | 78.617 | 1.465 | 76.72 | 1.89 | 97.59 | 2.41 |
| 1.40 | 77.117 | 1.378 | 75.86 | 1.26 | 98.37 | 1.63 |
| 1.30 | 75.467 | 1.289 | 74.80 | 0.66 | 99.12 | 0.88 |
| 1.20 | 73.967 | 1.197 | 73.78 | 0.19 | 99.74 | 0.26 |

Besides estimating the extractions for different grades of ore, based on the experiments, the extractions for different grades of ore applied to mining were calculated, as shown in the lower part of Table 1, assuming 25 per cent. of the ore extracted by development and other work before the ore caves, when mining 35 ft. of ore, and 15 per cent. when mining 70 ft. of ore. For each grade, 1 per cent. copper was taken as the limit of commercial product. This point had to be interpolated from the product just above 1 per cent. and the product just below 1 per cent.

This way of estimating the results possible in mining a 1.80 per cent. copper ore, according to test No. 3 indicates a total extraction of 82.88 per cent. of 1.72 per cent. ore, 78.95 per cent. of which was original ore, the chute product being 95.26 per cent. ore and 4.74 per cent. capping.

In the next two tests, Nos. 5 and 6, the fines were screened out of the ore and capping, making both ore and capping represent on the scale of 1 in. = 5 ft., coarse chunks of 4 to 12 in. diameter. The chute interval and method of drawing were the same as in tests Nos. 3 and 4. Test No. 5 represented drawing 35 ft. of ore and test No. 6 represented 70 ft. of ore. Compared with tests Nos. 3 and 4, as shown in Table 2, both tests show higher extractions.

Tests Nos. 7 and 8 were duplicates of 5 and 6 except that the chute interval was 8⅞ ft. instead of 6¼ ft. These two tests show lower extractions than Nos. 5 and 6.

Tests Nos. 11 and 10 were the same as Nos. 5, 6, 7 and 8 except that the chute interval was 12½ ft. As will be noted in Table 2, still lower extractions were obtained with these two tests, showing conclusively that the greater the distance between chutes the less the ore extraction.

To determine the effect on extraction if no care is taken as to the amount drawn at one time, two tests, Nos. 12 and 13, were run. They were both made with conditions representing 35 ft. of ore and 35 ft. of capping, drawn at a chute interval of 8⅞ ft. In these two tests each chute was drawn direct to capping in rotation, then direct to one-fourth capping and then direct to one-half capping. Comparing these two tests with No. 7, with which all conditions were identical, except the method of drawing, it was found that in No. 7 a greater extraction of clean ore, before capping appeared, was obtained, but that in Nos. 12 and 13 a greater total extraction was obtained to the limit of 1 per cent. copper, as shown in Table 2.

Tests Nos. 14, 15 and 16 were special experiments made with conditions representing a chute interval of 8⅞ ft. and the column of ore 35 ft. In No. 14 the ore was coarse, 12 to 30 in. diameter, and the capping fine; a low extraction was obtained. Test No. 15 was the reverse of No. 14, the ore was fine and the capping coarse; the extraction was but very little higher than in No. 14. The last test, No. 16, was made with conditions representing both ore and capping fine. This apparently was the worst condition, the extraction being very low.

### *Conclusions Derived from Experiments*

The conclusions derived from the experiments were as follows: (1) The less the distance between chute centers, the greater the extraction; (2) the higher the ore column, the greater the extraction; (3) the less the amount drawn at a time from each chute, uniformly, the greater the extraction of clean ore before the capping appeared, but not the greater the total

TABLE 2.—*Comparison of Extractions Based on 1.80 Per Cent. Copper Ore*

| Test No. | Condition of Ore and Capping | Method of Drawing | Distance between Chute Centers, Feet | Depth of Ore, Feet | Extraction, Per Cent. Total | Grade of Product, Per Cent. Copper | Per Cent. Total | | Per Cent. Product | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | Ore | Capping | Ore | Capping |
| 3 | Coarse and fine mixed. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 35 | 77.180 | 1.681 | 71.84 | 5.34 | 93.08 | 6.92 |
| 4 | Coarse and fine mixed. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 70 | 93.950 | 1.705 | 88.76 | 5.19 | 94.48 | 5.52 |
| 5 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 35 | 86.400 | 1.717 | 82.30 | 4.10 | 95.26 | 4.74 |
| 6 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 70 | 95.030 | 1.765 | 93.13 | 1.90 | 98.00 | 2.00 |
| 7 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 8⅞ | 35 | 78.570 | 1.717 | 74.85 | 3.72 | 95.26 | 4.74 |
| 8 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 8⅞ | 70 | 89.550 | 1.764 | 87.71 | 1.84 | 97.94 | 2.06 |
| 10 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 12½ | 70 | 85.110 | 1.741 | 82.24 | 2.87 | 96.63 | 3.37 |
| 11 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 12½ | 35 | 67.390 | 1.707 | 63.81 | 3.58 | 94.69 | 5.31 |
| 12 | Coarse, 4 to 12 in. | Direct to cap.—¼ cap.—½ cap. each chute. | 8⅞ | 35 | 84.290 | 1.708 | 79.86 | 4.43 | 94.74 | 5.26 |
| 13 | Coarse, 4 to 12 in. | Direct to cap.—followed by 10 tons each chute. | 8⅞ | 35 | 83.980 | 1.703 | 79.33 | 4.65 | 94.46 | 5.54 |
| 14 | Ore coarse, 12 to 30 in. Capping fine, 2 in. to powder. | 25 tons each chute to capping—then 15 tons each chute. | 8⅞ | 35 | 76.050 | 1.600 | 67.36 | 8.69 | 88.57 | 11.43 |
| 15 | Ore fine, 2 in. to powder. Capping coarse, 12 to 18 in. | 25 tons each chute to capping—then 15 tons each chute. | 8⅞ | 35 | 80.040 | 1.564 | 69.24 | 10.80 | 86.51 | 13.49 |
| 16 | Ore and capping, fine. | 25 tons each chute to capping—then 15 tons each chute. | 8⅞ | 35 | 53.940 | 1.635 | 48.85 | 5.09 | 90.57 | 9.43 |

Above Tests Applied to Mining 1.80 Per Cent. Ore—25 Per Cent. by Development for 35 Ft.; 15 Per Cent. for 70 Ft.

| | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| 3 | Coarse and fine mixed. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 35 | 82.885 | 1.717 | 78.95 | 3.93 | 95.26 | 4.74 |
| 4 | Coarse and fine mixed. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 70 | 94.857 | 1.720 | 90.52 | 4.33 | 95.43 | 4.57 |
| 5 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 35 | 89.800 | 1.740 | 86.72 | 3.08 | 96.57 | 3.43 |
| 6 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 6¼ | 70 | 95.775 | 1.770 | 94.13 | 1.65 | 98.28 | 1.72 |
| 7 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 8⅞ | 35 | 83.927 | 1.742 | 81.15 | 2.78 | 96.69 | 3.31 |
| 8 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 8⅞ | 70 | 91.117 | 1.770 | 89.55 | 1.57 | 98.28 | 1.72 |
| 10 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 12½ | 70 | 87.343 | 1.751 | 84.90 | 2.44 | 97.20 | 2.80 |
| 11 | Coarse, 4 to 12 in. | Clean to cap.—¼ cap. ½ cap.—10 tons each chute. | 12½ | 35 | 75.542 | 1.738 | 72.87 | 2.67 | 96.46 | 3.54 |
| 12 | Coarse, 4 to 12 in. | Direct to cap.—¼ cap.—½ cap.—each chute. | 8⅞ | 35 | 88.217 | 1.734 | 84.89 | 3.33 | 96.23 | 3.77 |
| 13 | Coarse, 4 to 12 in. | Direct to cap followed by 10 tons each chute. | 8⅞ | 35 | 87.985 | 1.731 | 84.52 | 3.47 | 96.06 | 3.94 |
| 14 | Coarse ore, fine capping. | 25 tons each chute to capping—then 15 tons each chute. | 8⅞ | 35 | 82.037 | 1.661 | 75.53 | 6.51 | 92.06 | 7.94 |
| 15 | Fine ore, coarse capping. | 25 tons each chute to capping—then 15 tons each chute. | 8⅞ | 35 | 85.030 | 1.633 | 76.92 | 8.11 | 90.46 | 9.54 |
| 16 | Ore and capping, fine. | 25 tons each chute to capping—then 15 tons each chute. | 8⅞ | 35 | 65.455 | 1.698 | 61.64 | 3.82 | 94.17 | 5.83 |
| | | Average | | | 85.38 | 1.723 | 81.63 | 3.75 | 95.16 | 4.39 |

extraction to a 1 per cent. copper final product; (4) the coarser the ore and capping without fines, the greater the extraction; (5) the higher the grade of ore, the greater the extraction, when a uniforn limit of grade is allowed at the end of the chute drawing. (6) Applied to mining, the results of the experiments indicate that the greater the amount extracted by development and undercutting, the greater will be the total extraction. This is on the assumption that when the ore caves it occupies the same volume as in place.

Some suggestions deduced from these experiments considered desirable to carry out in mining were: (1) Chute centers for drawing caved ore should be as near together as the ground will allow; (2) ore should be caved in as high columns as practicable; (3) when capping appears at the chutes, no more should be drawn unless the bottom slice is being mined.

While all of the tests described were made with a barren capping, in the actual mining of a sulphide orebody having mixed or oxidized ore over it in place of barren capping, the recovery of sulphide ore would be considerably increased over the experimental results, provided the mixed or oxidized ore could be treated at a profit.

In deciding what extraction could be expected by caving from 35 ft. to 70 ft. of ore in average caving ground, the results obtained in tests Nos. 10 and 11, as detailed in Table 2, were considered conservative, provided the chutes were closely watched and ore of below the commercial limit not drawn. If the ground under the caved ore, the location for raises and chutes, is hard and strong, results of tests Nos. 7 and 8 could be taken as obtainable. This opinion is based on the supposition that the capping will break up in about the same way as the ore and not into a fine powdery sand.

## Method of Mining

The Inspiration company's method of mining is one of the so-called Caving systems whereby the ore is caused to cave and crush itself, thus reducing to a minimum the blasting and handling. It is a modification of the method introduced by Felix McDonald at the Ohio Copper Co.'s mines in Utah, and was put into operation at the Inspiration under his supervision. The method consists, essentially, of undercutting the ore (taking out a horizontal slice), allowing the ore above to cave and crush, and drawing off the crushed ore through small inclined raises driven under the caved ore, into main inclined raises that lead down to the haulage-drift chutes.

### *Preliminary Development Work*

By referring to the accompanying illustrations showing the progressive steps of mine development, a good understanding of the method will be obtained.

After shafts have been sunk, the haulage drifts are driven under the ore of the section or sections to be mined at intervals of 100 ft. as shown in Fig. 2. These drifts are of large size, being 9 ft. wide at the base of the rails, 7½ ft. wide at the cap, and 7½ ft. high above the rail base. Where timbered, the above refer to inside timber dimensions.

After the haulage drifts are in, or during their driving, "pony" sets are put up every 25 ft. along the drifts. The "pony" set is about 5 ft. high, placed on top of the regular drift set and is the place from which the car loaders operate the chute gates discharging into the haulage cars.

Inclined raises are then started from the "pony" sets at an inclination of from 50° to 54°, depending on whether the sublevels are to be

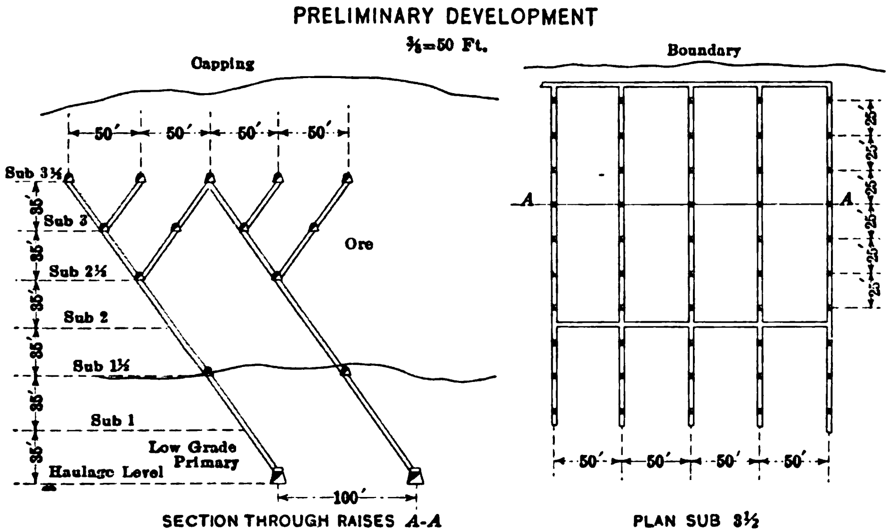


FIG. 2.—PRELIMINARY DEVELOPMENT UNDERTAKEN BY DRIVING INCLINED RAISES FROM HAULAGE-LEVEL DRIFTS.

30 or 35 ft. vertically apart, in a plane at right angles to the haulage drifts. When these raises are up from 10 to 15 ft., chutes are built at the "pony" sets and ore gates of steel are installed.

The raises are then advanced until the first sublevel is reached. "Sub" drifts are then started, from the first raises to reach the required level, and are driven parallel to the haulage drifts. These drifts break into the other raises every 25 ft., as they advance, thus cutting down the cost of handling the drift "muck." The raises are then again advanced to the next sublevel where drifting is to start. The drifts on this level are then driven, meeting the raises as they advance. All of these development raises are about 4 ft. in diameter, and the subdrifts 6 by 7 ft.

This method of raising and sublevel drifting is continued until the

height is reached where the first undercut is to be made, and the drifts are driven as before.

The first undercut sublevel will be located below the top of the ore, at about the height at which it is intended to cave. When the preliminary development is finished, the sublevel drifts on the first undercutting level are 50 ft. apart, as they are on the "sub" just below, as shown in Fig. 2. On all the "subs" below, the drifts are 100 ft. apart. All the drifts on each "sub" are also connected with inclined raises every 25 ft. Cross drifts, connecting all the drifts on each "sub," are also driven at intervals of about 150 ft.

The next work, before undercutting is commenced, is to drive on the level to be undercut other "sub" drifts, between and parallel to those

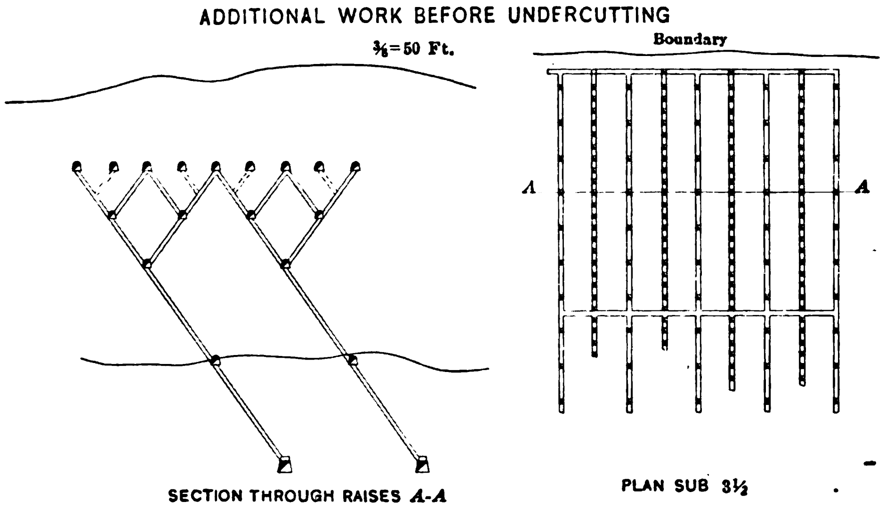


FIG. 3.—PRECEDING UNDERCUTTING, THE TOPMOST LEVEL IS DEVELOPED BY DRIFTS SPACED ON 25-FT. CENTERS AND CONNECTED WITH THE BRANCH, OR "FINGER RAISES."

already driven, as best shown by Fig. 3, making the drifts on this "sub" 25 ft. center to center. These drifts, as they advance, meet branch raises of the main inclined raises at 12½- or 25-ft. intervals as desired. These small branch raises, shown in Fig. 4, are called "finger raises," many of which are put up just before or ahead of the undercutting operations.

### *Undercutting*

Undercutting the ore is accomplished by starting at a cross drift, on the boundary of the section to be mined, and in retreating from that cross drift, as best shown in the plan of Fig. 5, drilling deep holes, at nearly right angles to the drifts, into the pillars between them, and blasting out the ground. Three holes are drilled in each side at different angles in

the same vertical plane, with one hole in the back of the drifts, thus making seven holes to the round. For this work a water-hammer one-man drill with large steel is used. The holes are from 8 to 10 ft. in depth. Usually the rounds are blasted one at a time, the undercutting receding from the caved ground.

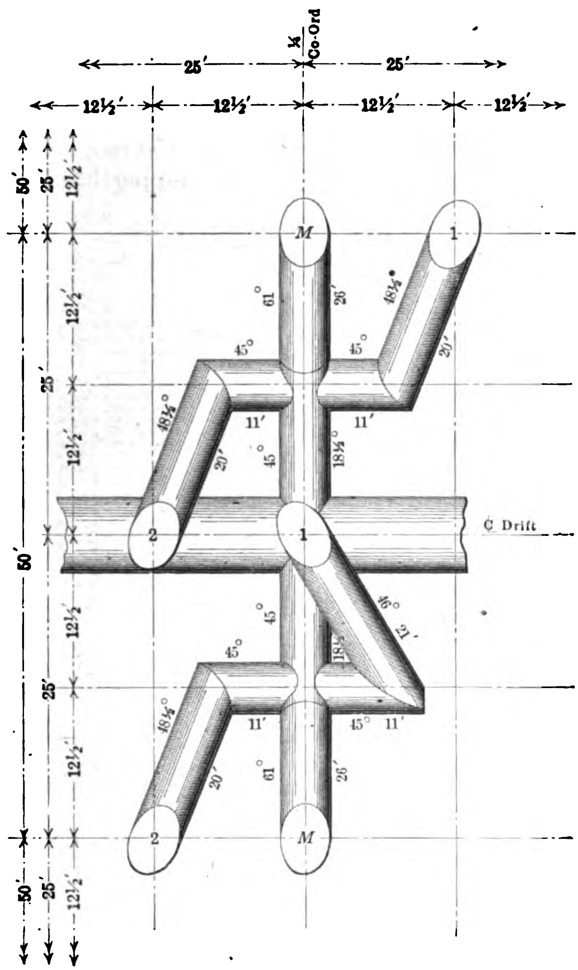


FIG. 4.—"FINGER RAISES", AN IDEAL TOP VIEW SHOWING THEIR CONNECTIONS WITH MAIN RAISES, *M*, AND SUBLEVEL DRIFT.

Ore from the undercut is drawn through the "finger raises," in each of which has been built an ordinary board chute to control the drawing. The "finger-raise" chutes are located about 4 or 5 ft. below the sublevel, so that they will not be blasted in shooting the undercutting holes.

While in some ground, the ore begins to cave as soon as it is undercut, in hard ground caving does not start until the undercutting has receded a considerable distance.

### *Ore Drawing*

After the ore caves it is drawn off as desired. Large boulders that will not pass the raise chutes are blasted. In drawing off the ore, the chute "tappers" work in pairs. One "tapper" goes up a raise, from the "grizzly sub," which is the first "sub" below the undercutting level, opens a chute gate and draws the ore, while the other "tapper" works the ore through the grizzly on the "sub" below.

All grizzlies have about 1-ft. openings and are made of timber or steel rails. They are placed over all raises on the "grizzly sub" and are set at right angles to the drift. Pieces of ore too large to pass the grizzly are broken with an 8-lb. hammer by the "tapper" tending the grizzly.

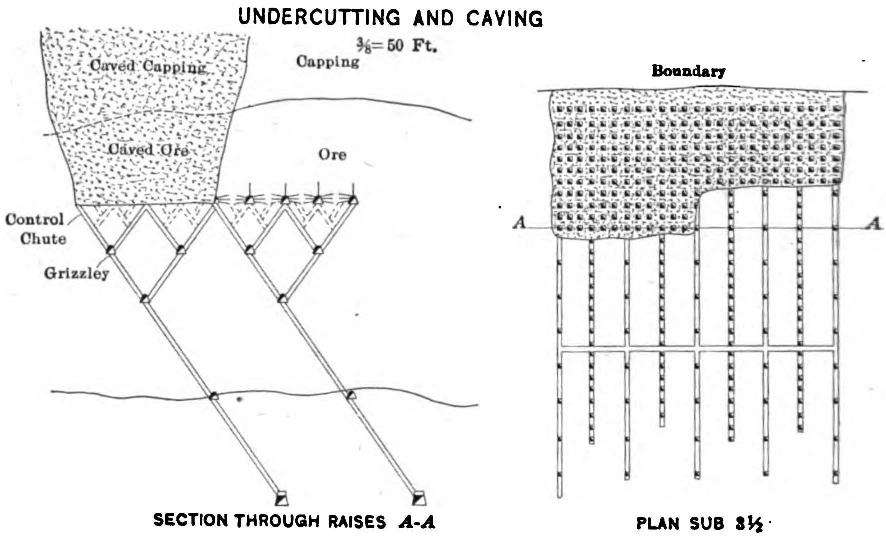


FIG. 5.—STARTING UNDERCUTTING BY BLASTING DOWN THE PILLARS BETWEEN SUBLEVEL DRIFTS.

After passing the grizzlies, the ore falls into the main raises and down to the haulage-drift chutes where it is loaded into 5-ton cars and hauled in trains, of 15 to 20 cars, to the shaft bins.

The second undercut can be located at one, or any number of sublevels below the first undercut, according to the height of ore which it is desired to cave. It could be located at the bottom of the ore if desired. In fact, the first undercut could be located at the bottom of the ore if it were known that the orebody to be mined could be caved throughout its total height.

### *Underground Haulage and Hoisting Arrangements*

At the Inspiration mine the ore trains are hauled by compressed-air locomotives built by the H. K. Porter Co. of Pittsburgh. Arrived at the

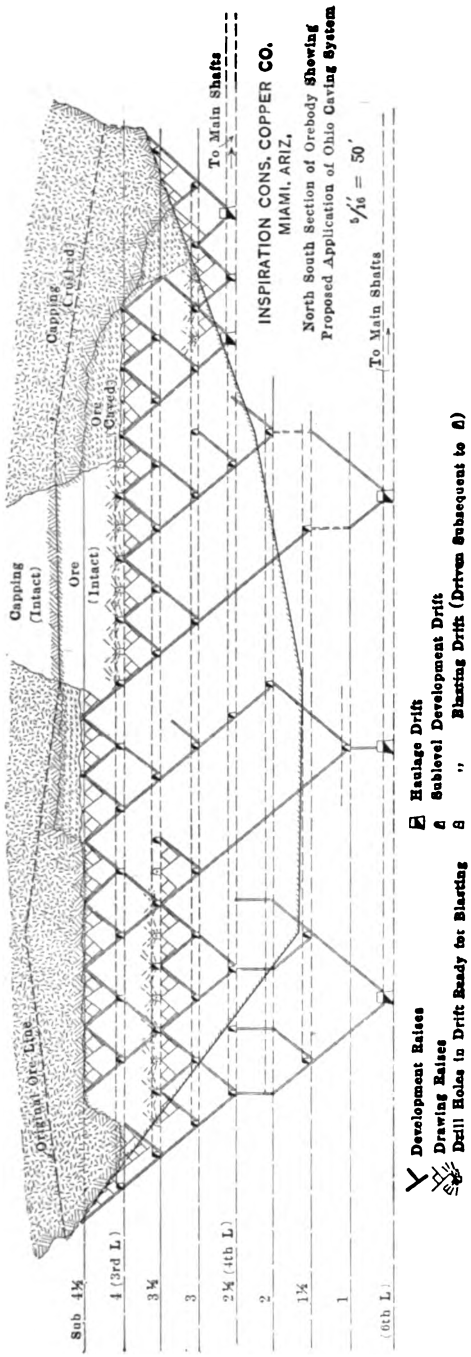


FIG. 6.—ARRANGEMENT OF MINE LEVELS AND RAISES UTILIZED IN INSPIRATION CAVING SYSTEM.

shaft, the cars are tippled into the shaft bins by barrel-shaped tipples, five cars at a time. From the shaft bins the ore is loaded automatically into 12½-ton skips and hoisted by electrically driven automatic hoists to the steel bins on the surface.

For hoisting ore, two main shafts 101 ft. apart are used. Both shafts have three compartments, of which two in each shaft are skipways. The third compartment in one shaft contains the ladderway, pipes and electric conduits, while in the other shaft the third compartment contains an Otis double-deck elevator for hoisting and lowering men. Both shafts, the underground bins, and the stations are all lined with reinforced concrete.

Only two levels are used for hauling ore, the 4th and 6th. The vertical distance between these two levels is 130 ft. A good idea of the arrangement of the mine is obtained from Fig. 6. On the 6th level the shafts are connected by two drifts, one for each shaft. In each drift there is a tipple over the bin at the shaft. The arrangement of the drifts, bins, tipples and stations is symmetrical with a center line between the two shafts so that the tipples are operated by one man from a central location.

On the 4th level one double-track drift passes between the two shafts and is connected to the shaft stations by a small cross drift. The ore is dumped by one tipple, on this level, into a small bin which is connected with the 6th level bins by a concrete-lined inclined raise. This arrangement makes necessary only one loading level for the skips.

The Inspiration Consolidated Copper Co. commenced underground development on a large scale toward the latter part of 1913 but did not start regular mining operations until August, 1915. The tonnage mined gradually increased as the concentrator was able to handle it until at present (June, 1916) an average tonnage of 16,700 tons is being mined daily.

The cost of mining is now 60 c. per ton, including 20 c. for development and all fixed charges.

The percentage of ore and copper extraction obtained by the Inspiration method cannot be determined positively until some section has been entirely mined from top to bottom of the ore. But based on results in six sections of the mine from the first undercut to the capping estimated to contain 1,886,450 tons, and which have been almost completely drawn to capping, the ore extraction is 102.44 per cent. and the recovery of copper 86.52 per cent. The second undercut is expected to increase the copper recovery.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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# The Block Method of Top Slicing of the Miami Copper Co.

BY E. G. DEANE,* B. S., E. M., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

A METHOD of top slicing has been devised at the Miami Copper Co.'s mine at Miami, Ariz., which differs radically in some ways from the customary methods of top slicing.

The area of that section of the orebody in which top slicing is used is about 800 ft. square. The ore, while for the most part soft, is, nevertheless, considerably harder than the capping. The latter is siliceous, seldom containing any clay or other binding material, and breaks into fine particles so that it runs like sand if given the opportunity. Because of these facts, and because the ore is above the average grade of the mine ore, it has been mined by top slicing.

Haulage levels are opened up 150 ft. apart, vertically, with two sublevels between at 50-ft. intervals, to facilitate the building of ore chutes. These sublevels are used during slicing for distributing air in the ventilation system. On the haulage level the drifts are spaced on 50-ft. centers, and raises along these drifts are also spaced on 50-ft. centers, except the incline raises as hereafter noted. The raises are cribbed where necessary. Where the wear will be excessive, ¼-in. iron plates are spiked to the top of every third set of cribbing, for its protection.

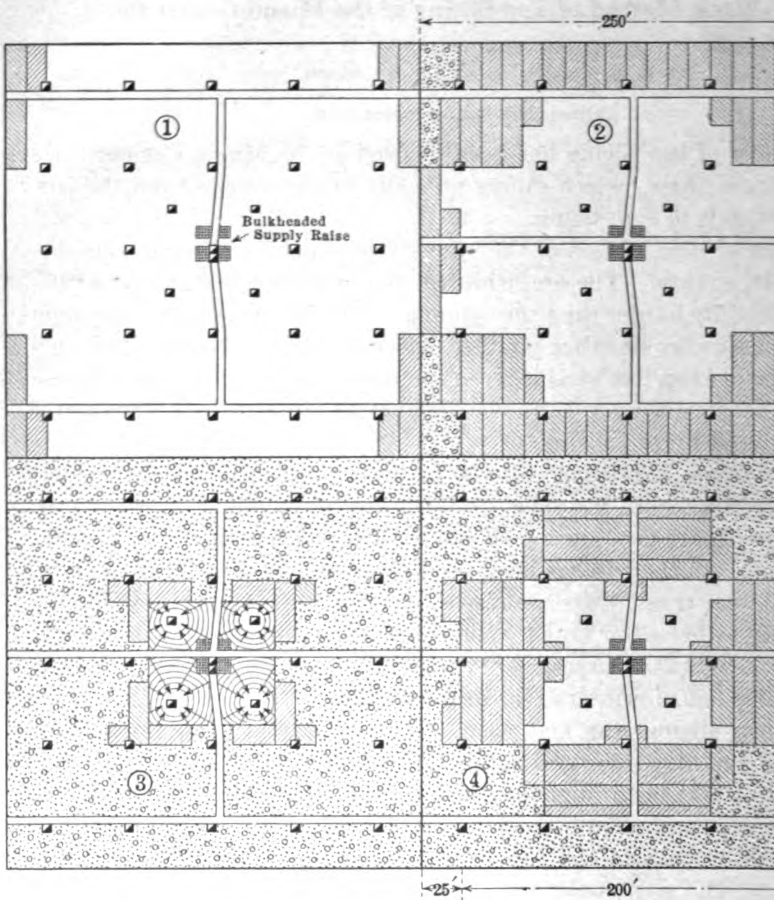
When top slicing was first used, an attempt was made to carry a slicing face from fifty to several hundred feet long. Timber and other supplies were brought in through long drifts from an auxiliary shaft. Great difficulty was experienced in keeping these drifts open, the side pressure breaking the posts and the top weight breaking both caps and posts. Furthermore, the men could not work efficiently while these drifts were being repaired. The slicing faces advanced irregularly, due to varying conditions, and in many ways the results were not all that could be desired. It was then decided to divide the slicing area into blocks 200 ft. square and this was later changed to 250 ft. square.

At the center of each block a two-compartment raise is put up as a supply raise, the compartment being 2 ft. 6 in. and 4 ft. by 4 ft. 4 in., the smaller being used as a manway. Station sets of 12 by 12 timber with

* Slicing Efficiency Engineer, Miami Copper Co.

9- or 10-ft. posts are put in and an Ingersoll stretcher bar air mounted above the larger raise compartment to use in hoisting the steel, etc. Four bulkheads, built solidly of blocks of square tim put in as shown in Fig. 1. Two of these are 7 by 11 ft. in size, other two 7 by 7 ft.

Two drifts, usually untimbered at first, are run out 100 ft. on axis of the supply raise. At the end of each of these and at righ




FIGS. 1 TO 4.—PLAN SHOWING PROGRESSIVE STEPS IN BLOCK METHOD SLICING.

to them, two drifts are run 125 ft. to the block limits. When the are reached, slices are started toward the corners of the block. slices are timbered either with single sets consisting of two 8-ft. p a 12-ft. cap, or with a double set consisting of three 8-ft. posts 7-ft. caps, depending upon the ground. The ore is taken to t above.

As soon as the first slices have advanced a few feet, second and third slices are started, and also first slices toward the centers of the block limits. Every man who can work to advantage is put on and the ore mined with the greatest possible speed.

Where there is a sufficient mat of old timbers in the back to obviate the danger of the capping running into the ore there is no permanent floor laid, the planks used in shoveling and wheeling the ore being taken up later. Where there is no mat or it is not sufficient, a floor of 2-in. plank spiked to 2 by 10 sills is laid. Formerly 5 by 10 and 4 by 8 sills were used, but it was found that after being subjected to the pressure and heat of a completed stope, a 5 by 10 or 4 by 8 sill seemed to have practically no more strength than a 2 by 10.

As soon as the timbers in the slices show signs of taking weight, bulkheads are built of old timbers, obtained either from the mat in the back, or from repair work in other parts of the mine. As soon as possible, the posts are drilled and the slices shot down.

By the time slicing has started, four drifts have been run to the centers of the sides of the blocks as shown in Fig. 2 and slicing is also done from these. Working as intensively as possible, all ore except the four central pillars is quickly mined out as shown in the series of illustrations. As about the last of this ore is being taken, crosscuts are driven to incline raises, put up to about the center of the four central pillars, as shown in Fig. 3, and slicing continues, working from the outside of the remaining ore to the supply-raise bulkheads first put up as shown in Fig. 1, thus completing the stope. By this time, these bulkheads, which were 10 ft. high when put in, have squeezed to from 4 to 6 ft. in height. Upon completion, the stope is shot down and another may then be started below, though it is best to let the ground settle for a few weeks.

At first thought the criticism suggests itself that with such weight the mining method used intensifies this weight as the block approaches completion. But experience has shown that, as a rule, the maximum weight is taken by the timbers at about the time the outside pillars are completed, and as mining progresses a larger proportion of the weight is taken by the bulkheads in the outside slices. At no time does the weight on the remaining ore and the slices still necessarily open get beyond control. These central pillars constitute our cheapest ore, not only because of the pillar raises but because the ground has been fractured by the weight, and lifters are the only holes necessary to break the face.

All drilling is done with plugger machines, using a water spray attached to a 5-gal. can. No cars are used in the slices, all ore being shoveled directly into the chutes or wheeled in barrows. Round timber is used almost exclusively in the stopes, because of its superior strength.

Ten feet has been taken as the standard height of a slice. If a greater

height is taken the ore sloughs off the top of the slicing face faster tha it can be mucked out and the bottom shot, thus shortly caving the slic It is possible that later, when a good mat has been formed, sublev caving may be used, but so far, where tried, it has not been successful.

The ventilation of these slice blocks is very important because, wit out it, the heat coming out of the mat is excessive and prevents efficie work. It is accomplished by connecting one of the sublevels below t slicing floor with the discharge end of a 60,000-cu. ft. fan. Openin are maintained from the sublevel to raises through which it is desired force air.

As a result of the change to the block method of slicing, and to usi forced ventilation, the production per mucker-shift has been raised fro 9 to 20 tons, the production per man-shift from 5 to 10 tons.

While the details of this method have been fully worked out and large tonnage of ore has already been extracted by its use, the work date has to a large extent been preparatory to systematic work for low lifts. It is not possible to give representative costs, but the followi is an estimate of what is expected:

*Mining Costs per Ton*

| | | |
|---|---|---|
| Preliminary development | $0.035 | |
| Haulage development | 0.025 | |
| Other development (raises, etc.) | 0.080 | |
| Total development | | $0.140 |
| Stope Costs: | | |
| Miners, at $3.75 | 0.080 | |
| Muckers, at $3.75 | 0.160 | |
| Drills | 0.030 | |
| Explosives | 0.040 | |
| Timbering, labor | 0.070 | |
| Timbering, supplies | 0.130 | |
| General—bosses, nippers, etc. | 0.040 | |
| Total stoping | | 0.550 |
| Haulage | | 0.055 |
| Hoisting | | 0.04 |
| Pumping | | 0.005 |
| General underground | | 0.025 |
| Ventilation | | 0.015 |
| Engineering and sampling | | 0.016 |
| Underground lighting | | 0.004 |
| Mine surface | | 0.030 |
| Total | | $0.880 |

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# Cost and Extraction in the Selection of a Mining Method

BY C. E. ARNOLD, B. S., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

IN attacking the problems of mining and treating large disseminated copper orebodies such as those occurring in the Miami or the Ray district of Arizona, one of the vital questions to be decided is, "What is the value of the orebody per ton in place?" The importance of this question lies in the fact that its answer helps to reach a decision concerning the method of mining to be employed.

Let it be assumed for the moment that milling and smelting practice, especially the latter, is more or less standardized, and has a cost incapable of much variation, while the cost of mining has a considérable range owing to the large number of mining methods by which a mine may be exploited. For instance, a method may be adopted whereby all the available ore can be extracted perfectly clean (*i.e.*, unmixed with barren material) at a relatively high cost; or a method may be used involving the necessary dilution of a portion of the ore with waste, in order to recover that portion, thus giving a relatively poorer extraction but nevertheless a lower working cost than the former method. Naturally, between these two extremes there lie intermediate methods.

In order to arrive at a solution of a mining method problem that is of any value, it is necessary that accurate and complete estimates of mining, milling, and smelting performances and costs shall be established. By way of illustration is given a case of mining, by the system of undercutting and caving, a large orebody with an average content of 1.50 per cent. copper supposedly all in sulphide form, to which case has been applied those results in mining operations which one may fairly assume are to be expected when viewed in the light of G. R. Lehman's ore-drawing experiments, which form the subject of a paper to be presented at this meeting.

It is assumed that the total costs will be as shown in the following table:

*Costs on 1 Ton of Clean Ore*

| | |
|---|---|
| Mining, including prepaid development | $0.600 |
| Coarse crushing | 0.024 |
| Freight on ore | 0.015 |
| Concentrating | 0.462 |
| Freight on concentrate | 0.002 |
| Smelting, freight on copper, refining, marketing, etc | 0.737 |
| Amortization | 0.120 |
| Total | $1.960 |

To attain these costs, a daily production and treatment of 14,400 tons has been assumed. Other assumptions are as follows: 15 per cent. of orebody is to be extracted clean by development work and undercutting; 63.8 per cent. (75 per cent. of 85 per cent.) is to be extracted clean by caving, leaving 21.2 per cent. part of which is to be extracted by diluting with waste, and the remainder lost. Dilution and extraction are to be related as shown by the curve in Fig. 1. The copper tenor of the concentrate is to vary uniformly between 25.33 per cent. and 30.00 per cent. on 0.80 per cent. and 1.50 per cent. ore respectively. Mill extraction is to vary uniformly between 69 per cent. and 83 per cent. on 0.80 per cent.

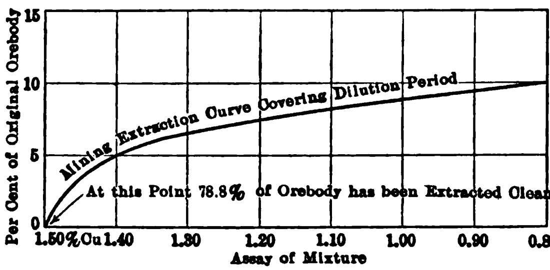


FIG. 1.—SHOWING EFFECT OF DILUTING THE LAST DRAWN ORE WITH CAPPING UPON COPPER ASSAY OF THE MIXTURE.

and 1.50 per cent. ore respectively. The selling price of copper is to average 14 c. per pound. The life of property will be 12 years. Smelting is to be performed under contract with an outside company.

In the above set of assumed costs the charge of $0.12 per ton of ore for amortization represents the outlay necessary to redeem in 12 years all expenditures for plant (not including smelter), lands, railroad, water supply, etc., other than the expenditures required in preparing the mine for the assumed daily production. This latter expense is covered by a charge of $0.20 per ton, which is included in the estimate of $0.60 per ton for mining. The amortization fund is considered as an annuity invested at 4 per cent.

It should be borne in mind that as long as the ore is undiluted with waste, the above total cost of $1.96 is all that is applied to the ton in place, and that the instant dilution starts the cost per ton in place increases because the added waste, until it disappears in the mill tailing, goes through all the operations to which the ore is subjected. Further-

more, when the ore is diluted, the mill extraction, and consequently the smelter return, decreases, likewise the copper tenor of the concentrate decreases. These disadvantages, however, are to a certain extent offset by the fact that on dilution, the increasing concentration ratio and the decreasing copper content of the concentrate lower the smelting cost on the ton of ore in place.

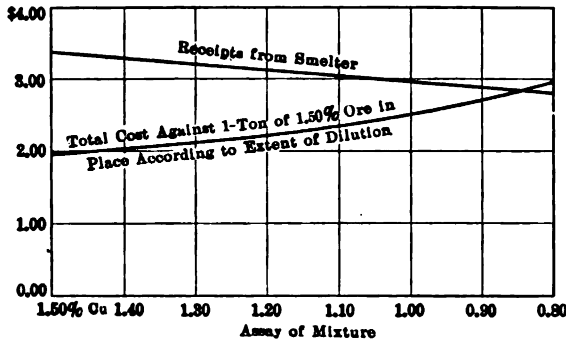


FIG. 2.—SHOWING DECREASING SMELTER RECEIPTS AND RISING COSTS WITH INCREASED DILUTION.

Taking into account all these variables, the curves shown in Fig. 2 were plotted, and then by using these in conjunction with the curve of Fig. 1, the curve of Fig. 3 was established, showing the extent to which dilution could be carried to secure the maximum profit from that ore remaining in place after the extraction of all available clean ore.

The conclusions resulting from this set of calculations are as follows:

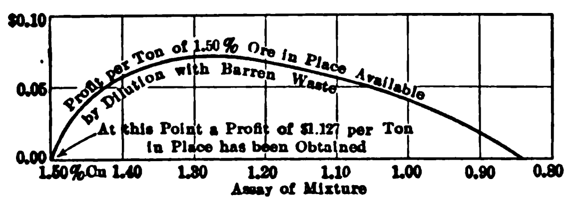


FIG. 3.—SHOWING MAXIMUM POSSIBLE PROFIT WITH ORE DILUTION INCREASED UNTIL ORE ASSAY IS REDUCED TO 1.28 PER CENT. OF COPPER.

1. To obtain a maximum profit the average assay of the dilution product will be 1.28 per cent. copper, the end point being 0.845 per cent., as shown in Fig. 3, at which point receipts from sales of copper just balance total costs.

2. The total ore recovered will be 85.5 per cent., as shown in Fig. 1, of which 78.8 per cent. will be extracted clean and 6.7 per cent. will be extracted in conjunction with 1.15 per cent. of waste.

3. From 100 tons of ore in place there should be recovered:

| | |
|---|---|
| 78.8 tons clean, at a profit of $1.430 | $112.68 |
| 6.7 tons diluted, at a profit of 1.063 | 7.12 |
| 85.5 tons at a total profit of | $119.80 |

Expressed otherwise, the above might be stated thus:

| | |
|---|---|
| 78.8 tons clean ore at a profit of $1.430 | $112.68 |
| 6.7 tons clean ore at a profit of 1.273 | 8.53 |
| Gross profit | $121.21 |
| 1.152 tons waste at a loss of 1.221 | 1.41 |
| Net profit | $119.80 |

This is equivalent to $1.20 per ton in place.

To duplicate this performance by the employment of a mining system that would give, say a 95 per cent. extraction of clean ore (this representing the end of mining operations), a mining cost increased from $0.60 up to $0.77 per ton would have to be borne as indicated below:

| | Per Ton |
|---|---|
| Receipts from smelter | $3.39 |
| Costs other than mining | 1.36 |
| Difference | 2.03 |
| Cost of mining | 0.77 |
| Difference | 1.26 |
| Mining extraction, per cent | 95 |
| Profit per ton in place | $1.20 |

Further, it may be stated that on some particular ore, of higher copper content than 1.50 per cent., it would be equally profitable to apply either a low-cost, low-extraction mining method, or a high-cost, high-extraction method; and with further increase in the copper content of the ore the high-cost, high-extraction method would yield the greatest profit per ton in place.

By the application of similar reasoning to various mining problems such as that outlined above, it will be found that the resulting figures indicate plainly the mining method that will yield the greatest ultimate profit. Should the calculations show that there is little difference in the results of several methods, the logical decision would be to use that method in which the element of doubt is smallest regarding the realization of the estimated costs.

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# Shaft Sinking through Soft Material

BY EDWARD A. SAYRE, E. M.,* DES MOINES, IOWA

(Arizona Meeting, September, 1916)

In shaft sinking for coal mines, the cost item greatly influences the method adopted. This holds true especially when soft material must be traversed. The average life of a coal mine is short. This is due either to the limited area of the coal basin or to the great expense of maintenance and haulage underground when the entries are extended a considerable distance from the hoisting shaft. Therefore, the engineer in opening a coal mine is confronted with the fact that the cost must be kept within certain bounds else the venture will prove unprofitable. For this reason he may be prohibited from adopting certain safe, but expensive, methods that are used in sinking caissons for foundations, driving transportation tunnels, or other work of permanent construction.

A common method of sinking through difficult ground employs the aid of a steel shoe pushed ahead of the shaft timbers. Another is the drop-shaft method. These two methods were used in sinking the main shaft and the air shaft of the Eagle No. 3 mine, at Des Moines, Ia., and the following data show the relative success and cost of the two methods of sinking under the same conditions.

A drill log of the material to be penetrated was as follows (see Plate 1, Fig. 1):

| | Thickness, Ft. | In. | Total Depth, Ft. | In. |
|---|---|---|---|---|
| Drift | 38 | | 38 | |
| Sand | 2 | | 40 | |
| Drift | 2 | 6 | 42 | 6 |
| Jointed clay | 5 | 6 | 48 | |
| Drift, firm | 10 | 6 | 58 | 6 |
| Clay, with sand streaks | 3 | | 61 | 6 |
| Sand | 12 | | 73 | 6 |
| Shale, with two thin beds of coal and thin strata of rock | 86 | 6 | 160 | |
| Rock | 5 | | 165 | |
| Coal | 3 | 9 | 168 | 9 |

*Main Shaft Sunk by Steel-Shoe Method*

This record shows the first 73 ft. to be of drift material, the lower 12 or 15 ft. of which was sand. The remaining 92 ft. to the coal was shale and rock, and offered no trouble in sinking.

* General Manager, Eagle Coal & Mining Co.

The general equipment, purchased from a company that had twice attempted to sink on this property and failed, consisted of one 80-hp. boiler, a hoisting engine, one No. 5 horizontal and one No. 6 vertical

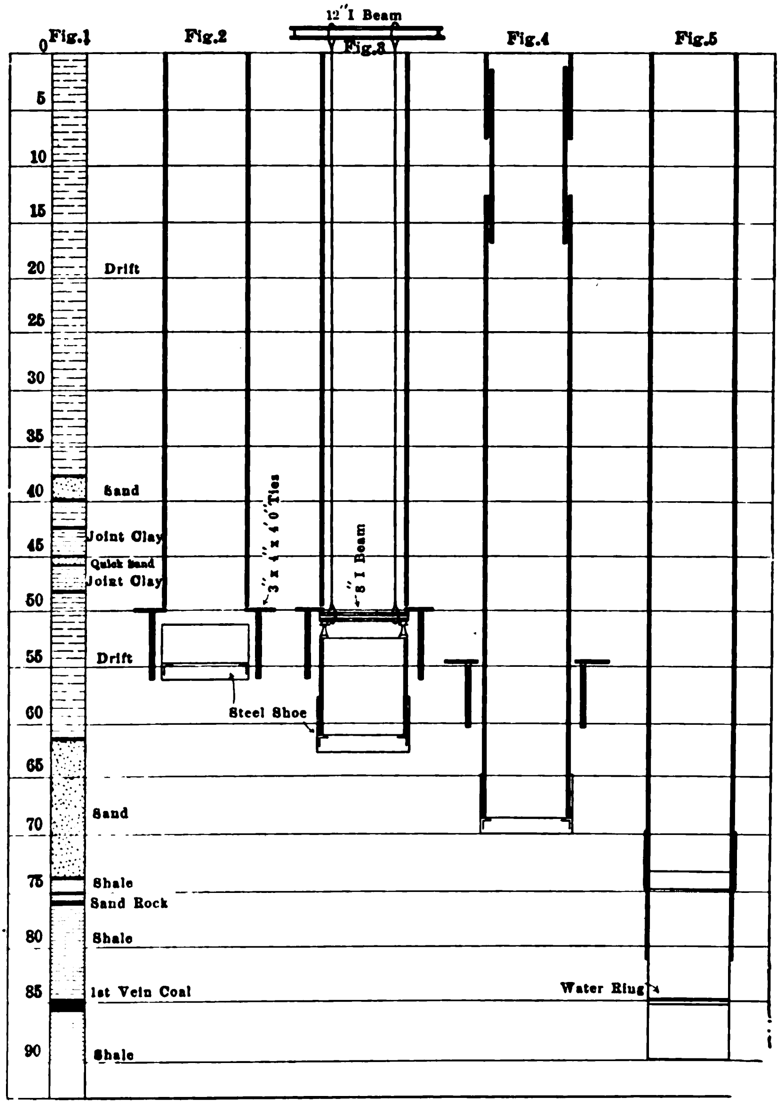


PLATE 1.

Cameron pump, a dump car, hoisting buckets, etc. The men employed were a day and a night engineer, one timberman, and three sinking crews, each consisting of three sinkers and one top man. Work was

continuous 24 hr. a day until the sand was reached, when it was cut to two shifts, the remaining time being used in repacking pumps, etc.

The excavation was made 8 by 14 ft. The curbing for the first 50 ft. consisted of 2 by 4 and 2 by 6 timber laid flatwise, alternately with a quarter shaft bratticed off with 3 by 12 material, and the hoisting compartments divided by 4 by 6 buntons.

No trouble was experienced in sinking until the bed of sand at the 38-ft. level was encountered. This sand carried a small amount of water and, as a shaft protection, a concrete ring was put in behind the curb. The next trouble was experienced from the 42- to 48-ft. level. It had been the intention to assemble the shoe at this level, but what appeared to be a firm green clay proved to be jointed clay with a seam of quicksand through it and the exposed wall would not stand. Temporary 2 by 12 curbing was put in place and then regular curbing carried on down.

At the 50-ft. level, in order to safeguard the assembling of the steel shoe, 125 wooden mine ties, 3 by 4 in by 4 ft. long, were sledged into the clay just below the curb as shown in Plate 2. Steel angles, 6 by 4 in., were fastened with lag screws to the under side of the ties. An excavation 6 ft. deep was then made, undercutting the ties 16 in., and a false curb of 2 by 6 boards built. The shoe was then assembled at this point.

The shoe consisted of four sheets of 3/8-in. steel, two 8 ft. long by 5 ft. high and two 14 ft. long by 5 ft. high, with 3-in. angles at the corners, and a 6-in. pressure angle placed 18 in. from the bottom of the shoe. The curbing was then built inside the shoe to a height permitting the jack screws to be placed against the curb timbers of the shaft, as shown in Plate 2.

The clay was then excavated, the shoe lowered into the sand, and pumping started. The sand carried about 60 gal. of water per minute, which would rise about 18 ft. above the sand when pumping was stopped.

In order to avoid the formation of cavities behind the curb, as little sand as possible was excavated. To prevent an inrush of sand under the shoe, at least 1½ to 2 ft. of sand had to be left within the shoe. The method of sinking was to agitate the sand at the bottom of the shoe and force the shoe through it by means of the jack screws bearing against the shoe and the curb of the shaft. Two methods were used to agitate the sand. First, while the men could reach the bottom of the shoe, they stirred the sand with spades. By this method the shoe was lowered about 18 in. in the sand. Later the pumping system shown in Plate 2 was used.

In this second process the discharge of the No. 6 pump could, when desired, be sent through five 3/4-in. pipes, as indicated in the drawing, and these five jets of water could be played upon the sand at the bottom of

the shoe. It was found that these jets would agitate the sand sufficiently to permit the jack screws to push the shoe down, except when the sand was at too high a level inside the shoe, in which case sand would have to be excavated before the jet process could be resumed. In using this process the men would stir the sand with the jets for about 10 min., and then tighten the jack screws. When sufficient space was obtained between the timbers in the shoe and the curbing above, the jack

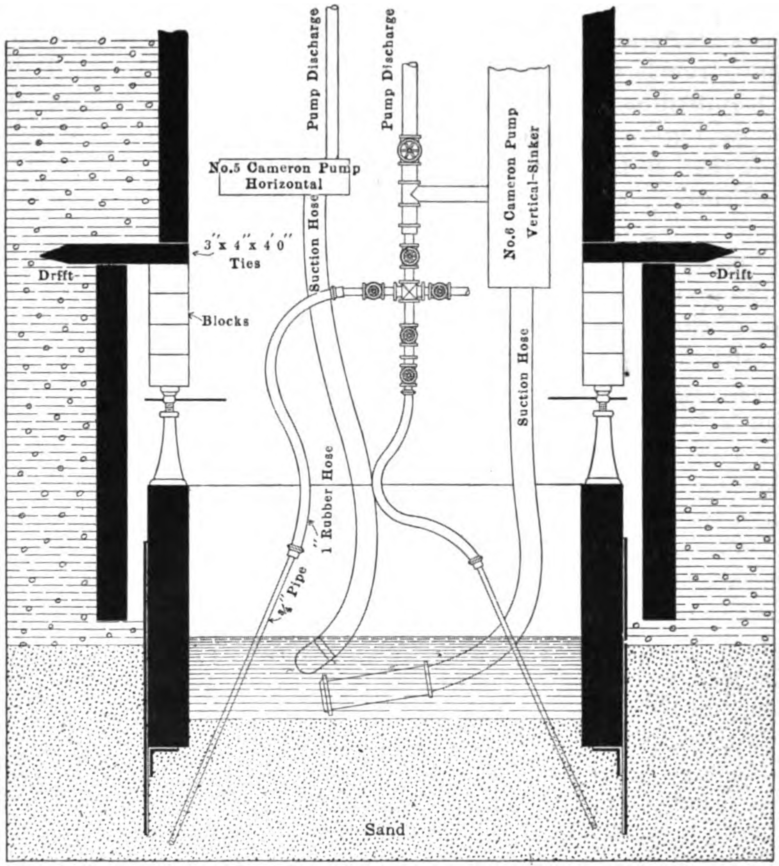


PLATE 2.—DETAILED SKETCH OF SINKING SHOE, JACK SCREWS AND PUMPS.

screws of one side were removed, and sets of timber put in place on top of the timbers in the shoe, and then the other jack screws were moved. An advance of 18 in. per day was exceptional by this method. More often it was less than a foot.

Necessarily some sand had to be excavated at times. This had a tendency to cave the dirt around the shaft, which in turn caused an excessive down pressure and broke the curbing apart a number of times.

An attempt was made to overcome this trouble by supporting the curb with I-beams and cables from the surface as shown in Plate 1, Fig. 3. Ten 12-in. I-beams were supported on cribs at the surface, and ten 8-in. I-beams were swung under the 4 by 6 angle below the ties, these beams being connected by twenty ⅞-in. steel cables. When the sinking was resumed, the curbing continued to break, the I-beams bent, and two cables were broken. Since it appeared impossible to hold the curb, it was decided to timber the shaft solid from the 8-ft. level (where the most uniform break occurred) to the bottom of the shoe, and then drop this portion of the shaft through the remaining 5 ft. of sand.

To do this, the ties were driven back into the wall and solid timbering put in between the shoe and the upper curbing of the shaft. The entire shaft curbing from the 8-ft. level down was then tied together with 2 by 6-in. stringers. At the 8-ft. level, 2 by 4-in. by 16-ft. planks were spiked to the lower curb, the upper ends projecting above the break as shown in Plate 1, Fig. 4, preventing the loose material from falling down the shaft. The jet system, with occasional excavations of sand, was resumed, and the shoe, with 50 ft. of curbing, was lowered through the sand.

In landing the shaft on the solid, seven boulders from 1 to 2 ft. in diameter were encountered. Six of these were under the cutting edge of the shoe, and were removed only after being broken up by means of a long chisel and sledge.

While lowering the shaft through the sand, the upper timbers of the of shaft buckled 18 in. out of line. This necessitated retimbering of the shaft from the top of the sand to the surface, an expensive undertaking because the old timbers had to be cut out and replaced in sections.

On completion of the retimbering, sinking through the shale was commenced. Three shifts of four sinkers each were used with an average daily advance of 5 ft. The only problem involved in sinking through the shale was the elimination of the water, a considerable proportion of which was choked off when the solid was reached. This water was taken care of by placing a water ring at the 85-ft. level with a pump located at that point to elevate the water to the surface as shown in Plate 1, Fig. 5.

### *Air Shaft Sunk by Drop-Shaft Method*

The air shaft was located 350 ft. from the main shaft, and its sinking conditions were similar except that the surface at this point was 10 ft. lower than at the main shaft, making the actual distance to be traversed to the solid 63 ft. 6 in.

The equipment was the same as that used at the main shaft. The air shaft followed a drill hole tapped by an entry from the main shaft, so that most of the water was drained through this drill hole and then

pumped to the surface. Two shifts of three sinkers, one top man, and an engineer were employed.

The size of the excavation was the same as before, 8 by 14 ft. The steel shoe was similar to the one in the main shaft, except that it was 10 ft. high instead of 5 ft. The timbering for the first 30 ft. above the shoe consisted of 4 by 6 laid flat, tied together by lag screws 5/8 in. diameter by 10 in. long, spaced 2 ft. apart. The shaft was divided into three equal compartments by 4 by 6 buntons as shown in Plate 3. The middle com-

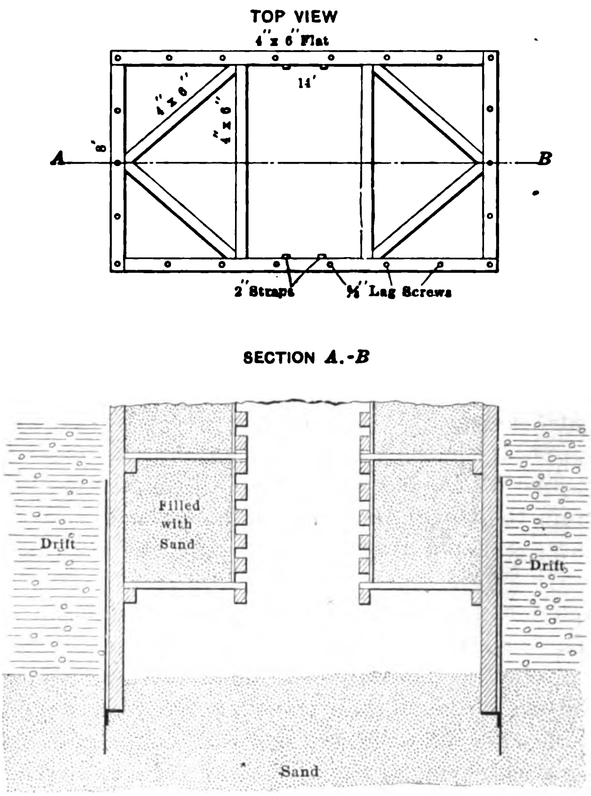


PLATE 3.—METHOD EMPLOYED IN SINKING AIR SHAFT.

partment was left free and was used for hoisting purposes. The end compartments were braced by 4 by 6 extending from the ends of the shaft to the sides. Four steel straps 3/8 in. by 2 in. tied the shaft timbers together from top to bottom.

An excavation 10 ft. deep was first made, the shoe assembled and lined with timbers. The sinking was continued until the shoe was hung up by the friction on the sides. Then a platform was built every 5 ft. in the end compartments. These 5-ft. chambers were filled with sand to give additional weight, and the sinking continued. The ground surrounding

the shoe gradually broke in an oval shape. At one place it was necessary to fire a small charge of powder in order to loosen the ground sufficiently at one end of the shaft.

As the shoe was sunk, timbers were added at the top of the curbing. This method of building the curb at the top is decidedly better than that of adding timbers at the bottom since the timbers are placed much more accurately and expeditiously.

One difficulty experienced in the drop-shaft method was to keep the bottom of the shoe level. When one side of the shoe got lower than the other it kicked the opposite side outward. To right it, the lower side was blocked until the higher side caught up.

The progress through the drift material, until the sand was reached, was slow, much slower than at the main shaft. The drop-shaft went much faster, however, after reaching the sand. In fact, the difficulty at that time was to keep the bottom of the shaft from moving faster than the top. When within 10 in. of the bottom of the sand, the shaft broke apart 20 ft. up from the shoe. This was due to the fact that the movement of the shoe was faster than that of the top of the shaft, and to the insufficient strength of the straps connecting the top and bottom of the curb. At this point (20 ft. above the shoe) the curb separated from 6 to 8 in., and the upper part of the shaft kicked over 9 in. north and east of the lower part. A temporary platform of 8 by 8 timber was put in the end compartments of the shaft at this place, and time given for the upper part of the shaft to settle down before starting the excavation again. The sinking was then continued and the shoe landed on the solid without further difficulty, aside from hitting two small boulders at the bottom of the sand.

As the excavation was larger than necessary for an air shaft, it was decided to cement the shaft for a distance of 28 ft. from the bottom of the shoe, in order to shut off the water. A wall of cement 4 to 8 in. thick was accordingly then constructed.

After the cement was given time to set thoroughly, the excavation was again started in the shale and continued without difficulty to the coal. Sinking through the shale in the air shaft cost slightly more than in the main shaft because work in the mine prevented as careful supervision being given the sinkers in the air shaft.

One difficulty encountered in drop-shaft sinking was in keeping the position of the shaft vertical. At one time this shaft was 2 ft. out of plumb. By regulating the movement at the bottom of the shoe, the shaft partly righted itself, until at the finish, in a total depth of 63 ft. 6 in. to the shale, the bottom of the shaft was 16 in. to the south and 10 in. to the east of the top. Part of this variation was remedied in the cementing of the shaft.

A much larger amount of sand was removed in sinking the air shaft

by the drop-shaft method than in sinking the main shaft. This could be done without danger of a cavity forming, because the surface dirt followed the air shaft down as it descended. When the sinking through the sand was completed, the surface directly surrounding the air shaft had caved to a depth of 15 to 16 ft. and for a distance of 20 ft. in all directions from the shaft. In fact, all the shale that was removed through the remaining 92 ft. to the coal did not fill this space at the surface.

A comparison of the costs of the two shafts is as follows:

| | Main Shaft | | Air Shaft | |
|---|---|---|---|---|
| Labor: | | | | |
| Through drift material | $916.80 | | $789.08 | |
| Through sand | 1,941.80 | | 541.62 | |
| Through shale | 1,065.68 | | 1,213.30 | |
| | $3,924.28 | | $2,544.00 | |
| Superintendence | 600.00 | | 435.50 | |
| Retimbering | 1,343.23 | | | |
| Cementing | ......... | | 207.68 | |
| Total labor cost | | $5,867.51 | | $3,187.18 |
| Materials: | | | | |
| Curbing | $1,878.62 | | $1,195.17 | |
| Supplies | 900.14 | | 642.74 | |
| Power, light, water, insurance, etc | 1,248.50 | | 649.71 | |
| Total curbing, etc., cost | | $4,027.26 | | $2,487.62 |
| Total costs of shafts | | $9,894.77 | | $5,674.80 |

## *Conclusions*

In this particular work there was no question about the superiority of the drop-shaft method of sinking. It made a net saving of $4,300 in the total cost of the air shaft compared with the main shaft. A saving of $2,700 was effected in the labor cost, while in the cost of materials, power, etc., the saving was $1,600. A saving in time also resulted, 30 days being required to traverse the sand with the main shaft, while the air shaft was dropped through it in 17 days.

From the results obtained in these two shafts, and from the experience of others in the western interior coal field, we believe that the drop-shaft method of sinking is the safest, most economical, and most successful that can be adopted for sinking through soft material that lies within 100 ft. of the surface. At greater depths a variation of the method can be used by first sinking a larger shaft close to the soft material, and then telescoping a drop-shaft within it.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Power Plant of the Burro Mountain Copper Co.

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(Arizona Meeting, September, 1916)

THE power plant of the Burro Mountain Copper Co. is located near Tyrone, N. M., at 5,950 ft. elevation. It is interesting because it uses the largest stationary Diesel engines in the United States.

The general layout of the plant is shown in Figs. 1 and 2. The building is of steel-frame construction with hollow-tile walls and corrugated-iron roof; the floor is concrete and all windows have steel frames, making a fireproof building.

The building is made to accommodate one more electric generator unit which is being installed; also two 4,000-cu. ft. air compressors direct-connected to three-cylinder engines with the same size cylinders as the electric units.

The power plant consists of two 815-kva., 60-cycle, three-phase, 6,600-volt, 180-r.p.m. generators, direct-connected to five-cylinder, two-cycle Diesel engines; the necessary circulating pumps for cooling water, cooling towers and oil storage tanks.

A portion of the power is used in the same building as the generating units to run one of two 200-kw. rotary converters delivering 260-volt direct current, and a 2,500-cu. ft. air compressor, delivering air at 95 lb. pressure, direct-connected to a 400-hp., 6,600-volt, three-phase synchronous motor.

The remainder of the power is used to run a 1,500-ton concentrating mill 3½ miles from the power house, various hoists, pumps and motors about the mines and the lighting and water-works of the town of Tyrone.

The direct current is used for electric traction in the mine, also to run hoists and motors that were installed before the new power plant was built.

Due to the large hoists that take starting peaks of 250 kw. and the 10-ton electric locomotive that takes at times 150 kw., the load is extremely variable, as shown in Fig. 3.

The two five-cylinder vertical Diesel engines, which are rated at 1,250 b.hp. at sea level, have cylinders of 525-mm. diameter (20.6 in.) and

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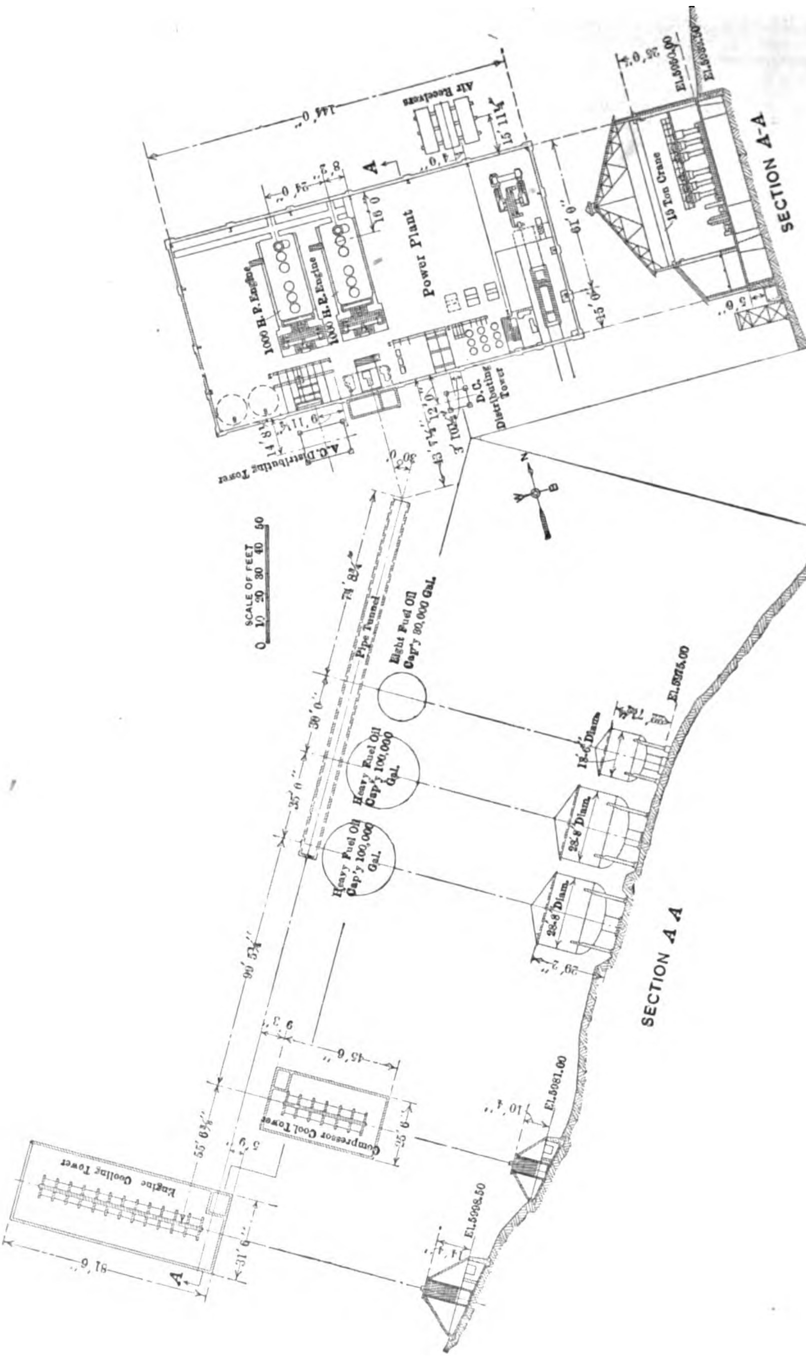


FIG. 1.—GENERAL ARRANGEMENT OF POWER PLANT EQUIPMENT, BURRO MT. COPPER CO., TYRONE, N. M.

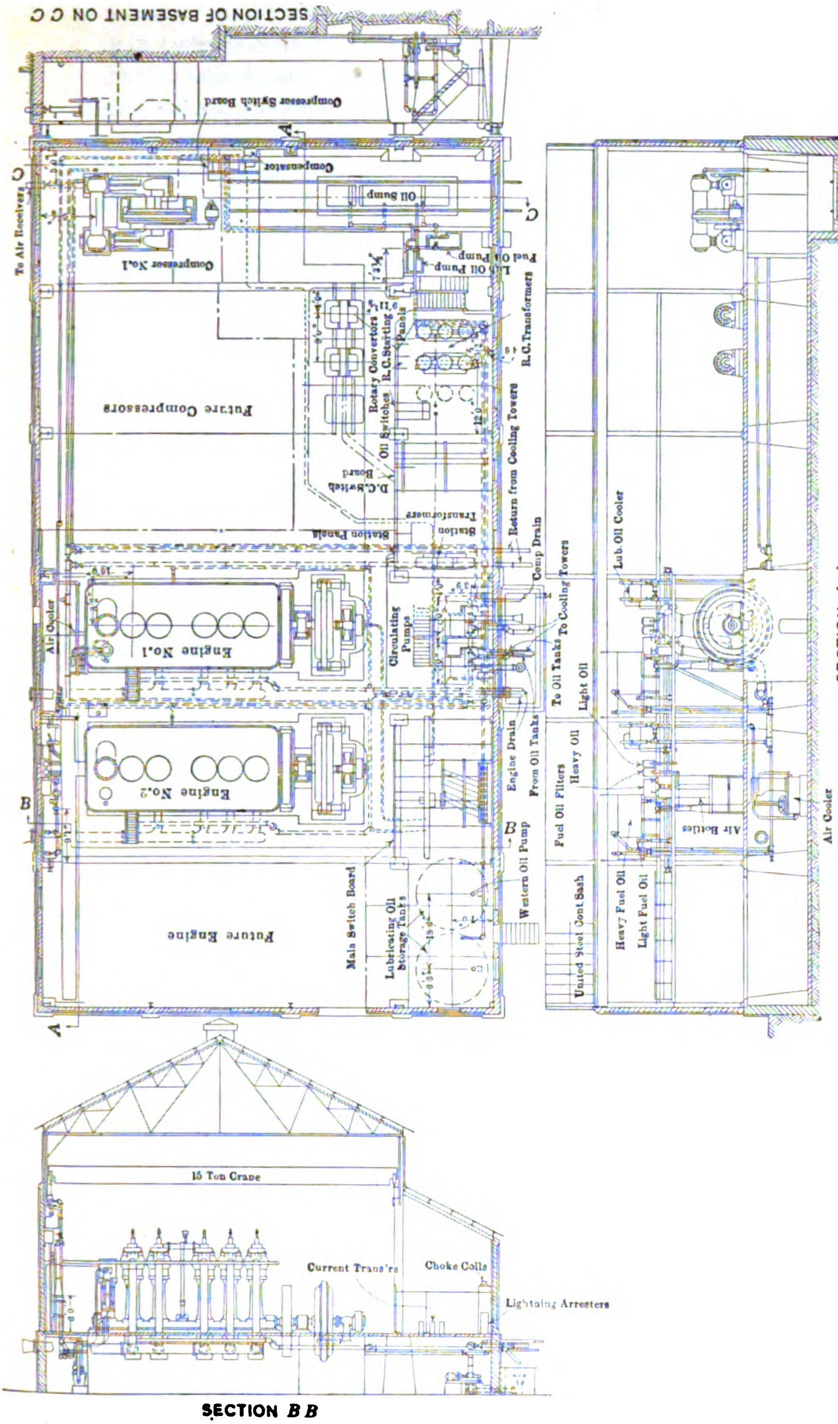


FIG. 2.—GENERAL ARRANGEMENT OF POWER PLANT OF BURRO MT. COPPER CO.

660-mm. (26 in.) stroke; each engine has a scavenging cylinder of 1,050-mm. (41.25 in.) diameter and 600-mm. (23.6 in.) stroke; also a three-stage, four-cylinder high-pressure vertical compressor, both directly connected to the engine. This compressor delivers the air necessary for fuel injection and for starting the engine. The scavenging pump, which is larger than usual, delivers the air to blow off the products of combustion and fill the cylinders with fresh air at the beginning of the stroke This pump was increased in size to be able to fill the cylinders with air at 2½ lb. gage pressure at the beginning of the stroke. This gives nearly the same initial absolute pressure and allows the engine to generate nearly the same indicated horsepower as it would at sea level. The work done in the scavenging pump is, however, increased and the horsepower available is approximately 95 per cent. of sea-level output.

The fuel consumption per horsepower is increased over sea-level conditions because of the extra work of the scavenging pump and the resulting lower mechanical efficiency of the engine when operating at the elevation of this plant.

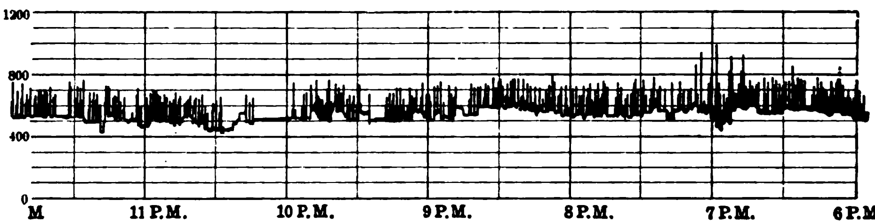


FIG. 3.—SECTION OF CHART SHOWING VARIATION IN POWER LOAD.

The plant has been in operation for 17 months, but as the mill was not ready until the middle of April, 1916, up to that time the load was carried on one engine.

The fuel used is California asphaltum base oil 14° to 18° Bé. gravity, averaging 16° Bé. and 18,360 B.t.u., costing $1.85 to $1.98 per barrel. The number of heat units in the oil was not determined at Tyrone, but at another plant which gets the oil from the same shipping point and under the same contract.

The pistons, cylinders, heads and exhaust pipes are water-jacketed; the jacket water of the exhaust pipes, which can be easily varied in temperature without danger, is used to heat the fuel oil to 120° F. as the heavy fuel oil cannot be used if cold. The engines are started with a lighter fuel oil of about 25° Bé. and run on this oil until the heavy oil is heated to the required temperature; when shutting off an engine, it is run for a few minutes on light oil to fill up the oil piping with light oil.

The curve in Fig. 4 shows the oil consumption per kilowatt-hour delivered at the switchboard when engines were tested new; it is the aver-

age of several tests. It is interesting to compare the test figures with the actual consumption as shown on power reports of the company. Although individual months show variations from the curve, the yearly average for 1915 checks very closely with it, showing that the efficiency of the engines is well maintained and that the variable load has very little detrimental effect on the fuel efficiency.

By the courtesy of Phelps, Dodge & Co., owners of the Burro Mountain Copper Co., I am allowed to publish in the accompanying table the operating costs from Jan. 1, 1915, to June 1, 1916. These costs are as they stand on the books and are higher than is to be expected. They cover the running cost of the complete plant, exclusive of the air compressor. These costs do not include taxes, overhead charges or deprecia-

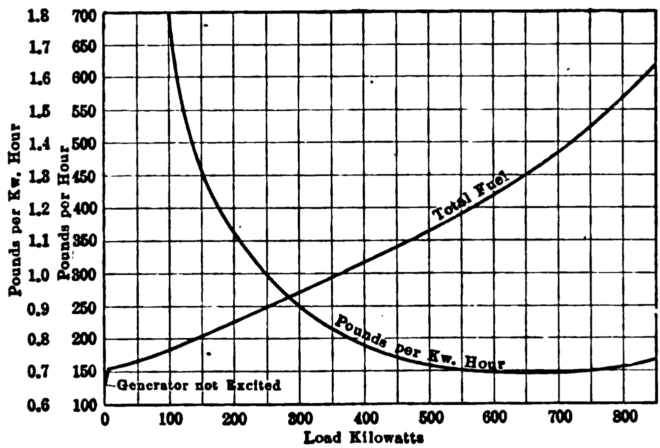


FIG. 4.—AVERAGE FUEL CONSUMPTION TESTS ON DIESEL ENGINE, BURRO MT. COPPER CO.

Diesel Engine: 5 cylinder, 1250 hp. (sea-level rating), 2 cy. ch.
Plant Elevation: 5,950 ft.
Fuel: 16° Bé California crude oil, 18,360 B. t. u. per pound.

tion. The cost per kilowatt-hour available is based on total power generated less that used for auxiliaries in power plant, such as lighting, circulating pumps for oil and water, etc. To simplify bookkeeping, the compressor is not charged with the cost of circulating its water or with its proportion of the lighting.

The labor cost is high because we have operated from the first with the full labor needed for operation when the power house is delivering the power necessary to run the mines and mill at the rate of 1,500 tons per day, a condition that has not yet been reached.

The repair cost is high, for soon after operations were started the construction account was closed and several items, such as labor for installing recording thermometers, measuring gages, some lubricating-

TABLE 1.—*Burro Mountain Copper Co., Cost of Power*

| 1915 | Labor | Fuel Oil | Lubricating Oil | Miscellaneous Supplies | Repairs | Total | Total, Kw.-hr. | Available Kw.-hr. | Cost per Kw.-hr. Available | Pounds Fuel Oil per Kw.-hr. | Load Factor | Average Load |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Jan. | $602.15 | $754.79 | $98.50 | $34.50 | $63.93 | $1,553.87 | 101,315 | 93,322 | $0.01665 | 1.29 | 0.18 | 136 |
| Feb. | 796.60 | 742.50 | 72.60 | 33.31 | 198.44 | 1,843.45 | 77,400 | 69,512 | 0.02683 | 1.68 | 0.15 | 115 |
| Mar. | 782.72 | 859.50 | 80.06 | 50.71 | 179.18 | 1,952.71 | 143,820 | 135,025 | 0.01446 | 1.02 | 0.26 | 192 |
| Apr. | 731.83 | 919.71 | 71.45 | 85.67 | 291.32 | 2,099.98 | 134,473 | 125,795 | 0.01669 | 1.13 | 0.25 | 187 |
| May | 740.86 | 959.00 | 128.86 | 28.26 | 456.64 | 2,313.62 | 130,690 | 122,703 | 0.01880 | 1.24 | 0.24 | 176 |
| June | 705.83 | 882.92 | 142.96 | 29.47 | 393.57 | 2,154.75 | 152,646 | 145,347 | 0.01482 | 1.00 | 0.28 | 212 |
| July | 728.01 | 1,043.44 | 111.90 | 51.56 | 443.64 | 2,378.55 | 159,742 | 151,089 | 0.01574 | 1.16 | 0.30 | 228 |
| Aug. | 762.68 | 1,037.96 | 110.99 | 34.08 | 477.01 | 2,422.72 | 155,800 | 149,397 | 0.01622 | 1.14 | 0.28 | 209 |
| Sept. | 673.05 | 1,046.87 | 122.77 | 15.76 | 507.69 | 2,366.14 | 185,753 | 177,853 | 0.01330 | 1.00 | 0.34 | 253 |
| Oct. | 705.01 | 1,284.25 | 122.66 | 17.97 | 689.58 | 2,819.47 | 275,836 | 267,174 | 0.01055 | 0.81 | 0.49 | 370 |
| Nov. | 692.50 | 1,393.87 | 128.69 | 40.31 | 363.06 | 2,618.43 | 288,394 | 280,323 | 0.00934 | 0.84 | 0.53 | 400 |
| Dec. | 702.45 | 1,492.40 | 132.50 | 9.81 | 316.58 | 2,653.74 | 335,702 | 326,580 | 0.00813 | 0.77 | 0.60 | 450 |
| ........ | 8,623.69 | 12,417.21 | 1,324.48 | 431.41 | 4,380.64 | 27,177.43 | 2,141,571 | 2,044,125 | 0.01330 | 1.00 | 0.33 | 244 |
| 1916 | | | | | | | | | | | | |
| Jan. | $712.15 | $1,527.09 | $124.88 | $30.60 | $285.79 | $2,680.51 | 349,154 | 339,822 | 0.0078880 | 0.764 | 0.63 | 470 |
| Feb. | 694.43 | 1,671.94 | 127.57 | 21.66 | 264.84 | 2,780.44 | 353,119 | 344,553 | 0.0080697 | 0.750 | 0.70 | 520 |
| Mar. | 791.45 | 1,399.03 | 135.93 | 20.60 | 497.61 | 2,844.62 | 370,790 | 362,149 | 0.0078548 | 0.750 | 0.68 | 505 |
| Apr. | 768.65 | 1,897.18 | 174.38 | 21.07 | 574.02 | 3,435.30 | 451,590 | 441,557 | 0.0077799 | 0.780 | 0.58 | 433 |
| May | 961.97 | 2,522.00 | 186.66 | 29.14 | 1,024.58 | 4,724.35 | 559,550 | 546,282 | 0.0086635 | 0.837 | 0.57 | 425 |
| | 3,928.65 | 9,017.24 | 749.42 | 123.07 | 2,646.84 | 16,465.22 | 2,084,203 | 2,034,363 | 0.0080935 | 0.780 | 0.62 | 464 |

oil piping, etc., were charged to repairs. During the first year we also took apart the pistons, valves, etc., of the engine to satisfy ourselves that everything was going right and to determine if possible at what intervals this work would have to be done. This is shown by the relation of repair labor to material used in the repairs. The third engine is being furnished with scavenging valves having removable seats, and we decided to alter the valves of the two engines already erected while we needed only one engine in service.

The cost of repairs is shown in Table 2.

The repairs to circulating pumps and water piping are repairs outside of the engine water piping; the repairs on lubricating-oil system are also the repairs outside of the engine. The item of other repairs covers all repairs to building, switchboard, lighting, fuel-oil system and rotary converters. The latter should not be included under the cost of alternating-current power, but to separate them would have involved a great deal of work and the final result would not be appreciably different.

TABLE 2.—*Burro Mountain Copper Co. Power Plant Repairs*

| 1915 | Engines | | Circulating Pumps and Water System | | Lubricating-Oil System | | Other Repairs | |
|---|---|---|---|---|---|---|---|---|
| | Labor | Material | Labor | Material | Labor | Material | Labor | Material |
| Jan. | $46.40 | $5.21 | $1.50 | $2.82 | 1.10 | ..... | 6.90 | |
| Feb. | 142.95 | 2.61 | 36.10 | 2.49 | ..... | ..... | 11.70 | 2.59 |
| Mar. | 144.30 | 7.67 | 4.60 | ..... | ..... | 0.78 | 18.85 | 2.98 |
| April | 171.05 | 35.46 | 2.70 | 14.08 | 0.35 | 0.48 | 48.25 | 18.95 |
| May | 366.60 | 6.80 | 26.70 | 7.85 | 5.15 | ..... | 42.30 | 1.24 |
| June | 297.50 | 3.52 | 53.80 | 20.71 | 2.58 | 2.58 | 14.50 | 0.96 |
| July | 234.60 | 3.51 | 79.60 | 58.56 | ..... | ..... | 50.10 | 17.27 |
| Aug. | 303.55 | 70.65 | 63.80 | 6.56 | 12.00 | 1.10 | 12.95 | 6.40 |
| Sept. | 223.75 | 7.67 | 70.30 | 40.47 | 33.95 | 11.79 | 95.95 | 23.81 |
| Oct. | 280.80 | 261.37 | 61.15 | 16.43 | 16.85 | 6.15 | 33.30 | 13.53 |
| Nov. | 299.67 | 5.92 | 7.95 | 3.04 | 2.75 | 0.15 | 18.70 | 24.88 |
| Dec. | 229.75 | 51.85 | 21.00 | 0.05 | 0.95 | 0.43 | 8.80 | 3.75 |
| Jan. | 169.00 | 0.34 | 19.70 | 0.10 | 59.45 | 9.54 | 12.20 | 15.46 |
| Feb. | 167.75 | 3.68 | 17.35 | 2.39 | 14.25 | 0.44 | 21.40 | 37.58 |
| Mar. | 284.95 | 152.50 | ..... | ..... | 9.40 | 2.56 | 36.05 | 12.15 |
| Apr. | 74.30 | 3.33 | ..... | ..... | 142.00 | 120.43 | 122.55 | 111.41 |
| May | 320.95 | 483.38 | 8.15 | 0.47 | 80.70 | 29.45 | 52.80 | 48.68 |

The water used in cooling the engines is mine water, which apparently has no bad effect on cast iron, but has a strong action on steel, so that all of our oil filters had to be rebuilt.

The circulating pumps come in for a large repair item due to breakage of check valves above the pumps which fell in and wrecked the pumps.

The two largest items of repairs on the engines were, one main bearing

that was drained of its oil by a mistake of the oiler the first night we operated and one high-pressure air-compressor head that was wrecked by starting the engine without draining the air cylinder, after it had been idle some weeks. A small water leak had filled the cylinder with water and the head broke at the first revolution of the engine.

The greatest trouble we have had, not due to carelessness, is with the helical gears driving the camshaft. These gears have a tendency to get loose and one of them split from the keyway. We have not ascertained the exact cause of this trouble, which is now much reduced since iron bands were shrunk on the hub of the gears.

Two important items of running expense were doubtful when the engines were purchased: One was lubricating oil and the other the maintenance cost. The first one is now settled and we hope to reduce the cost from that shown in the table by the use of separate pumps to each point where the oil is delivered in the cylinders, insuring a definite quantity of oil. The present engines have one pump delivering to four points and it happened once that one of the oil pipes became plugged and gave slight trouble on one piston.

The maintenance is still in doubt, and cannot be determined until we reach and maintain regular operations for some time; but the writer thinks it will show a decrease from the figures shown.

The water-cooling connections to the pistons were expected to give some disagreeable troubles, but we have had practically none.

The operation of the engines in parallel is very satisfactory even when the load is light. In actual operation, with one engine in service, the peak load carried has been higher than was expected to be possible. Under test, 900 kw. was the limit with water-rheostat load, but in service, peaks up to 1,000 kw. have been carried for several seconds without seriously slowing down the engine.

On the whole, the operation of the plant has been satisfactory and the costs obtained are considerably better than with a steam plant of the same capacity run under the same conditions, amply justifying the extra investment for this type of engine where the cost of fuel oil is high.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Basic-Lined Converter in the Southwest

BY L. O. HOWARD, GLOBE, ARIZ.

(Arizona Meeting, September, 1916)

WHAT was perhaps the first attempt at basic converting in the Southwest was made by the late Charles F. Shelby at Cananea early in 1907, when he removed the acid lining from one of the 8 by 12-ft. barrel-type converters then in use at the reduction works of the Cananea Consolidated Copper Co. and substituted a lining of one course of magnesite brick. In this shell, he blew 21 taps (about 180 tons) of matte to white metal before the brick gave out along the tuyère line. In every case, the white metal so made was transferred and blown to copper in other converters. The cost of the lining was $700 and Mr. Shelby gave up the experiment as impracticable. No further serious effort to bessemerize copper mattes in a vessel having a more or less permanent lining of basic brick was made by any of the large copper smelters in the Southwest until some months after Messrs. Peirce and Smith had brought the question permanently before the public, and had proved it to be a metallurgical and financial success by their work at Baltimore and Garfield.

Early in 1911, the Cananea Consolidated Copper Co. had in operation at its reduction works at Cananea, Mex., several 8 by 12-ft. barrel-type converters lined with magnesite brick which had previously been operating with acid linings. In the 18 months following, these converters produced 50,000 tons of copper. In June, 1912, this company installed the first of its new equipment of six stands of electrically operated 12-ft. Great Falls type converters, and shortly afterward discontinued the use of the smaller shells.

In September, 1911, the Consolidated Kansas City Smelting & Refining Co. had installed at its El Paso plant two 10 by 26-ft. Peirce-Smith converters. Later, in March, 1913, one standard Great Falls type converter was added.

About the time the Peirce-Smith converters were going in at El Paso, the Calumet & Arizona Mining Co. was experimenting with basic linings in 7-ft. by 10-ft. 6-in. barrel converters at its plant in Douglas, Ariz. These efforts were sufficiently successful to enable the plant to handle the total output of copper in these small shells until the standard Great Falls converters were substituted in June, 1914.

During the year 1911, the Anaconda Copper Mining Co., at its Great Falls plant, proved the worth of the so-called Great Falls converter. It was to this type, now being recognized as standard, that the Copper Queen Consolidated Mining Co. turned when it was decided to equip the reduction works at Douglas with basic-lined converters. In April, 1912, the first of these shells was put in operation. A second was added during the following month, and these two were quickly followed by six others.

In the meantime, the Consolidated Kansas City Smelting & Refining Co. had in operation, at its Hayden, Ariz., plant, two 10 by 26-ft. Peirce-Smith converters and later, in July, 1913, added one standard Great Falls converter.

The Arizona Copper Co. first used the basic-lined converter in October, 1911, at its old plant at Clifton, Ariz. The acid lining in some of the small converters was removed, and one of magnesite brick was substituted. The use of these small converters was continued until October, 1913, when the new plant with its equipment of three stands of electrically operated standard Great Falls converters was put in operation.

By January, 1913, the Old Dominion Copper Mining & Smelting Co. at Globe, Ariz., had installed a 12-ft. Great Falls type converter. It was the intention to handle in this shell the total output of copper, amounting to possibly 36,000,000 lb. yearly, but later, to insure sufficient converting capacity in case of accident to the larger shell, two of the old 7-ft. by 10-ft. 6-in. acid converters were lined with magnesite brick. It was not the intention, nor has it been the practice, to use these small converters unless absolutely necessary, and to date they have together turned out somewhat less than 1,500,000 lb. of copper.

The Detroit Copper Mining Co. at Morenci, Ariz., with a yearly copper output of about 11,000 tons, has never considered it advisable to adopt the large basic-lined converter. Instead, the 7-ft. by 10-ft. 6-in. acid-lined shells have been relined with magnesite brick and with these some remarkable work has been done, despite the serious difficulties that accompany the bessemerizing of copper mattes in such small basic-lined vessels. A record of over 3,100,000 lb. of copper converted with one lining, before any patching of the brick was necessary, has been made.

The Consolidated Arizona Smelting Co., at Humboldt, Ariz., likewise has continued in the very successful use of the small basic-lined converter, but contemplates the purchase of a large unit in the near future.

The International Smelting Co. at Inspiration, and the United Verde Copper Co. at Clarkdale, Ariz., have installed the standard Great Falls electrically operated converters in their new plants, both of which were put in operation during 1915. Unfortunately, owing to the newness of the latter plant and the fact that a proper working basis had not yet been

**TABLE 1.—*Operating Data of Basic-Lined Converters in Southwest***

| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
|---|---|---|---|---|---|---|---|---|---|---|
| Blast pressure in pounds | 13 | 13.2 | 14 | 12 | 12.4 | 12–15 | 10–14 | 14 | 11.3 | 13 |
| Cu. ft. air per minute per converter | 5,445 | 7,691 | 10,600 | 4,283 | 6,054 | 7,250 | ...... | 8,181 | 4,033 | 4,117 |
| Cu. ft. air per ton bullion produced | 152,449 | 178,411 | 141,000 | 228,116 | 271,577 | 137,582 | 118,328 | 225,674 | 245,975 | 370,566 |
| Cu. ft. air per ton iron and sulphur oxidized | 96,145 | 142,224 | 156,000 | 98,847 | 139,361 | 112,950 | 118,328 | 119,475 | 240,937 | 123,583 |
| Oxygen efficiency, taking into account total iron and sulphur oxidized | 84.0 | 54.1 | 57.2 | 76.8 | 56.3 | 74.0 | 81.0 | 50.6 | 69.8 | 59.0 |
| Total tons iron and sulphur oxidized per stand per month | 2,075 | 1,610 | ...... | 1,647 | 1,903 | 2,868 | 1,991 | 2,576 | 768 | 1,440 |
| Punchers used per shell per shift | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1.5 | 2 |
| Tons bullion per puncher per month | 221 | 204 | 522 | 119 | 478 | 392 | 255 | 137 | 129 | 240 |
| Tons iron slagged per puncher per month | 192 | 153 | ...... | 169 | 533 | 251 | 171 | 156 | 147 | |
| Average time of blow, hours and minutes | 4–00 | 4–11 | 3–09 | 6–44 | 9–20 | 8–00 | 8–30 | 7–12 | 4–51 | 10–00 |
| Average time to blow one ton bullion, minutes | 28 | 23 | 14 | 54 | 45 | 19 | 28 | 28 | 51 | 90 |
| **Average tons bullion per blow** | 8.5 | 10.8 | 13.3 | 8.22 | 12.7 | 25.3 | 18.0 | 15.6 | 5.8 | 6.7 |
| **Average weight of matte per charge, tons** | 21.9 | 26.0 | 25.9 | 33.9 | 40.5 | 60.0 | 50.0 | 51.2 | 19.7 | 26.6 |
| **Tons ore charged per ton of matte** | 0.23 | 0.20 | 0.155 | 0.201 | 0.17 | 0.25 | 0.30 | 0.21 | 0.286 | 0.283 |
| **Tons iron slagged per ton of available silica** | ...... | 2.44 | ...... | 2.45 | 3.85 | 1.75 | 1.80 | 2.36 | 2.03 | 2.60 |
| **Size of converters** | 12 ft. | 12 ft. | 12 ft. | 12 ft. | 12 ft. | 10 by 26 ft. | 12 ft. Gt. Falls<br>10 by 26 ft. P.-S. | 12 ft. | 7 ft. by 10 ft. 6 in. | 9 ft. by 7 ft. 6 in. |
| **Number of tuyères** | 28 | 24 | 24 | 24 | 24 | 35 | 24 Gt. Falls.<br>35 P.-S. | 22 | 12 | 12 |
| **Size of tuyères, inches** | 1¼ | 1¼ | 1½ | 1½ | 1¼ | 1¼ | 1¼ Gt. Falls.<br>1½ P.-S. | 1½ | 1½ | 1¼ |
| **Thickness of brick used on tuyère line, inches** | 24 | 30 | 30 | 14 | 30 | 18 | 18 | 24 | 18 | 15 |
| **Average slag analysis:** | | | | | | | | | | |
| $SiO_2$ | 19.6 | 21.7 | 17.2 | 17.4 | 15.6 | 23.6 | 25.7 | 29.2* | 22.2 | 23.8 |
| FeO | 68.2 | 67.9 | 69.1 | 70.7 | 67.3 | 66.2 | 60.0 | 62.1 | 63.4 | 61.5 |
| CaO | 0.8 | 1.1 | 0.6 | 0.8 | 0.4 | ...... | ...... | 1.6 | 0.9 | |
| $Al_2O_3$ | 3.4 | 3.0 | 3.7 | 3.0 | 2.9 | ...... | ...... | ...... | 2.8 | |
| **Average matte analysis:** | | | | | | | | | | |
| Cu | 37.7 | 43.37 | 51.2 | 28.22 | 34.7 | 43.7 | 42.8 | 31.9 | 35.21 | 25.0 |
| Fe | 32.5 | 29.2 | 21.6 | 30.9 | 34.9 | 28.0 | 27.4 | 28.8 | 36.1 | 46.0 |
| S | 27.4 | 23.5 | 24.5 | 25.3 | 24.9 | 25.1 | 25.6 | 18.7? | 25.0 | 25.0 |

1. Arizona Copper Co., Clifton, Ariz.
2. Old Dominion Copper Mining & Smelting Co., Globe, Ariz.
3. International Smelting Co., Inspiration, Ariz.
4. Calumet & Arizona Mining Co., Douglas, Ariz.
5. Copper Queen Mining Co., Douglas, Ariz.
6. Consolidated Kansas City Smelting & Refining. Co., Hayden Plant, Ariz.
7. Consolidated Kansas City Smelting & Refining Co., El Paso Plant, Texas.
8. Cananea Consolidated Copper Co., Cananea, Sonora, Mex.
9. Detroit Copper Mining Co., Morenci, Aris.
10. Consolidated Arizona Smelting Co., Humboldt, Aris.

* Insoluble.

arrived at, it was impossible to obtain any figures covering the conv operations.

There is no question about the superiority of basic over acid linin where the output of copper is sufficiently large to insure efficient o ing, of the standard type over the smaller basic-lined converter. inability properly to control the temperature of the charge in the vessel, and the resulting destruction of the brick lining, are the serious difficulties with which the operator using a small basic-lined verter has to contend. Throughout the Southwest, the preferenc been for the standard upright rather than for the horizontal conv The greater flexibility of the former, and the greater ease with repairs may be made, strongly recommend it.

The operating data in Table 1 will give an idea of the working ditions and the results obtained at some of the large smelting pla the Southwest. It is to be regretted that replies to questions pert to power consumption were at such variance that it was thought b omit them entirely, as no reconciliation seemed possible.

TABLE 2.—*Old Dominion Copper Mining & Smelting Co.*

Campaign of basic-lined converter No. 2.

| | |
|---|---|
| Put in operation | June 27, 1913 |
| Removed for initial patching | Dec. 7, 1915 |
| Total hours blowing | 13,734 |
| Total number blows made | 3,288 |
| Average time of blow | 4 hr. 11 min. |
| Total number taps matte | 9,316 |
| Total tons matte charged | 85,578 |
| Average weight of matte per charge | 26 tons |
| Average copper content of matte | 43.37 per cent. |
| Total tons bullion produced | 35,431 |
| Average time to blow one ton copper | 23.2 min. |
| Average tons copper per blow | 10.80 |
| Average blast pressure | 13.2 lb. |
| Average air used to blow one ton copper | 178,411 cu. ft. |
| Average air used per minute | 7,691 cu. ft. |
| Average air used per ton iron slagged | 242,502 cu. ft. |
| Total ore fed | 17,097 tons |
| Tons ore fed per ton matte blown | 0.200 |
| Magnesite brick used for repairs | None |

Average Slag Analysis

| | Per Cent. |
|---|---|
| Cu | 2.1 |
| $SiO_2$ | 21.7 |
| Fe | 52.2 |
| CaO | 1.1 |
| $Al_2O_3$ | 3.0 |
| S | 1.8 |

Average Matte Analysis

| | Per Cent. |
|---|---|
| Cu | 43.37 |
| Fe | 29.2 |
| S | 23.5 |

Separate operating data for the Peirce-Smith and Great Falls converters at El Paso could not be obtained, so the figures in column 7 are a combination of results obtained in these different types. The figures in columns 9 and 10 are for former acid-lined shells now operating with a lining of magnesite brick. All figures are for periods of 1 month or more. Those in column 5 are for the year 1915. The data in column 2 cover a period of 29 months' operation—the life of one original lining to the time of first patching. Although some of the figures in Table 2 have already been given, this more complete summary of the campaign may prove of interest.

I am greatly indebted to the managers and superintendents of the different properties who so graciously and promptly furnished me with the necessary information for this article, and take this opportunity to thank them.

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JECT TO REVISION]

ION OF THIS PAPER IS INVITED. It should preferably be presented in person at the ing, September, 1916, when an abstract of the paper will be read. If this is impossible, n in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West New York, N. Y., for presentation by the Secretary or other representative of its author. arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion fter should preferably be in the form of a new paper.

# -Fire Methods Employed by the United Verde Copper Co.

BY ROBERT E. TALLY,* B. S., JEROME, ARIZ.

(Arizona Meeting, September, 1916)

RGROUND fires have been common in the mines of the United pper Co. for the past 22 years. The first fire started in the 300 stope in the fall of 1894, following a cave in that orebody. The highly pyritic nature of the ores is responsible for most of these ice ore characteristics, methods of mining and ventilation, have ant influence on mine fires, they will be described in this paper.

## *Orebodies*

res are mostly chalcopyrite except on the upper levels where ble chalcocite, black oxides and other secondary minerals occur. odies are in the form of lenses and vary in size. Some are small, ers are several hundred feet in length by a hundred or more feet These orebodies extend, in some cases, from the surface to the els of the mine and usually at a steep angle of inclination.

are three different classes of ores, classified in accordance with e rock. The copper in all cases, except as above specified, is rm of chalcopyrite. The largest and most important of these are in a pyrite gangue. Next in order of importance are the s and lastly the quartz porphyries. The pyrite, locally termed average from 30 to 42 per cent. sulphur. The schist and quartz s vary from 3 to 25 per cent. sulphur.

## *Mining Methods*

e orebodies were worked by the overhand square-set method of rior to 1908; since then the cut-and-fill, shrinkage and mill-hole ave been largely employed.

5 per cent. of the total tonnage now extracted is from timbered Most of this is from the upper levels where the ore is soft, heavy or less broken to the surface. Large reserves of these ores are served for more efficient methods of mining.

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* Superintendent of Mines, United Verde Copper Co.

There was no choice in the past with reference to the methods working the upper levels because the smelting plant and other surfa buildings were directly over the orebodies. Only such places were work as were considered necessary and by methods which caused as little se tling as possible. In most of these places it was necessary to use spilir and to fill to the roof. Bulkheads were used on the levels to retain t gangways and were carried up through the stopes for ore chutes.

Top slicing could not be used on account of surface settling and b cause this method is unfavorable, in heavy sulphide orebodies, for t control of mine fires.

The smelting plant and other surface interferences have now be removed and it is planned to start steam-shovel work in the near futur

### *History of United Verde Fires*

The first fire of 1894 was caused by spontaneous combustion, followi a cave. It was not extinguished and the district was bulkheaded fro the remainder of the mine. The soft and broken nature of the grou in this district was such that the smoke and gas worked its way over t orebody on the several different levels and cut off a large productive ar of high-grade ore.

Later, another serious fire occurred in the Chrome or quart porphyry orebodies, which extended over a large area and cut off produ tion from this district.

A third fire followed, cutting off the middle district which lies betwe the Chromes and Hamptons. Many smaller fires occurred in the mea time, which were extinguished. There has never been a serious fire the schist orebodies which join the Hamptons on the south, yet there always danger in that or any other heavily timbered district, where t ore carries any appreciable amount of sulphur.

### *Fire Stopes*

Many attempts were made to extinguish these fires and recover t large reserves of high-grade ore. The districts were first flooded wi water but, owing to the large area involved and the soft, broken-up cond tion of the ground, this method was a failure. Sufficient air alwa found its way through fractures to feed the fire.

The next attempt to extinguish the fire was by $CO_2$ gas. A ga generating plant was installed, pipe lines were laid and large volumes gas were forced into the district, but without effect. Steam was th tried and was likewise a failure. Water, steam or $CO_2$ gas will readi extinguish a fire if the district in which the fire occurs can be tightly seal up and the extinguishing agent brought into contact with the fire

sufficient quantities. Fractures to the surface and underground workings outside the fire district made an impossible condition to seal up.

In 1905, the Plenum system was tried on this district with success. Actual combustion was extinguished and the latent heat reduced to a temperature of about 120°. This work was planned and directed by

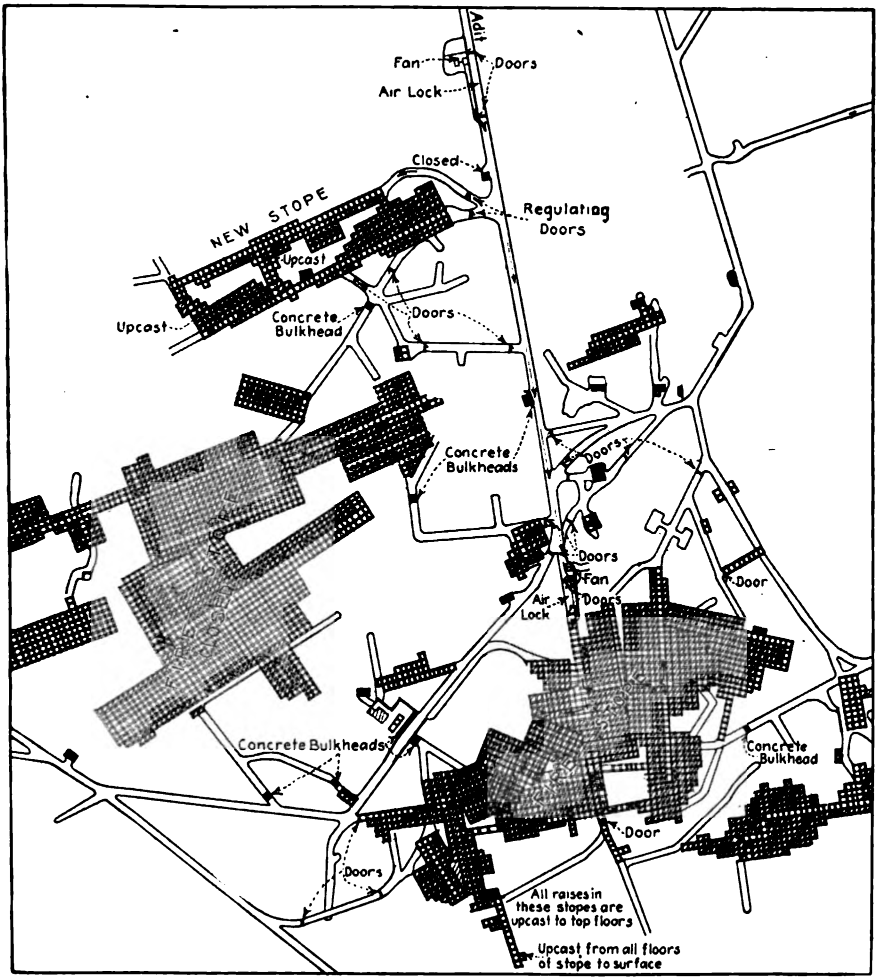



Fig. 1.—Plan of Fire Stopes on 300 Level Showing Method of Ventilation.

Joseph J. Shaw, a mining engineer from the Mountain Copper Co. of Keswick, Cal., where this system was successfully used. It has been used here continually since its inception and consists in forcing the air under a pressure, varying from 2 to 5 lb., into the fire district. The air pressure varies in accordance with the gas pressure and must be sufficient

to keep back the gas and to cool the ground, so that work can be accon plished.

When a bulkhead, in a connection leading to a fire stope, is opene there is invariably an outward pressure and rush of gas or smoke, whic must be overcome. The Plenum system consists in forcing the air again this gas at a pressure slightly greater than that of the gas.

Fig. 1 shows a large fire stope on the 300 level, with doors for cor trolling the pressure, and the method of ventilation. The air is furnishe from the adit and forced under pressure into the fire district. All chut and manways are upcast, and the spent air and gases outlet to the a raise through connections on the different floors.

The installation used with this system consists of a No. 8 double-inl Sirocco fan, direct-connected to a 50-hp. variable-speed General Electr motor; also the necessary airways, doors, etc. The air from the fan mu be under perfect control in order to regulate the gas. This is accon plished by doors, as shown in Fig. 2.

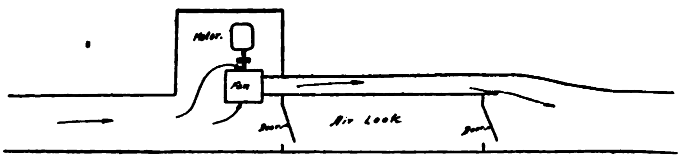



FIG. 2.—SKETCH OF FAN STATION.

An airway about 3 ft. square and from 50 to 200 ft. long is connecte to the outlet of the fan. Doors are placed in the tunnel at both ends c this airway. These doors prevent the fan from drawing its own air, fo when one door is open the other must be closed. All workings betwee the fan and the fire district must be closed off by doors; otherwise the ai pressure will be reduced and trouble result from gas. Two doors ar always used so that one will remain closed when the other is open. Fo electric haulage these doors should be from 100 to 200 ft. apart. Th doors are made of 2-in. grooved lumber, and concrete frames. The ai ways are made of lumber or concrete.

With a variable-speed motor and the proper system of doors, it is a easy matter to increase or diminish the air pressure in order to meet an new conditions within the fire district.

When the bulkheads to this district were first opened, the ground wa red hot. The roof and walls, where opened, were aglow. All timber wa burned and the soft sulphides, where exposed, were burning. The con tinual blowing of air on this hot mass gradually cooled it, so that withi a few weeks the actual extraction of ore was under way.

The timber often took fire before it was blocked, making it necessar to keep it well sprinkled with water. The air reduced the temperatur

00° F. to 120° F. in about 6 weeks. The conditions gradually until the average temperature in the fire stopes was 100° F. ecent years a system of ventilation has been introduced whereby erature has been reduced to about 75°, except in places that are e line of ventilation.

erly the air was forced into these stopes and found an outlet fractures into the other workings and to the surface, with the it none of the upper levels were free from gas.

tem of air raises was driven in the foot wall to the surface and ns from these raises to the stopes were made at different eleva- Doors were placed in these connections and the air outlet thereby d. If an outlet is too free, it will not only release the pressure e gas, but, by suction, will also cause fire. These raises were ground that did not require timbering. Cast-iron sets were re necessary in the connections from the raises to the stopes. hould be avoided if possible in airways, since there is a constant fire in these places.

stopes have now been opened for 11 years and fire occurs on an f about once a month in a rather extensive area of old timbered It is therefore important to have an efficient fire-fighting ion and to have good control of the ventilation. This is an t factor in connection with mine fires. In the first place, every e measure must be used, and, in the second, the ventilation under absolute control in order to extinguish the fire.

cess of air increases the flames in the same manner that an open eases the fire in a stove or furnace. Air must be furnished only nt quantities to carry away the smoke so that the firemen can e fire. Where the smoke is thick it is impossible, even with hel- handle the fire satisfactorily. Many unsatisfactory results in n with mine fires are due to these causes.

### *Causes of Mine Fires*

bly 90 per cent. of all underground fires in pyrite or heavy sul- es are caused by spontaneous combustion. In the fire district ine, many fires are due to red-hot dust dropping on the timbers, ctures in ground above. Some fires are from friction due to ome from defective electric wires or cables; others from burning n contact with timber, and a certain number from incendiarism essness with candles, lamps, etc.

from spontaneous combustion usually occur in the interior of a led stope and are due to the oxidization of fine sulphides in con- timber. Very small amounts of air are necessary for spontane- bustion and this air works its way through the filling along the ulkheads, timber or walls, where it comes into contact with sul-

phide dust. In the process of oxidization this produces sufficient heat to start a fire. This theory has been proven here by experiments. Heavy sulphides should not be used for filling in timbered stopes. Considerable fine sulphide ore is lost in the fire stopes or in any other heavy ground where the square-set method of mining is used and where the floors are kept filled to the roof.

### *Prevention of Mine Fires*

Careful consideration should be given toward making heavy sulphide mines as fireproof as economy will permit.

Main hoisting shafts in such properties should be constructed of concrete or other fireproof material. Where timber is used, a sprinkling system should be installed.

Mine methods should be developed in which timber will be eliminated as far as possible.

Heavy sulphide material should not be used for filling in the timbered stopes. In these places fine waste should be used and sprinkled so that it will pack and make conditions unfavorable for fire.

In fire stopes or stopes adjacent to the fire district, the timber next to the walls or ends should be removed.

Careful attention should be directed toward ventilation so that the temperature will be unfavorable for fires to start.

In hot places the timber should be kept damp by sprinkling with water. Water should never be put on red-hot or burning ground, as this will invariable cause an explosion, one of the great dangers in handling mine fires.

Water and air lines, hose and hose connections and all tools used in connection with mine fires should be kept convenient and in good working condition where fire is liable to occur.

Helmets, pulmotors or lungmotors, electric lamps and supplies should be kept in stock in sufficient quantities and convenient for immediate use.

Carefully selected men should be trained in the use of the helmets and resuscitating machines, and in the methods of fighting fires. The average man, unless specially trained, is useless in fighting fires. The foremen, shift bosses, electricians, pumpmen, watchmen and selected miners working in different places which are subject to fires should familiarize themselves with all things appertaining to the ventilation, and should know just what to do if fire occurs in their respective places. Rules to this effect should be printed, distributed, and understood by all concerned.

Watchmen should inspect the air outlets often and regularly in order to detect smoke as soon as possible.

Iron fire doors should be erected in connections near all timbered stopes, so that in case a fire gets beyond control the doors can be closed and the district quickly sealed up, in order to protect the remainder of

Similar doors should also be erected near timbered shafts, air-
., for protection against fires.

signal should be in use at all mines and understood by all em-
A practice signal should be given occasionally. The following
l is in use here:

**e of fire, ring nine bells on the electric cage call signal. The engineer on hese bells will flash all electric light throughout the mine, nine times. This e repeated three times and followed by flashing the station signal on which e is, three different times. Carmen and all others working where there are its will notify those employed in stopes and other places where there are The trained firemen on the various levels will then take charge of the Their first consideration will be for the safety of the men and then for ishing of the fire and protection of property."**

a fire occurs in the fire district no signal is given, for the system uch that the fire can be easily confined without danger from gas to the other parts of the mine.

fire occurs in any other part of the mine, the signal is given and orking where there is danger from suffocation are instructed to surface or other places of safety. The next step is to get water r the fire as quickly as possible. When the fire is in the interior b, it is sometimes necessary to drift from a chute or manway he filling in order to get at it. Often, however, the fire can be ned by water from above. This is usually accomplished by oles in the filling with a pinch bar and turning water into these or this work, hose nozzles made of 1-in. pipe with about a 3/8-in. sed. By this method, water can be scattered all over the stope. ep the fire from spreading and usually extinguish it.

e a fire gets beyond control, the fire doors should be immediately ter which concrete bulkheads should be erected. It is some-sible to extinguish a fire in a sealed-up stope by water, through d or churn drill hole. In sulphide stopes, however, it is usually to burn the timber before the stope can be recovered. A fire, arently extinguished, may start again when the bulkheads are nless the timber is consumed.

present time one large fire district here is being experimented g these lines and with apparent success. Air in small amounts lly forced into the stopes to burn the timber. This causes the settle, and, to avoid caving, more waste is put in from above.

### *Ventilation*

ation has received considerable attention here during the past not only in the fire district but throughout the entire mine. mines are naturally warm and disagreeable unless ventilated. ventilation is used where possible; otherwise, fans are used to

blow the air where it is required. The ventilation of the fire stopes inter-feres with the remainder of the mine, since the air is taken from its natura source and forced into the fire district. Artificial ventilation is used fo the middle and lower levels.

Fig. 3 is a plan of one level showing the location of the fan and th method of ventilation. All levels are connected to the air raise.

Fig. 4 is an ideal vertical section of the orebodies, showing the direc tion of the air currents, and the methods of ventilation.

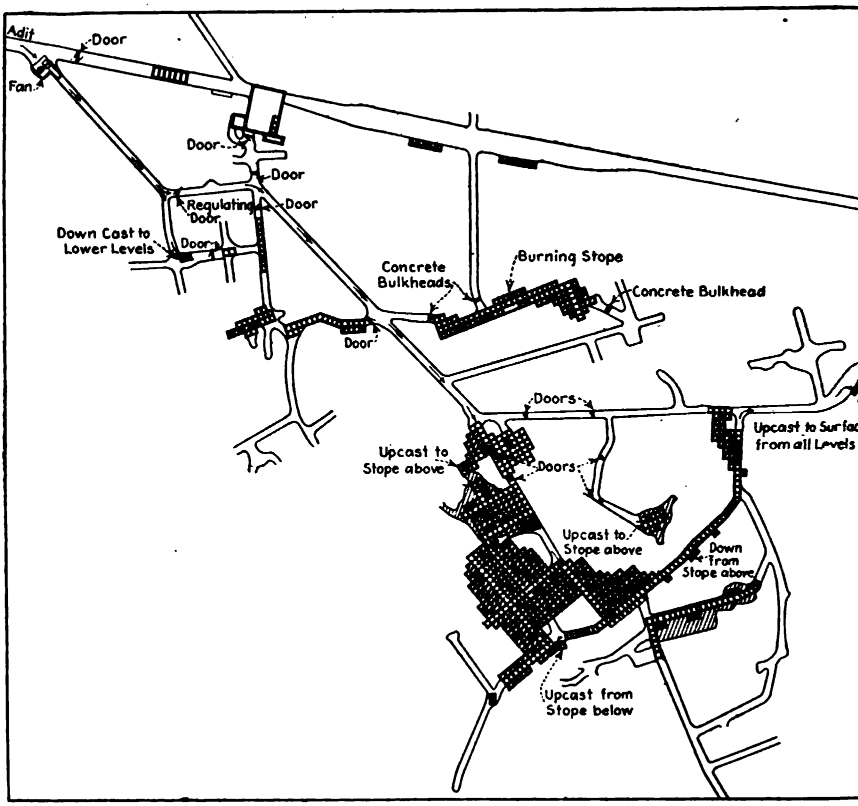
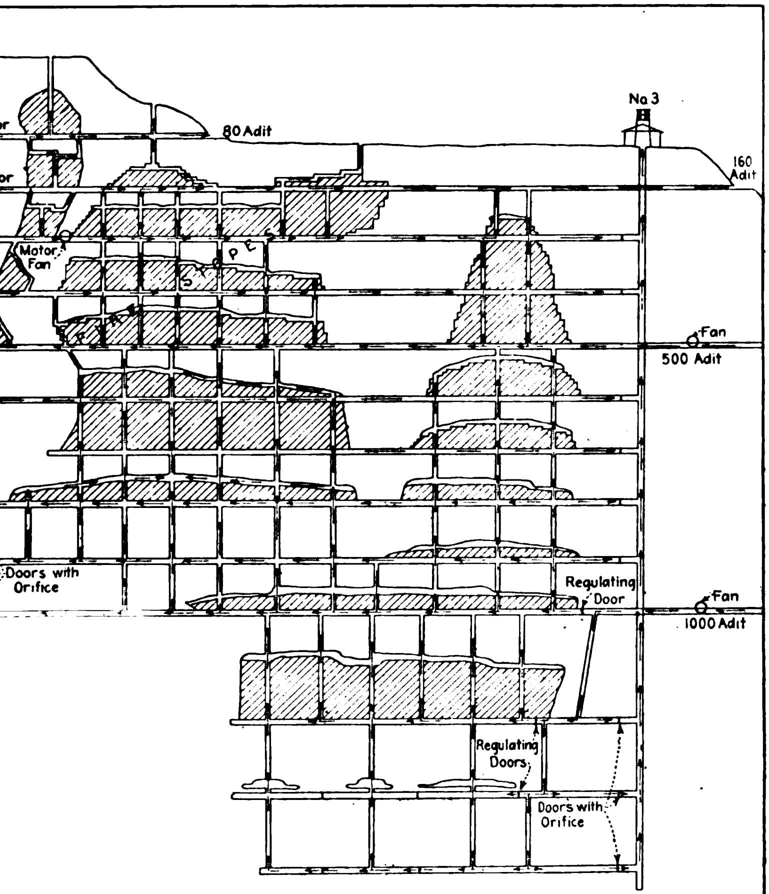



FIG. 3.—PLAN SHOWING LOCATION OF FAN AND THE METHOD OF VENTILATION.

Fans are installed in suitable places where they can draw air from th outside and force it into the different workings, thence to the air raise to the surface. The same airways and doors are used as for the fire dis-trict, except that no particular pressure is maintained, the air having a free an outlet as possible to the surface. The ventilation for any particu-lar place or section is regulated and controlled by doors. For the prope ventilation of stopes, the air should have an inlet at one end and an outle at the other. Most air is forced into the main working places and only a

amount is sent to the old workings to preserve the timber and
nditions unfavorable for fire.

maximum temperature in the middle and lower levels is 80°F.
from 60 to 80° F. and averages about 72°, with small amounts
lity.




VERTICAL SECTION SHOWING VENTILATING SYSTEM OF UNITED VERDE MINE, JEROME, ARIZ.

## *Conclusions*

andling mine fires satisfactorily, the Plenum system as employed
been a marked success, but the mining methods in connection
h are necessarily expensive and inefficient.

fires are the most disagreeable and dangerous conditions that
untered in metal mining and every possible precaution should be
l to prevent them. In addition to preventive measures, there
e carefully trained firemen who thoroughly understand how to
nd extinguish fires.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Cyaniding Clayey Ore at the Buckhorn Gold Mine

BY PAUL R. COOK,* B. S., GUAYAQUIL, ECUADOR

(Arizona Meeting, September, 1916)

THE ore deposit of the Buckhorn Mines Co., Buckhorn, Nev., is peculiar in being a shallow kaolinized mass of material with basalt walls, and having apparently no direct connection with any of the usual gold-bearing rocks. The average ore contains 16 per cent. water of hydration, and the cyaniding of this hydrous clayey material offered unusual difficulties as compared with the typical gold quartz ores of Nevada.

The orebody was thoroughly developed; then the mill was built according to the latest cyanide practice, with such changes as were thought to be demanded by the peculiar nature of this ore.

Upon starting the mill, the ore proved more difficult to handle than had been anticipated. It is hoped that an account of how these difficulties were met may prove of interest to anyone having a clayey ore to handle and to the profession in general.

## GEOLOGY

The Buckhorn orebody lies along a north and south fault plane of perhaps 1,000 ft. dislocation, that can be traced for miles; but the only other known mineralization consists of similar ore in the Murphy mine, a mile farther north.

The east or hanging wall is hard and smooth, being a typical fault plane. The best ore is along this wall, gradually grading down toward the west until at 30 to 60 ft. it is too low-grade to mine. The country rock on the west consists of alternating layers of hard and soft basalt and basalt scoria, pitching toward the mine.

One of these basalt layers on the hillside a little above the mine is marked, for 3 or 4 miles in length, by a line of springs which seep out along it. Perhaps the meeting of the surface drainage, passing down these basalt layers, with the fault-plane solutions explains the formation of the Buckhorn orebody.

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* With Buckhorn mines during the first year of mill operation (December, 1913, to December, 1914).

Beneath the oxide ore a smaller body of almost pure marcasite occurs with about the same assay value ($8 per ton). Beneath the 250-ft. level there appears to be no further mineralization.

## Mining

The first difficulty was to get the ore out of the mine. Since the orebody extended from the grass roots to a depth of 175 ft., with a width of 50 to 80 ft. and a length of 1,400 ft., the "glory-hole" system of mining was adopted. The ore was broken through 8-in. grizzlies into 18 chutes extending from the surface to an intermediate level, on which it was trammed by hand to three chutes leading to the 200-ft. level, and hauled by electric motors through a 1,000-ft. tunnel to the mill.

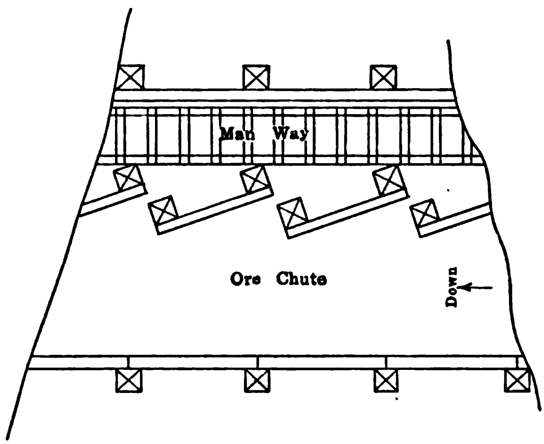



FIG. 1.—SECTION THROUGH ORE CHUTE.

In the summer little trouble was experienced; but the ordinary rain and snowfall of a Nevada winter made it almost impossible to keep the chutes open, as first built. After reconstructing, so as to provide an opening for barring at each set of timbers (see Fig. 1), it became possible to keep the mill supplied with ore during all kinds of weather. During wet weather all the cars had to be scraped with a shovel after dumping, and unusual care was required to keep a sticky load from carrying car and all into ore bin or over the dump; 30 per cent. of the tonnage mined went to the waste dump.

## Milling

### *Ore Bin and Crusher*

The next problem was to get the ore out of the mill bin and crushed. The bin was an ordinary circular steel bin, with natural earth bottom and

side gate. This ore absolutely refused to run from the bin. The mill was built to treat 300 tons a day, but even with one man in the ore bin, and two at the crusher, it was impossible to get over 150 tons through in 24 hr. The large kaolin lumps gave the most trouble in crushing. They had to be practically chiseled to pieces and poked through the jaw crusher by hand.

The replacement of the jaw crusher with a high-speed toothed roll (see Fig. 2) gave the desired crushing capacity. This machine was developed at one of the Bingham Canyon (Utah) properties, and is manufactured by a Salt Lake firm. It is well adapted for sticky ores. The Buckhorn ore contains an occasional "nigger head" of very hard "malapi" or basalt. The mill crew was afraid one of these would break off the teeth of the crusher shell, and they were very carefully picked out at first,

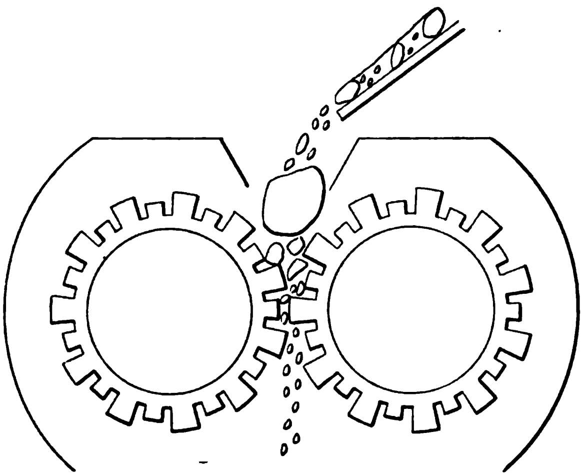


FIG. 2.—SECTION THROUGH WALL CRUSHER SHELLS.

but after a few had gone through by accident, and a few pieces of drill steel, etc., had been chopped up, no further concern was felt. All the boulders that had been sorted out were later put through as it was found that the easiest way to keep the teeth free of adhering clay, was occasionally to throw in a boulder of hard rock.

To do away with the necessity of a man on each shift to shovel the ore out of the bin, a 36-in. conveyor belt was installed to feed the crusher automatically. The opening in the bottom of the bin, over the belt, was about 2 ft. wide, and extended clear across the bin. It was closed by means of short pieces of mine rail that could be removed a few at a time, permitting the ore to be drawn from any point desired. The belt was driven from the crusher line shaft by means of ratchet and dogs. It could be started or stopped, from either floor of the crusher building, by means of a rope connected with the dogs. With these

improvements, crushing required only a part of one man's time. Two men each shift attended to crusher, rolls, two ball mills, two classifiers and two tube mills.

### *Rolls*

The 45 by 15-in. Anaconda-type rolls with smooth shells would clear themselves fairly well, if one of the shells had a channel about 1 in. wide by ½ in. deep machined in it, but it was troublesome to keep a groove in the shells as they wore down. A corrugated or toothed shell would have been better for this ore.

### *Ball Mills*

One 6-ft. Hardinge ball mill was intended to handle the whole tonnage. After plastering the balls to the side of the mill with clay, a few times, the shiftmen learned to tell by the sound of the mill, when it was beginning to coat up. By shutting off the feed at this time, it took only a few minutes for the coating to be ground out.

The installation of a duplicate ball mill made it possible to keep the rest of the plant going while grinding out the ball mills, one at a time, and allowed the rolls to be set coarser on troublesome ore. With a good run of ore, 300 tons per day was sometimes put through one mill.

### *Classification and Tube Milling*

This ball mill discharge was classified in two 36-in. Akins classifiers, the sand from which was fed to two 5 by 18-in. tube mills with Komata liners. The tube-mill discharge was classified in a home-made drag classifier. The small percentage of material requiring regrinding consisted largely of fragments of the basalt "nigger heads." This material was almost as hard as the pebbles themselves and of low assay value. Occasionally it accumulated in the circuit enough to be troublesome and was thrown away. A small amount of it was thought to help the grinding. About 80 per cent. of the product delivered to the treatment plant would pass a 150-mesh screen.

### *Agitation*

About 80 per cent. of the mill-head value was dissolved in the crusher plant. Only a trifling additional extraction could be obtained either in the mill or experimentally. The real trouble was to remove the dissolved value from the clayey pulp. Accordingly the three 32 by 14-ft. Dorr agitators were changed to thickeners.

THICKENING

e three "converted" agitators settled 300 tons per day of 1 to as delivered from crusher plant, to a specific gravity of 1.15. . ft. of settling area provided per ton of this ore settled in 24 hr. e sufficient to settle an average Nevada quartz ore to a specific of 1.33. The overflow was precipitated, and the underflow mixed barren solution and fed to six 36 by 12-ft. Dorr thickeners, g a 1.23 specific gravity underflow to the filters. The 20 sq. ttling area per ton settled in 24 hr. is three times the area to settle an average Nevada quartz ore to a specific gravity Primary thickeners were held with 2 ft. of clear solution; the y thickeners with 6 in. It was impossible to settle the raw n ore beyond a specific gravity of 1.26, either in the mill or ntally.

*Filtering*

maximum capacity of each of the four 14-ft. diameter by 12-ft. er filters was 50 tons per day, about one-half their capacity on a quartz ore. An additional filter, 14-ft. diameter by 24-ft. face, e installed to handle 300 tons per day.

*Dehydration*

nple of Buckhorn ore carefully dried at a temperature below 110° specific gravity of 1.9. A higher temperature gave an additional 3 per cent. in weight, and entirely changed the physical properties re. The dehydrated sample had a specific gravity of 2.4, and nd filtered almost as well as a quartz ore. Dehydrating also the sticky milling qualities. Both samples, however, gave the raction with cyanide. The temperature of a laboratory electric e was sufficient to dehydrate a sample nicely. As $CO_2$, etc., would riven off at this temperature, this loss in weight must be due to hydration.

a cheap fuel supply, dehydration before milling would be the best it for this class of material. The ore would mill and classify sily; the thickeners and filters would have normal capacity; and d values would be more completely removed. The temperature mercial drier would dehydrate the ore with about the same fuel otion (100 lb. of coal per ton of ore) as in removing the 18 per $H_2O$ if it existed in the form of moisture.

high price of fuel delivered at Buckhorn prevented the adoption dration at this mill. The ore was milled raw at the cost of $1.59 (see Table 1). Power cost $8 per horsepower per month.

The careful drying at a temperature below 110° C. of a large number of samples, with the equipment of an ordinary cyanide plant assay office, would be a rather tedious operation; so the regular moisture and assay samples at Buckhorn were dehydrated. All assay, moisture, tonnage, etc., figures are on this basis.

To compare with other ores the figures obtained by drying below 110° C. should be used. Both sets of figures are given in Table 2. Table 3 shows the mill flow sheet.

TABLE 1.—*Buckhorn Mining and Milling Costs*

Ore Milled, 10,000 Wet Tons, 8,100 Dry Tons, $H_2O$ 19 Per Cent.

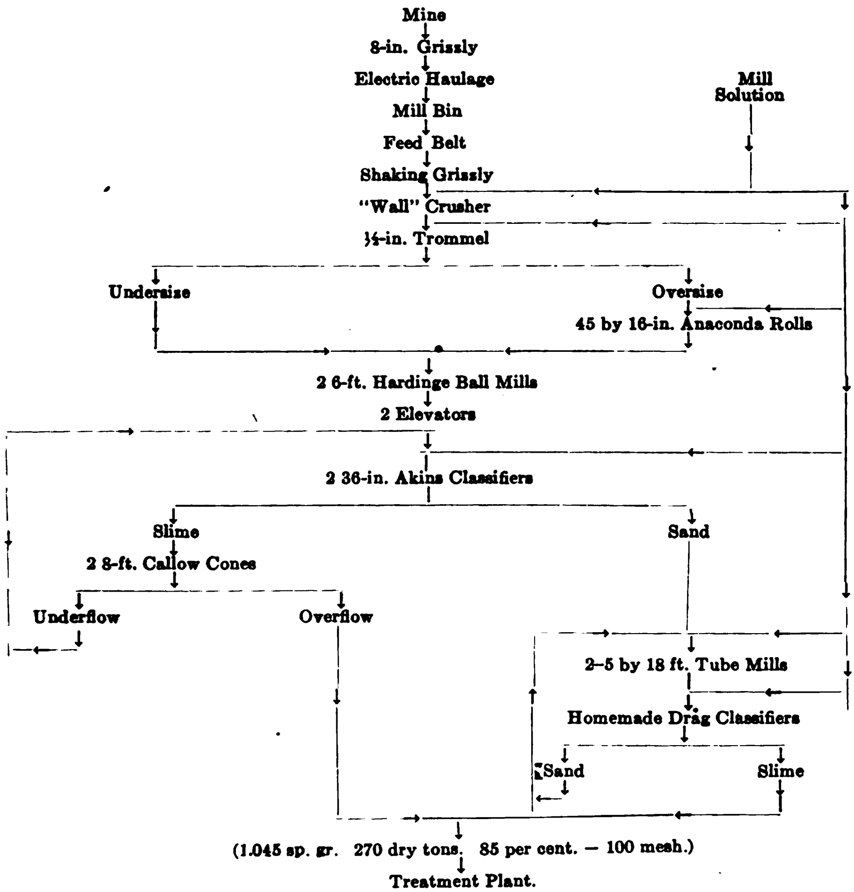
| | Per Ton | |
|---|---|---|
| **MINING** | | |
| *Ore Breaking:* | | |
| Labor | $0.259 | |
| Supplies | 0.098 | $0.357 |
| *Tramming 100-ft. Level:* | | |
| Labor | 0.044 | |
| Supplies | 0.002 | 0.046 |
| *Timbering* | .... | 0.013 |
| *Electric Haulage:* | | |
| Labor | 0.084 | |
| Supplies | 0.006 | |
| Power | 0.022 | 0.112 |
| *General Expense:* | | |
| Surface drainage | 0.004 | |
| Haulage tunnel repairs | 0.039 | |
| Assaying and sampling | 0.027 | |
| Surveyor | 0.011 | |
| Superintendence | 0.067 | |
| Incidentals | 0.024 | |
| Development | 0.083 | |
| Overburden and waste | 0.174 | 0.429 |
| *Grand Total* | .... | $0.957 |
| **MILLING** | | |
| *Crusher and Rolls* (Wall-toothed Roll and 45-in. Roll): | Per Ton | |
| Labor | $0.057 | |
| Supplies | 0.003 | |
| Power | 0.034 | $0.094 |
| *Hardinge Ball Mills* (Two 6-ft. Mills): | | |
| Labor | 0.050 | |
| Supplies | 0.026 | |
| Power | 0.076 | 0.152 |

| | | | |
|---|---|---|---|
| *ing and Separating* (Two 36-in. Akins Classifiers): | | | |
| abor | 0.006 | | |
| upplies | 0.001 | | |
| ower | 0.002 | 0.009 | |
| *Milling* (Two 5-ft. by 18-ft.): | | | |
| abor | 0.011 | | |
| upplies | 0.049 | | |
| ower | 0.092 | 0.152 | |
| *tion* (Three 32-ft. by 4-ft. Dorrs): | | | |
| abor | 0.028 | | |
| upplies | 0.006 | | |
| ower | 0.027 | 0.061 | |
| *icals:* | | | |
| yanide | 0.218 | | |
| ime | 0.293 | | |
| ead acetate | 0.024 | 0.535 | |
| *nuous Decantation* (Six 35-ft. by 12-ft. Dorrs): | | | |
| abor | 0.004 | | |
| ower | 0.017 | 0.021 | |
| *ing and Discharging* (Six 14-ft. by 12-ft. Olivers): | | | |
| abor | 0.062 | | |
| upplies | 0.045 | | |
| ower | 0.088 | 0.195 | |
| *pitation:* | | | |
| abor | 0.025 | | |
| upplies | 0.067 | | |
| ower | 0.019 | 0.111 | |
| *n Pumping:* | | | |
| abor | 0.009 | | |
| ower | 0.012 | 0.021 | |
| *ing:* | | | |
| abor | 0.023 | | |
| upplies | 0.021 | | |
| ower | 0.001 | 0.045 | |
| *ying and Sampling:* | | | |
| abor | 0.018 | | |
| upplies | 0.008 | 0.026 | |
| *rintendent and Foremen* | | 0.090 | |
| *riments* | | 0.010 | |
| *al Expense* | | 0.043 | |
| *r Supply* | | 0.021 | |
| *d Total Milling* | | | $1.586 |
| *d Total Mining* | | | 0.957 |
| *d Total Mining and Milling* | | | $2.55 |

TABLE 2.—*Milling Data*

| | Figures Obtained by Dehydrating Samples | | Figures Obtained by Drying Below 110° C. | | Comparative Figure for Quartz, 2.7 Specific Gravity |
|---|---|---|---|---|---|
| | $H_2O$, Per Cent. | 2.4 Specific Gravity, Dry Tons | Moisture, Per Cent. | 1.9 Specific Gravity, Dry Tons | |
| Ore milled per month, 10,000 gross tons | 19 | 8,100 | 4 | 9,600 | 2 per cent. $H_2O$ |
| Ore milled per day, 333 gross tons | 19 | 270 | 4 | 320 | 2 per cent. $H_2O$ |
| Ball mill discharge, specific gravity, 1.439 | 48 | 270 | 36 | 320 | |
| Tube mill discharge, specific gravity, 1.394 | 52 | 234 | 40 | 279 | Specific gravity 1 64; moisture, 3[illegible] per cent. |
| Akins classifier, sand product | 38 | 175 | 25 | 208 | Moisture, 24 pe[illegible] cent. |
| Slime to treatment plant, specific gravity, 1.045 | 1 to 11.5 | 270 | 1 to 10 | 320 | 1 to 7, specific gravity, 1.08 |
| Primary thickeners underflow, specific gravity, 1.15; settling area, 8 sq. ft. per ton | 1 to 3.5 | 270 | 1 to 2.57 | 320 | 1 to 2, specific gravity, 1.26 o[illegible] better. |
| Secondary thickener underflow, specific gravity, 1.23; settling area, 20 sq. ft. per ton | 1 to 2.1 | 270 | 1 to 1.5 | 320 | 1 to 1, specific gravity, 1.46 or better. |
| Filter cake | 43 | 270 | 30 | 320 | Moisture, 29 pe[illegible] cent. |

TABLE 3.—*Flow Sheet.*



[TRAN]SACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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[DISCUS]SION OF THIS PAPER IS INVITED. It should preferably be presented in person at the [Arizona Mee]ting, September, 1916, when an abstract of the paper will be read. If this is impossible, [discussi]on in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West [39th St.,] New York, N. Y., for presentation by the Secretary or other representative of its author. [Unless speci]al arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion [offered there]after should preferably be in the form of a new paper.

# [Po]ssibilities in the Wet Treatment of Copper Concentrates

BY LAWRENCE ADDICKS,* B. S., NEW YORK, N. Y.

(Arizona Meeting, September, 1916)

[At th]e San Francisco meeting of the Institute last year, I presented, [through] the courtesy of Dr. James Douglas, some results of experiments [on r]oasting and leaching of concentrator tailings. After it became [evident] that flotation rather than leaching was clearly the better [means] of handling the particular problem under consideration, before [dismantl]ing the experimental equipment, some data were secured on [roasting] and leaching the concentrate itself in competition with smelting. [Some of] the results obtained are given in the following paper and they are [of partic]ular interest at this time in view of the large quantities of flota[tion con]centrates with their somewhat difficult smelting characteristics [now bein]g produced.

[The co]mplete wet process† consists of roasting and leaching the calcines [with] sulphuric acid produced from the roaster gases, roasting the [residue w]ith salt and leaching with dilute tower liquors (the well-known [Longmai]d-Henderson process) and recovering the copper, silver and gold [by precipi]tation or electrolysis or a combination of both. It is evident, [however], that the residue from the first leaching, carrying about 20 per [cent. of t]he copper and all of the silver and gold, can be smelted if pref[erred.] In considering the application of the scheme to individual [cases, it] must be remembered that freight plays a large part in any [compariso]n process wherein smelting is not conducted at the mouth of the [mine, an]d that it is not practicable today to build small smelting plants [for indiv]idual operations.

[The e]xperiments may be grouped under four main heads: Roasting, [leaching,] chloridizing residue, and recovery of copper from solutions. [The pro]ducts of two concentrators were used: The Nacozari concentrates [being the] product of a large modern mill not using flotation, the copper [mineral] being largely chalcopyrite; and the Tyrone concentrates, the [product] of an experimental mill including flotation, the copper mineral [being chi]efly chalcocite. Typical analyses would be as follows:

---

[*] Consulting Engineer. † Patent applied for.

| | Nacozari | Tyrone |
|---|---|---|
| Copper, per cent | 14.0 | 14.0 |
| Silver, ounces per ton | 4.0 | 0.5 |
| Gold, ounces per ton | 0.01 | Trace |
| Iron, per cent | 31.0 | 28.0 |
| Sulphur, per cent | 34.0 | 30.0 |
| Silica, per cent | 13.0 | |
| Alumina, per cent | 3.0 | |
| Lime, per cent | 0.6 | |

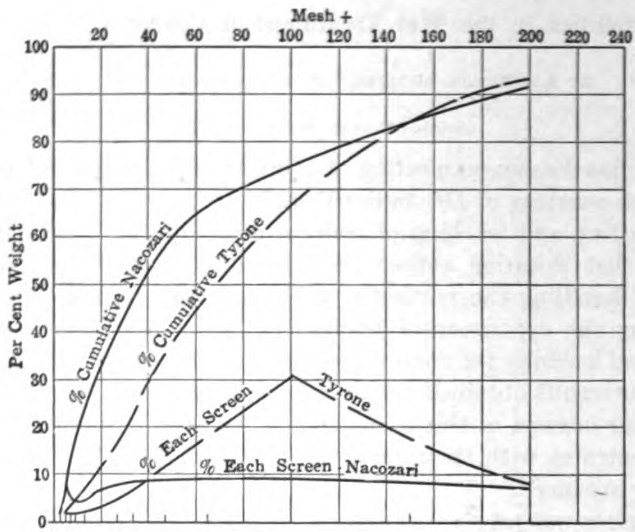



FIG. 1.—SCREEN ANALYSES OF CONCENTRATES CALCINES.

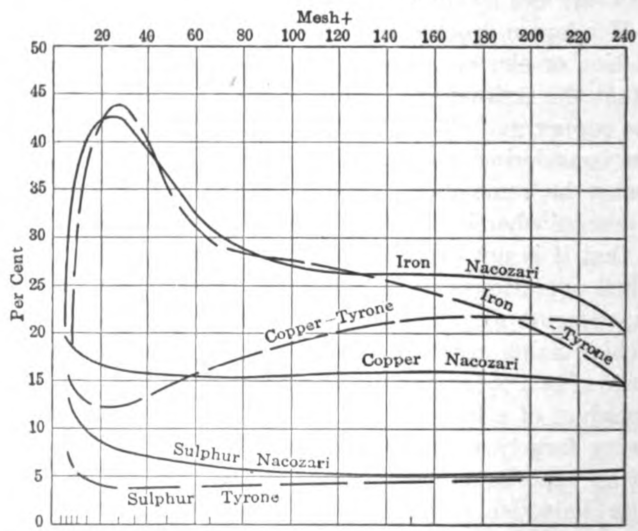
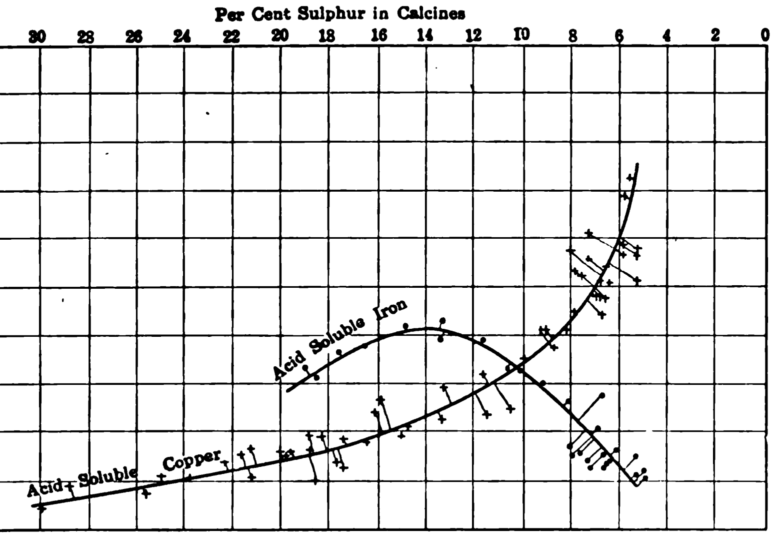



FIG. 2.—VARIATION OF COMPOSITION WITH SIZE OF PARTICLE.

1 shows screen analyses of calcines obtained by dead roasting the
centrates as described later. Fig. 2 shows the analysis of the
for copper, sulphur and iron for each screen size. The Nacozari
rates carry considerable coarse jig product while the Tyrone
l comes from an ore where the values are finely disseminated and
ntity of 100-mesh is quite marked. The presence of the flotation
rates in the Tyrone material brings up the copper contents of the
s.

### *Roasting*

object in roasting is to make as much of the copper and as little
on as possible soluble in dilute sulphuric acid. The work is simi-
oasting pyrites fines in sulphuric acid manufacture, except that
ibility ratio rather than the complete utilization of sulphur is the




NACOZARI CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH MCDOUGALL
ACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

ing factor. Small-scale work is not very satisfactory as a guide to
results as it is practically impossible to prevent overheating due
l oxidation of sulphur in a laboratory experiment. An 18-ft.,
oled, six-hearth McDougall was used, the speed of rotation
radually cut down until dead roasting conditions were obtained.
tonnages could doubtless have been obtained in a seven-hearth

y samples were taken from various hearths and the acid-soluble
and iron determined by leaching with 4 per cent. $H_2SO_4$ in the
ory. The results of these tests are given in Figs. 3 and 4. The

results of tests on a series of sixth-hearth samples to determine the relation between tonnage and sulphur elimination are plotted in Fig. 5. It is

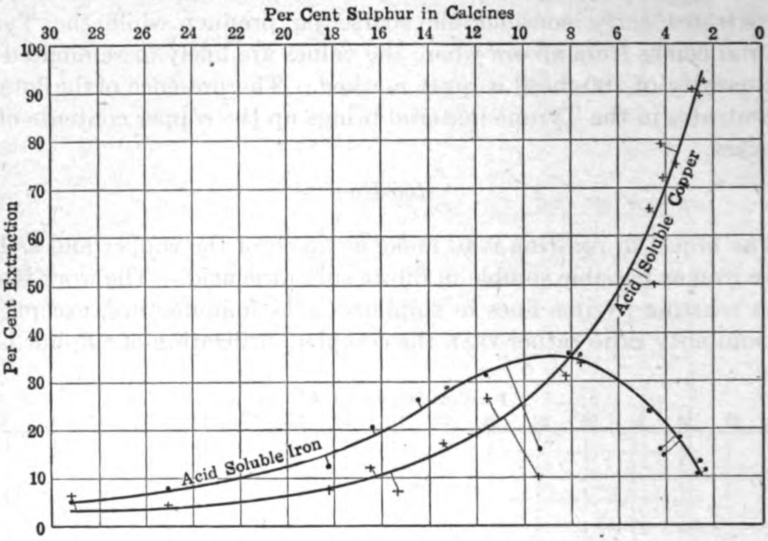



FIG. 4.—TYRONE CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH MCDOUGALL FURNACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

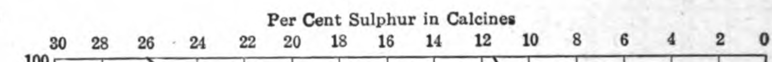



FIG. 5.—ROASTER CAPACITY VS. ELIMINATION OF SULPHUR.

evident that the chalcocite can be oxidized much more readily than the chalcopyrite, although size of particles has something to do with this. An investigation of the solubilities of the various sizes of particle was carried

out by screening some of the calcines, as shown in Fig. 6. As would be expected, the finer particles are the more thoroughly oxidized; the jig product in the Nacozari concentrate is one reason for the poorer results obtained in the treatment of this material.

In general, these large-scale experiments indicate the possibility of reasonably obtaining the results desired—high copper and low iron solubility—but it is obvious that the residue after leaching will contain sufficient copper values to require retreatment, aside from the fact that any silver and gold will remain in this residue.

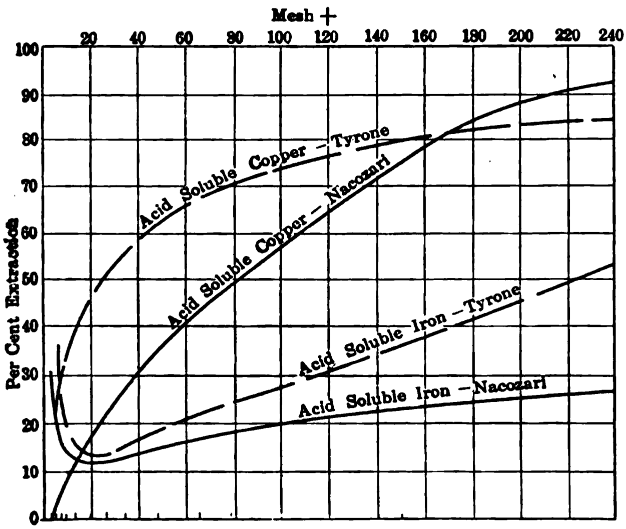



FIG. 6.—RELATIVE SOLUBILITIES OF VARIOUS SIZED PARTICLES OF CALCINE.

## *Leaching*

As shown in the paper presented last year, such satisfactory results in the extraction of copper values from tailings were obtained by dumping the hot calcines from the furnace into a leaching trough, the few seconds' agitation thus obtained extracting almost as much as prolonged treatment in other apparatus, that the same idea was tried out with the concentrate calcines. It was not possible, for various reasons, to handle the output of the furnace directly, so the calcines were stored and then fed to a bucket elevator which in turn delivered into a V-trough in which the leaching liquor was flowing. The results were here disappointing, as although there was instant extraction of perhaps half of the soluble copper, a prolongation of the trough to give 60 sec. travel did not greatly increase this amount. It was definitely shown in the laboratory as well that prolonged agitation was necessary to extract all of the soluble copper. The

leaching trough delivered into an acid-proof drag consisting of an
belt with angle rakes, of the type commonly used in concentra
dewatering. This acted more or less as a classifier, the very fine
being carried over with the liquor, from which they were subse
separated by settling. As this still gave insufficient agitation
sands, a Parral tank was tried, but it was found that they were to
to yield readily to any sort of air-lift agitation. A Dorr classi
then added to the apparatus and this did better. It was found, h
that it was necessary to pass the residues six or seven times thro
leaching process in order to obtain an extraction equal to that sh

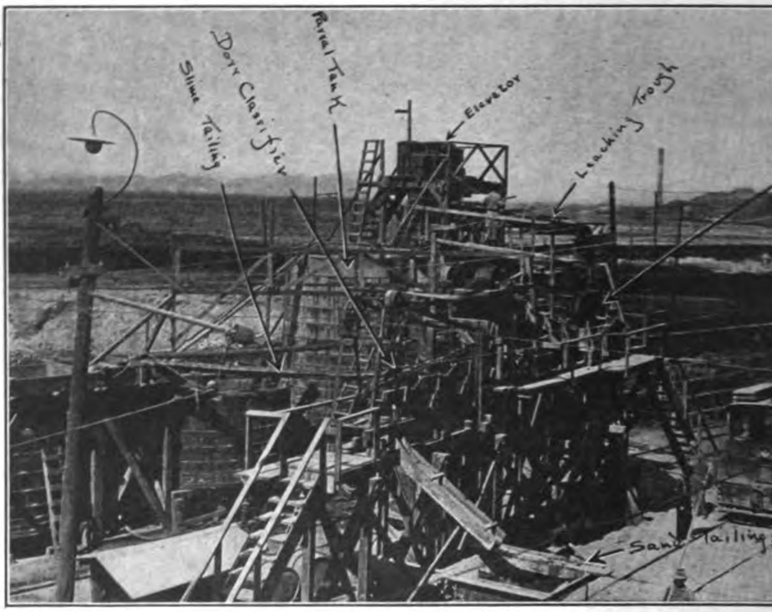



FIG. 7.—EXPERIMENTAL LEACHING PLANT.

laboratory tests on the calcines. Fig. 7 gives a general view
leaching apparatus.

The large-scale leaching tests were confined to the Tyrone ma
lot of 30 tons of calcines from some of the roasting tests being used
first runs on a lot of 17 tons of not quite dead-roasted material ru
per cent. sulphur gave results that were satisfactory except in t
much iron was dissolved, causing a needless consumption of a
embarrassing any electrolytic scheme of recovery. Later, better-
material was available and a careful record kept of the metal bala
acid consumption, with the results shown in Table 1.

These figures check reasonably close by except in the case of ir
it must be remembered that various iron parts in the apparatus us

TABLE I.

| Time Through | Per Cent. Cu in Tails | Trough Fed | Dorr Fed |
|---|---|---|---|
| 1st | 7.7 | Acid | Water |
| 2d | 6.0 | Acid | Water |
| 3d | 5.2 | Acid | Water |
| 4th | 3.6 | Acid | Acid |
| 5th | 3.3 | Acid | Water |

*Extraction by Heads vs. Tails*

| | Weight, Pounds | Copper | | Iron | | Alumina | |
|---|---|---|---|---|---|---|---|
| | | Per Cent. | Pounds | Per Cent. | Pounds | Per Cent. | Pounds |
| Heads | 8,360 | 15.48 | 1,292 | 31.00 | 2,590 | 5.60 | 468 |
| Tails | 5,600 | 3.50 | 196 | 43.52 | 2,440 | 7.02 | 393 |
| Extraction | ..... | 84.70 | 1,096 | 5.80 | 150 | 16.00 | 75 |
| Extraction per lb. of Cu. | ..... | ... | 1.00 | ... | 0.14 | ... | 0.07 |

*Extraction by Analysis of Liquors*

| | Weight, Pounds | Copper | | Iron | | Alumina | |
|---|---|---|---|---|---|---|---|
| | | Per Cent. | Pounds | Per Cent. | Pounds | Per Cent. | Pounds |
| Heads | 51,538 | 0.46 | 238 | 0.20 | 104 | 0.47 | 246 |
| Tails | 86,910 | 1.46 | 1,271 | 0.76 | 657 | 0.45 | 395 |
| Extraction | ...... | 80.00 | 1,033 | 21.30 | 553 | 31.90 | 149 |
| Extraction per lb. of Cu. | ...... | ... | 1.00 | ... | 0.53 | ... | 0.14 |

attacked by the liquor, which would artificially increase the iron taken into solution.

The acid consumption was 2,495 lb. of 100 per cent. $H_2SO_4$ for the run. This is equivalent to 2.28 lb. per pound of copper extracted. Laboratory tests on the same calcines indicated 2.0 lb. The leaching was done at about 125° F. with 5.6 per cent. free acid in the liquor entering the trough.

In general, when a 15 per cent. copper calcine is fed to the trough, the residue at the end of the trough will run about 8 per cent. Cu, the extraction representing the instantaneously soluble copper. This residue can be brought down to about 3.5 per cent. Cu by suitable agitating means, with a consumption of a little over 2 lb. of acid per pound of copper, and with the extraction of but little iron. The final residue weighs only about 60 per cent. of the original concentrate before roasting.

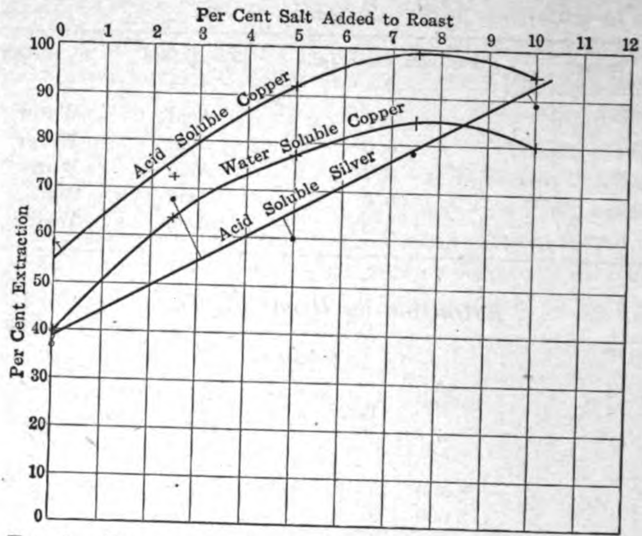



FIG. 8.—CHLORIDIZING LEACHED CONCENTRATES CALCINES.

**Roasted** 1½ Hr. at 975° F. with Addition of Salt and 10 Per Cent. Raw Concent
**Calcines**: 5.6 Per Cent. Copper, 1.9 Oz. Silver and 2.5 Per Cent. Sulphur.
**Raw** Concentrates: 14.4 Per Cent. Copper, 0.55 Oz. Silver and 34 Per
**Sulphur.**
**Liquor**: 5 Per Cent. $Na_2SO_4$, 5 Per Cent. Na Cl, 5 Per Cent. $Fe Cl_2$ and 0.
**Cent.** $HCl + H_2SO_4$.

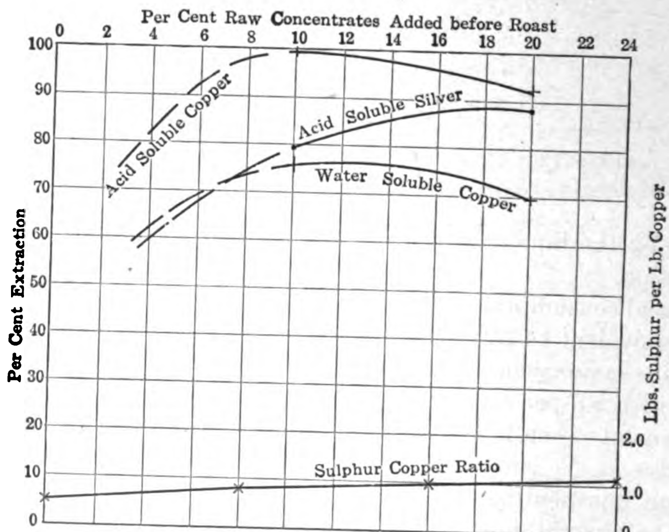



FIG. 9.—EFFECT OF SULPHUR-COPPER RATIO UPON EXTRACTION. 7½ PER C
SALT ADDED TO ROASTING "MIX".

*Chloridizing Residue*

No large-scale work was done on the chloridizing of the residues from the first leaching.. The analysis of these residues, however, differs from that of pyrites cinder, so long successfully treated by this process, only in the amount of silica present. Various small-scale experiments were tried and 50 lb. or so were sent for test to a plant where the Longmaid-Henderson process was in operation. Both sets of experiments were entirely satisfactory.

A small lot of leached residues was prepared for test. These contained 5.6 per cent. Cu, 1.9 oz. Ag, and 2.5 per cent. S. Raw concentrates for adjusting the sulphur-copper ratio were used, containing 14.4 per cent. Cu, 0.55 oz. Ag, and 34 per cent. S. Fig. 8 shows the extractions with varying percentages of common salt added to the "mix" after roasting in an electric muffle furnace 1½ hr. at 975° F. and leaching in a liquor carrying 5 per cent. $Na_2SO_4$, 5 per cent. NaCl, 5 per cent. $FeCl_2$, and 0.5 per cent. HCl + $H_2SO_4$. Fig. 9 shows the effect of varying the sulphur ratio. The results show a 99 per cent. copper and a 79 per cent. silver extraction. The report on the lot of residues sent away fully confirmed these results.

*Recovery of Copper from Solutions*

The liquor from the chloridizing plant would doubtless be reduced to argentiferous copper cement by iron. But 20 per cent. of the original copper is involved. The sulphate liquor from the first leach could be precipitated on iron if desired, or with certain limitations would be suitable for electrolytic deposition of the copper and regeneration of the acid. Sulphide concentrates carry from 1 to 2 lb. of sulphur per pound of copper, equivalent to from 3 to 6 lb. of 100 per cent. $H_2SO_4$, less process losses, if the roaster gas is oxidized to sulphuric acid. Since the leaching calls for but a little over 2 lb. of acid per pound of copper, plus tailings losses, it would seem possible, therefore, to figure on a simple cementation plant, considering electrolysis as a competitor on a basis of relative profit and not of necessity.

[DISCUS]SION OF THIS PAPER IS INVITED. It should preferably be presented in person at the [Arizona Mee]ting, September, 1916, when an abstract of the paper will be read. If this is impossible, [discussi]on in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West [39th St.,] New York, N. Y., for presentation by the Secretary or other representative of its author. [Unless speci]al arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion [offered there]after should preferably be in the form of a new paper.

# Smelting at the Arizona Copper Co.'s Works

BY F. N. FLYNN,* CLIFTON, ARIZ.

(Arizona Meeting, September, 1916)

## *Introductory*

[In 1]882, The Arizona Copper Co. Ltd., acquired producing copper [mines a]t Metcalf and Morenci (locally called Longfellow). Metcalf [is locat]ed a distance of 7 miles, and Morenci a distance of 6 miles from [the gene]ral office of the company at Clifton, Ariz. A reduction works [with] blast furnaces was built in the center of the town of Clifton in [... and] operated until January, 1914.

[Conc]entration of oxidized ores was begun in 1890–1891, in a mill [adj]oining the smelter, and in October, 1893, a leaching process was [started f]or treating tailings from the oxide mill. A sulphide concentrator [of ...] tons capacity was built at Clifton and started treating chalcocite [ores in J]uly, 1896. This was the first mill to treat low-grade copper-[sulphide] ores successfully. Another concentration mill for sulphide ores [was star]ted at Morenci in 1900. The sulphide mills are now being re-[modeled] to treat ores by concentration and flotation, with a capacity [of ...] tons at Morenci and 500 tons at Clifton. The leaching plant [has b]een running since September, 1914.

[The] average yearly production from this company's mines and [reductio]n works from 1885 to date has been:

| | Pounds |
|---|---|
| [...] years ended 1894 | 6,864,902 |
| [...] years ended 1904 | 20,439,614 |
| [...] years ended 1914 | 32,665,905 |
| [...]t of year 1915, to Sept. 12 | 30,206,106 |
| [...m]onths of 1916, to June | 15,964,840 |
| [Jun]e, 1916 | 4,900,000 |

[Const]ruction on a new smelting plant was started in February, 1912, [and finis]hed in February, 1914. In the *Transactions*, vol. 49, E. Horton [... pu]blished "Unit Construction Costs from the New Smelter of the [Arizona] Copper Co., Ltd.," which fully describes the construction of [the plan]t, with details of costs.

* [Su]perintendent of Smelting Department, The Arizona Copper Co., Ltd.

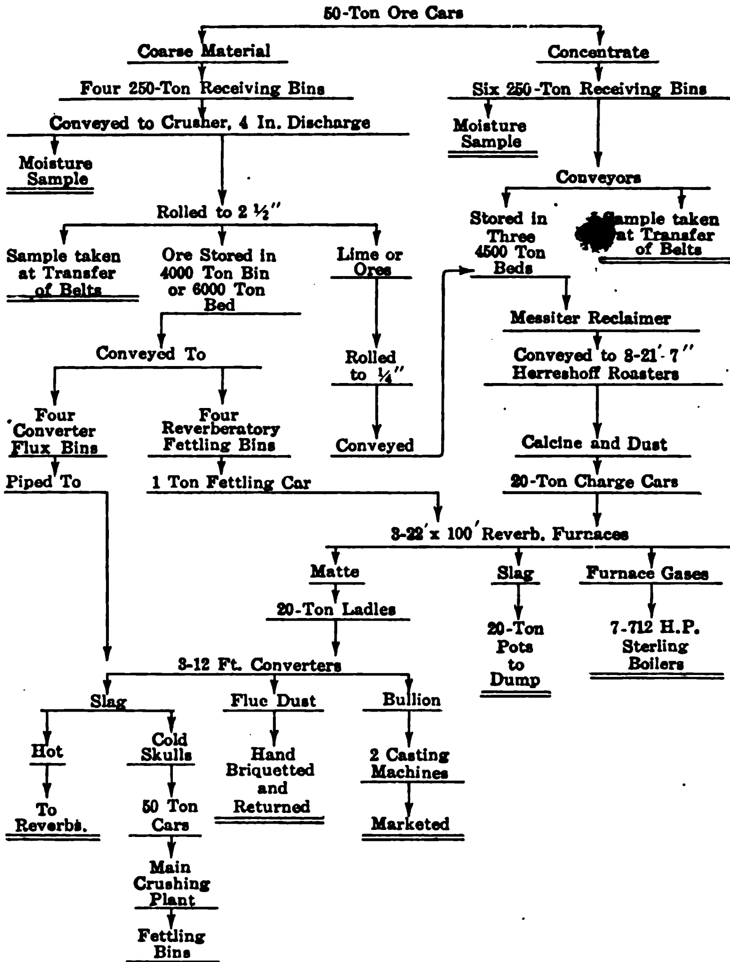
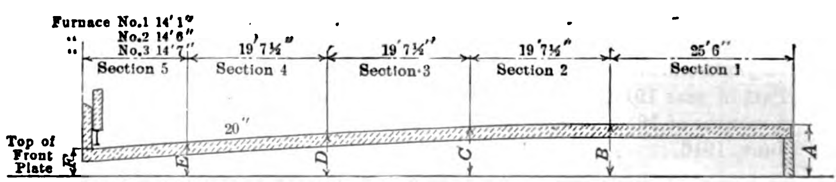



FIG. 1.—FLOW SHEET OF THE ARIZONA COPPER CO'S. SMELTING PLANT.




| | Reverb. No.1 | No.2 | No.3 | | Reverb. No.1 | No.2 | No.3 |
|---|---|---|---|---|---|---|---|
| A | 7'8⅝" | 6'10⅛" | 7'10" | D | 6'4½" | 6'7¾" | 6'0¾" |
| B | 7'10⅜" | 6'11½" | 7'4" | E | 5'4" | 5'6⅜" | 5'6" |
| C | 7'0⅜" | 6'9⅝" | 6'10⅝" | F | 4'0⅛" | 5'0½" | 4'8⅜" |

FIG. 2.—SKETCH SHOWING POSITION OF REVERBERATORY ARCHES ON CENTER LINE.

new smelter site is on gently sloping ground, 2 miles below the Clifton, and close to the San Francisco River. The Arizona Mexico standard-gage railroad and the Coronado narrow-gage serve the plant. Smelter yard standard-gage trackage consists iles for steam locomotives, and 1.72 miles for electric haulage. ence to the accompanying flow sheet will assist in following on of smelting practice at the new smelter.

## *Roaster Division*

er equipment consists of eight Herreshoff furnaces with steel ft. 7 in. in outside diameter. Hearths and linings in the different are built of materials as shown in the accompanying table.

*arth and Lining Materials in Herreshoff Roasting Furnaces*

| ier | Hearth Numbers From Top Downward | | | | | | Shell |
|---|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 | |
| brick | Firebrick | Firebrick | Gravel-concrete | Gravel-concrete | Gravel-concrete | 2-in. concrete | Red brick |
| brick | Firebrick shapes | Slag-concrete | Firebrick shapes | Firebrick shapes | Firebrick shapes | 2-in. concrete | Red brick |
| brick | Firebrick | Slag-concrete | Firebrick | Slag-concrete | Brick-concrete | 2-in. concrete | Red brick |
| brick | Firebrick | Firebrick | Gravel-concrete | Gravel-concrete | Gravel-concrete | 2-in. concrete | Red brick |
| brick | Firebrick | Firebrick | Firebrick | Firebrick | Firebrick | 2-in. concrete | Firebrick |
| brick | Firebrick | Firebrick | Firebrick | Slag-concrete | Slag-concrete | 2-in. concrete | Red brick |
| rick ks | Firebrick blocks | Firebrick blocks | Firebrick blocks | Firebrick blocks | Firebrick blocks | 2-in. concrete | Red brick |
| rick ks | Firebrick blocks | Firebrick blocks | Firebrick blocks | Firebrick blocks | Firebrick blocks | 2-in. concrete | Red brick |

designated "gravel concrete" were made of river gravel, 3-in. size and under, 2 parts; clean s; and cement, 1 part. Slag-concrete mixture consisted of crushed slag (all the size that in. screen) 3 parts; clean sand, 3 parts; and cement, 1 part. Composition of this slag was: r cent.; $Al_2O_3$, 8.5 per cent.; Fe, 27.5 per cent.; CaO, 3.3 per cent.: S, 0.5 per cent. In the of No. 3 furnace, crushed red brick was substituted for gravel in the concrete mixture.

irst nine concrete hearths built were reinforced with iron rods op and bottom courses. Reinforcing in those with inner drop ighed 3,093 lb., and those with outer drops 2,148 lb. When ted the fourth and fifth hearths, with heavy reinforcement, rose the center of the arch, which necessitated chipping off some of rete from the center of hearths. In later construction ½-in. reinforcing rods were used. This reduced the weight of iron lb. and 1,373 lb. respectively for inner and outer drop hearths. concrete hearths are preferred, providing the slag is highly

siliceous. All concrete hearths have given entire satisfaction. The[y] have been in service since May 3, 1914, without any repairs.

Provision is made for cooling the central columns and rabble arm[s] with air supplied from a motor-driven fan, but during the last 18 month[s] cooling air has not been required.

The roaster charge has varied from ore, limerock and concentrat[e] mixtures with 19.2 per cent. sulphur and 10.7 per cent. copper, to al[l] concentrates with 30.7 per cent. sulphur and 15.8 per cent. copper, wit[h] fuel requirements varying from a maximum of 40 lb. of coal per ton o[f] calcines to none when roasting all concentrate. The units of sulphu[r] burned off have ranged from 7.5 to 17.4, the usual practice being to leav[e] equal quantities of sulphur and copper in the calcine, because of th[e] oxidized ore used in reverberatory fettling.

Normally, the material roasted requires only a little more heat tha[n] is derived from the sulphur burned. Heavy crude oil was first used a[s] fuel, but with poor results, because with the low draft maintained burner[s] could not be obtained of suitable size to burn continuously the sma[ll] quantity of oil necessary to furnish this deficiency of heat, and distribut[e] it properly over the hearth. The flame was admitted through one doo[r] on the third or fourth hearth from the top. A burner that would sta[y] lighted in these comparatively cold hearths, produced an intense flam[e] and if kept running long enough to ignite the charge to the temperatur[e] desired, the brick hearth over the burner became too hot and charge fuse[d] on the hearth directly in front of the burners. Calcine produced was no[t] uniform, because the sulphur had to be controlled by burning oil inter-mittently, which resulted in first a hot furnace and low sulphur, then [a] cold furnace with high sulphur.

Coal-fired Dutch-ovens with 12 sq. ft. of grate area have been adopte[d] for supplying additional fuel required for roasting. Two ovens for eac[h] furnace admit the flame on the fifth hearth from the top. With thes[e] the roast can be quickly and closely regulated, and the furnace repair[s] have been greatly reduced as compared with oil-burning conditions.

Ignition usually takes place on the fourth hearth from the top, an[d] the temperature of the charge rises to 1,200 or 1,300° F. on the two lowe[r] hearths. With one reverberatory running, each roaster operated nor-mally makes 65 tons of calcine per day. When two reverberatories ar[e] taking charge, each roaster has to supply 80 tons of calcine. Result[s] show that while removing the same units of sulphur per ton, a furnac[e] will consume less coal per ton when producing 60 than when producin[g] 80 tons of calcine. Monthly averages of 113 tons of calcine pro-duced per furnace day show a very high coal ratio. Furnace damper[s] are kept closed just to the point of smoking. Gases leaving the fur-naces first enter a hopper-bottomed header-flue, through which the[y] travel at a velocity of 6 or 7 ft. per second, at 325 to 425° F. tempera-

d then pass into a large dust-chamber. Dust recovered from mber amounts to 0.8 per cent. of the dry charge to roasters.

dust-chamber roof, made of No. 11 plate steel, shows slight ation at the laps. A sheet-copper cap laid over the joint at apex lasted 2 years and has been replaced by concrete.

*Reverberatory Division*

lting for production of matte is all done in reverberatories, either two furnaces being operated as required. Three furnaces set to each other the long way, and 26 ft. apart, each with a hearth by 100 ft., are available, one of these always being a spare. The in No. 3 furnace is crushed silica fused in on a slag bottom. s Nos. 1 and 2 have hearths of crushed silica fused in on bottoms en quartz. Roofs were all originally built with 1 in. to the foot he arch. When the roof of No. 1 furnace was rebuilt, the arch en a rise of 1¼ in. to the foot. Six-inch holes, spaced 24 in. left in the roofs along the side walls and across the back, allow material to be piped into the furnaces from trough-shaped bins rtically over these holes. Five-inch diameter pipes on fettling being changed to 8-in., to permit using coarser material without the pipe openings. Other dimensions of these reverberatories ect their operation are given in an accompanying table.

*Reverberatory Dimensions*

| | Furnace Number | | |
|---|---|---|---|
| | 1 | 2 | 3 |
| atte tap hole to top of front plate... | 1 ft. 10⅝ in. | ..... | 1 ft. 3⅞ in. |
| p hole to skimming block, in use at ..... | 10⅞ in. | ..... | 1 ft. 1 in. |
| oil burners above front plate...... | 1 ft. 11⅛ in. | ..... | 1 ft. 8⅜ in. |
| f oil burners above skim block at in use...... | 1 ft. 11⅛ in. | ..... | 1 ft. 5⅞ in. |
| hroat area when slag line is level with ont plate, square feet...... | 17.4 | 26.2 | 22.0 |
| throat area, when slag line is raised ove top of front plate, square feet... | 13.2 | 22.6 | 18.0 |
| pening in verb shaft, square feet.... | 40.0 | 28.5 | 28.5 |
| ntake opening to header flue, square ...... | 45.6 | 55.2 | 54.6 |
| s of side walls, as rebuilt, above matte nes...... | 30.0 | 30.0 | 12.0 |

—The 12-in. side wall is more favorable for greater height of fettling.

In furnaces Nos. 2 and 3 the weight of the shaft has been taken off of the verb arch, by means of a telescopic shaft supported on steelwork. Fig. 2 shows the relative positions of the reverberatory arches on the center line of roof. This height is satisfactory when burning small quantities of oil, but might be raised considerably in Sections No. 1 and 2 for larger quantities of oil.

The following log of the third campaign of No. 3 reverberatory furnace is representative of operations on a one-furnace schedule:

Furnace started May 20, 1914.

6.208 days lost in August, repairing side walls, verb shaft, and 24 ft. of new roof on Section 1.

0.769 days lost in September, October, November and December, 1914, making minor repairs.

1.122 days lost in January, 1915, patching roof and front end.

2.736 days lost in March, 1915. New roof on Section 1.

5.642 days lost in July, 1915. 50 ft. roof repairs on Sections 1 and 2, and new jambs under header-flue arch.

Furnace stopped Sept. 12, 1915, on account of labor strike, but was in good repair at that time.

16.477 days lost time in 16 months total campaign.

*Dry Tons Solid Charge Smelted:*

| | |
|---|---|
| During total campaign | 178,165.0 |
| Daily average for total campaign | 384.6 |
| Daily average for one month, highest | 512.4 |
| Daily average for one month, lowest | 312.2 |
| Highest for one day, during highest average month | 543.0 |
| Lowest for one day, during highest average month | 413.0 |

*Barrels of Oil per Ton of Solid Charge:*

| | |
|---|---|
| Lowest month | 0.653 |
| Highest month | 0.951 |
| Average during campaign | 0.744 |

By a campaign is meant the elapsed time between starting a furnace and tapping out the matte. New roofs on Sections 1 and 2 were built upon ore centers, without tapping the matte. The last side walls repaired were in November, 1914. Since adopting greater fettling height, no further wall repairs are anticipated.

Two parallel charge tracks cross the furnaces near the back in a direction at right angles to their length. Fifteen-ton charges of calcine, at a temperature of 1,050 to 1,150° F., are dropped through one water-jacketed charge hole in the center of the roof on either track, but 90 per cent. of all charges are dropped from the back track. This part of the charge dropped through the roof is designated "direct charge." Seventy-five per cent. of the total solid charge smelted is introduced in this way.

Bad furnace conditions are always expected if such cold material as crude ore, slag or limerock of any size, or anything coarse like siliceous ore agglomerated with converter slag, either cold or warm, is dropped as direct charge without mixing it well with hot calcine. Ore, limerock or clinker produces "charge floaters." Cold slag sinks and makes sticky furnace bottoms. When it is necessary to add cold or coarse material to the direct charge, the best results are obtained when they are well mixed with hot calcines, in the calcine charge cars, but better still in the roasting furnaces.

Bed mixtures to be roasted are made of the required composition to produce a slag, when smelted, in which the ratio of oxygen in acids to oxygen in bases will fall between 2.00 and 2.25. A vast difference, however, is found in the smelting properties of calcines made from these mixtures, regardless of the similarity between their analyses or the analyses of the slags they produce. Calcine made from concentrate or sulphide ores of ½ in. size smelts readily, whereas if the calcine is an artificial mixture made of siliceous oxidized ore crushed to this size, and mixed with an excessive proportion of limerock, the charges melt more slowly. The results from the two following mixtures will fairly illustrate this point:

| Class of Material | Mix 103. Per Cent. of Total | Mix 120. Per Cent. of Total |
|---|---|---|
| Sulphide concentrate | 100.0 | 70.7 |
| Siliceous ore, low sulphur | ..... | 4.8 |
| Siliceous oxide ore | ..... | 12.9 |
| Limerock | ..... | 11.6 |
| Total | 100.0 | 100.0 |

| Mix No. | Analysis | | | | | | |
|---|---|---|---|---|---|---|---|
| | $SiO_2$, Per Cent. | $Al_2O_3$, Per Cent. | Fe, Per Cent. | CaO, Per Cent. | S, Per Cent. | Cu, Per Cent. | Oxygen Ratio |
| 103 | 18.3 | 4.6 | 28.2 | 0.7 | 29.0 | 14.10 | 1.43 |
| 120 | 20.7 | 4.7 | 24.1 | 7.6 | 22.5 | 10.98 | 1.45 |

A daily average of 365 tons of calcine per furnace was smelted during the time that mix 103 was on the charge. In comparison, 200 tons per furnace day was barely maintained while the charge consisted of calcine from mix 120. The practice is to crush siliceous ore as fine as 3/16-in. for roaster beds, and endeavor to keep the sulphide material as high as

75 per cent. of the total bed mixture. Ore and limerock mixtures req a much higher oil ratio than calcine from straight concentrate.

In other words, the greatest percentage of oxidized material wh it is permissible to smelt in the reverberatory furnace is found to be:

| | Tons per Furnace Day | Per Cent. | |
|---|---|---|---|
| Direct charge | 300 | 25 | Oxidized ore and limer |
| Fettling | 100 | 100 | Oxidized ore. |
| Total | 400 | 44 | Oxidized ore equivalent |

Above 44 per cent. oxidized ore, the charge could undoubtedly better handled in a blast furnace. The charges spread better when furnace is nearly full of matte, and also when the copper and sulphu the calcine are equal. Matte under 40 per cent. copper is favorable higher tonnage.

*Fettling.*—Ore and byproducts crushed to a 2½-in. size for fett constitute the remaining 25 per cent. of the solid charge smelted. I customary after each skim to feed these materials from bridge-wall shaft in the following proportions:

| Furnace | Section No. | Per Cent. | Material |
|---|---|---|---|
| Back | 1 | 25 | Old smelter slag. Converter slag skulls. |
| Back | 2 | 50 | Less siliceous ore. |
| Middle | 3 | 12 | Less siliceous ore. |
| Middle | 4 | 8 | Less siliceous ore. |
| Front | 5 | 4 | More siliceous ore. |
| Shaft | .. | 1 | Silica. |

| | Analysis | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | $SiO_2$, Per Cent. | $Al_2O_3$ Per Cent. | Fe, Per Cent. | CaO, Per Cent. | MgO, Per Cent. | S, Per Cent. | Cu, Per Cent. | Oxy Ra |
| Converter slag skulls | 19.6 | 3.9 | 47.2 | 0.9 | 0.0 | 3.6 | 9.00 | 0 |
| Old smelter slag | 37.2 | 8.9 | 22.0 | 10.3 | 5.2 | 0.3 | 3.56 | 2 |
| Less siliceous ore | 42.6 | 7.7 | 18.0 | 3.0 | 2.5 | 1.0 | 6.80 | 3 |
| More siliceous ore | 65.0 | 12.2 | 5.6 | 0.8 | 0.6 | 1.9 | 3.78 | 19 |
| Silica | 84.4 | 6.3 | 3.1 | 0.5 | 0.0 | 2.6 | 0.74 | 46 |
| Approximate average | 40.4 | 7.5 | 21.5 | 3.5 | 2.4 | 1.3 | 6.49 | 3 |

or to July, 1914, the height of fettling was carried not to exceed above the slag line. This height was gradually increased until er, 1914, when the practice of fettling to the holes in the roof was enced and later successfully adopted after the size of fettling al was increased from ¾ in. to the present 2½-in. size. The coarse ands better. It reposes at an angle of 45° above and 45° to 60° the slag line.

e results achieved in performing fettling operations as practised lepend upon the relative location and manner in which the material pped, as well as the amount of fines in it and its chemical and al composition, as may be concluded from the following account of ence with different fettling materials.

Slag and byproducts crushed fine work unsatisfactorily in any part furnace. However, if crushed coarse, results are good when such al is charged in the back of the furnace.

Less siliceous fine ore is not good fettling in any place in the e. It does not stand, and requires care to keep it from running out ne furnace. Crushed coarse, it makes good fettling wherever used.

More siliceous fine ore mixed with coarse byproduct make a bad nation. The only place the more siliceous fine ore alone can be vith any degree of success is on the bridge-wall.

More siliceous coarse ore is not so acceptable as less siliceous coarse cept in the front half of the furnace, because it makes too much us scum on top of the slag.

Coarse raw concentrate is good in the back half of the furnace, es not stand so well nor so high as coarser material. It gives bad in the front half because it melts easily to a magnetic mush.

Fettling with hot calcine can only be done with any degree of s in Section No. 1, and then only slightly above the slag line. At levation it runs out over the furnace bottom. An attempt was to fettle the entire furnace with calcine, but was abandoned when nnage dropped to less than half capacity. Calcine gave poorer than any other material experimented with.

Pyrite in the front half melts easily to a magnetic mush. In the alf when mixed with siliceous oxide ore it works well.

The ideal fettling material was found to be less siliceous copper-de ore containing 5 to 10 per cent. sulphur.

other words, basic or sulphide materials are charged near the end because of their lower fusing points. If charged in the front n, where the temperature is lower, they melt with sufficient ease ke a thick, mushy, semifused slag, and in the case of pyrite make ties of magnetite. Siliceous ores with 60 per cent. silica produce n which blankets the slag and causes higher oil ratio and lower e. If the siliceous ores contain much copper in oxidized form,

the scum will assay from 0.7 to 1.3 per cent. copper, whereas if th
is present as a sulphide the scum will assay very much lower.

The most important development as regards fettling has bee
the line of coarser crushing and greater height of material to
the side walls without encroaching on the furnace bottom area.
"floaters" are unknown. The successful results of this practice
seen from the photograph of one of the furnaces (Fig. 3).

Fuel is 14 Bé., 18,000 B.t.u., California crude oil, pumped
lb. pressure and delivered through steam heaters at a temper
170° F. to six burners and atomized with converter air. Two

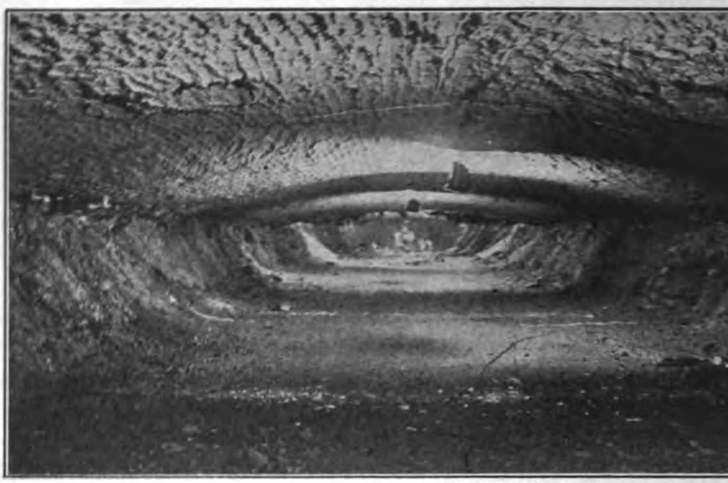


FIG. 3.—REVERBERATORY FURNACE NO. 3 OF ARIZONA COPPER CO.,

Campaign No. 3. Started May 20, 1914, finished Sept. 12, 1915 (dat
strike.)

Dry tons of solid charge smelted: total 178,165; highest daily averag
month, 512.4.

Fettled to the roof with "less siliceous" oxide ore and converter sl
crushed to 1½ in. Slope of fettling, from 45 to 60°. Fettling material amou
per cent. of total solid charge.

and forty to 365 bbl. of oil is burned per furnace-day. This v
covers a one-furnace schedule when hard firing for high tonna
quired, or two furnaces with light firing and correspondingly low
smelted.

Furnace-throat draft (inside the furnace) is 0.12 to 0.16 in. c
and in header-flue common to all furnaces, draft is 0.50 in. water.
leave the furnaces at temperatures from 1,800 to 2,000° F. Seven
boilers are available for reverberatory waste heat, each 712-hp. c
and all connected in multiple through cross-over flues from the
flue. Dampers are so arranged that gases can be sent to any o
of the boilers. There is no bypass to the chimney. Four or fiv

are used when running one furnace. Reverberatory dust caught in all the boilers does not exceed 5 tons monthly when running one reverberatory.

Furnace gases leaving the waste-heat boilers at temperatures varying between 471° and 588° F., are conducted through 6-ft. diameter steel pipes into a flue connecting with the main chimney. The temperature of gases in the flue ranges between 400° and 500° F. at different points, and at these temperatures the average velocity of gases is 10 ft. per second, when running one furnace. An average analysis of this gas is $SO_2$, 0.3 per cent.; $CO_2$, 5.9 per cent.; O, 12.9 per cent.; CO, 0.4 per cent.; N, 80.5 per cent.; $SO_3$, not determined. When the flue was cleaned recently, 100 tons of dust was removed that had accumulated during 2 years' furnace operations. The composition of this dust was:

| | $SiO_2$ | $Al_2O_3$ | Total Fe | Ferrous Fe | CaO | S | Cu | Free $H_2SO_4$ | $H_2O$ |
|---|---|---|---|---|---|---|---|---|---|
| Reverberatory chamber dust | 17.4 | 6.3 | 11.8 | .... | 2.3 | 10.3 | 9.65 | 3.0 | 9.5 |
| Soluble in water | .... | 1.8 | 3.7 | 2.6 | 2.1 | 9.1 | 6.60 | | |
| Per cent. of total, soluble in water | .... | 28.3 | 31.4 | .... | 91.4 | 88.3 | 68.4 | | |

The flue roof was originally made of No. 14 steel plate. This had to be repaired after 8 months' service. Holes were eaten in the plate and in spots the whole sheet was pitted and nearly eaten through. Moist flue dust covered the inside surface of the roof. This deposit consisted principally of sulphates of iron and copper, as is shown by the following composition calculated from analysis of dry sample:

| | |
|---|---|
| Insoluble | 3.0 |
| $CuSO_4$ | 7.7 |
| $Al_2(SO_4)_3$ | 8.0 |
| $FeSO_4$ | 18.7 |
| $Fe_2(SO_4)_3$ | 45.3 |
| $ZnSO_4$ | 3.2 |
| $CaSO_4$ | 2.9 |
| Free $H_2SO_4$ | 12.3 |
| Total | 101.1 |

This would not jar loose from the sheet, and metallic copper had deposited on the iron. Concrete slabs reinforced with wire netting were laid on top of the steel plate and the joints between slabs filled with asphalt. The concrete covering has served 2 years. The corrosive action of gases on the iron was arrested for 10 months of operations after the slab roof

was put on, but during an idleness of 5 months, due to labor trouble the plate had corroded completely.

Lime in the mortar used in laying tile walls of this flue was partl changed to sulphate by the furnace gases. While the flue was cold, th walls swelled far out of plumb. The tile broke and walls had to b rebuilt. New walls are brick laid in slime mortar.

Converter slag is poured through the back wall of furnace, at th center, through a launder discharging at the height of the oil burner Magnetite builds up under the discharge of the launder, but the bridge wall on either side of it requires fettling.

Considerable has been written regarding the fluxing value of converte slag poured into reverberatory furnaces. The prevailing opinion seem to be that very little fluxing value is within one's control. It is argue that the basic converter slag does not act upon the charge resting on th slag surface, does not mix with the trisilicate slag, and passes through th furnace like a submarine as a monosilicate, emerging near the skimmin end.. The technical reason advanced for these arguments is the rela tively high specific gravity of the basic converter slag. The writer ha proved to his own satisfaction that practically all of the iron present i liquid converter slag, other than iron in the magnetic state, is effectiv as flux. The violent boiling and the reactions that take place whe calcine charge is dropped through the central charge hopper are effectiv in mixing any slag that may have separated into layers in the furnac at the back. If it is true that the charge dropped in the back of th furnace partly melts to a trisilicate slag, underlaid by the bisilicate, whic in turn rests upon the monosilicate converter slag, it would be a difficul task to explain why the surface or trisilicate slag is so active in its con sumption of fettling material at the slag line. On the other hand, it i not difficult to imagine the monosilicate slag attacking the fettlin material along the side walls, from the matte line up. How it would fai to do so would be difficult to explain.

The effect of liquid converter slag was vividly shown on one occasion when a furnace had been tamped with moist quartz from floor to roo along the walls, and when fused was largely filled with hot converter sla and little regular charge. Within 24 hours there was little quart remaining.

On several occasions, two furnaces running on identical charges hav been used for illustrative purposes. Into one, all of the converter sla was poured. In the other, without converter slag, the full quantity o limerock equivalent to the total fluxing value of the converter slag wa required.

There are others who contend that even magnetite in converter sla is acted upon by the furnace matte and becomes effective as flux. Ou own experience is that whenever magnetite accumulates in the furnace

, be pulled out with the rabble as mush, the necessary limerock to the charge, and plenty of sharp drills made ready at the matte e.

absence of an accurate method for the determination of magnetic f iron in furnace byproducts is one of the most unfortunate and able conditions with which the metallurgist is confronted, and the elucidation of this obscure subject difficult. Without some of quantitatively determining this substance, accurate data as to ing value can not be obtained from balance sheets even if materials oducts are all weighed, because it is not certain from an analysis e magnetic iron has been changed in passing through the furnace.

*Converter Division*

production of magnetite in the converters is, within reasonable controlled by the converting practice. The converter slag poured e reverberatory furnace should not contain any magnetite visible eye. The safest practice is to reject the last 2 cu. ft. of slag when . The rejected slag with the ladle skull will eventually enter the as fettling material, where its detrimental effect to the furnace orarily turned to some good, because of its refractory properties. dency to stick to the sides when mixed with cold ore is quite ble when the fettling bank is removed from the walls when the is out of service. By this practice, magnetite is made to perform what similar service in the reverberatory furnace to that which rms in the protective coating in a basic converter. Unfortunately small portion of the total magnetite production can be so used. e metallurgists prefer to make a small quantity of magnetite on nverter charge, contending that in that way the coating remains constant thickness. They accomplish this by blowing a few s on the start or finish, either or both, but usually at the start, with age of silica. The writer is of the opinion that by intentionally magnetite with every charge, the quantity made must be very compared with the small amount which can be made to stick to ls by so haphazard a practice going on continuously. Our practice at the shells when the lines of brickwork can be seen through the ating. And, when a shell is being coated, no attempt is made anything with the matte other than form the coating. When the magnetite and copper mush is dumped on the floor under the ter it is charged in small quantities at intervals in other charges. coated, a shell is never blown without a large excess of silica. gh siliceous ore is used for flux, 350 lb. of 80 per cent. silica is used ulate the final slag on the copper finish, to insure the removal of traces of iron from the copper with the more active silica. Since

this quartz, with any magnetite which it has collected, remains in converter for the next charge, the magnetite is in contact with quartz particles from the start of the blow.

The converting division is equipped with five 12-ft. Great Falls-t shells lined with magnesite brick; three converter stands, each opera by an independent motor (the fourth stand is under constructic two 40-ton cranes; two straight-line casting machines, each with copper molds. Two stands are regularly used, for which air at 13 pressure is supplied from power house by two of three Nordberg blow engines.

When converting matte of over 40 per cent. copper, only a sn amount of cold matte shells and cleanings from the floor can be used; with lower-grade matte, all matte shells, converter flue dust, and converter byproducts made except converter-slag skulls can be hanc in this division.

Molten copper is transferred from converters to casting machi in cast-steel ladles lined with fines screened from ores regularly usec converter fluxes. Bars are cast in molds made of converter cop A 1¼-in. cast-iron splash plate covers half of the bottom area, and average of 73 tons of bullion is cast per mold. Bars weigh 240 lb. e and 35 min. is required for casting a charge weighing 7 tons. Considera chipping of bullion bars is necessary to remove edges and fine shot du blowing to gas finish of 99.60 per cent. copper.

*Comparison of One-Month Periods with One and Two Reverbera Furnaces in Operation*

| Roaster Division | One Reverberatory | Two Reverberato |
|---|---|---|
| Total tons dry charge roasted | 14,257.0 | 17,235. |
| Dry tons charge per furnace-day running time | 78.4 | 73. |
| Analysis of charge: | | |
| $SiO_2$, per cent. | 17.3 | 19. |
| $Al_2O_3$, per cent. | 4.7 | 4. |
| Fe, per cent. | 28.2 | 24. |
| CaO, per cent. | 1.6 | 6. |
| S, per cent. | 28.8 | 24. |
| Cu, per cent. | 13.61 | 12. |
| Oxygen ratio | 1.33 | 1. |
| Analysis of calcine: | | |
| $SiO_2$, per cent. | 20.0 | 22. |
| S, per cent. | 13.8 | 13. |
| Cu, per cent. | 15.70 | 13. |
| Weight of calcine produced as per cent. of dry charge | 86.7 | 89. |
| Units sulphur eliminated | 15.0 | 10. |
| Pounds coal used per ton of roaster charge | 4.2 | 14. |
| Waste-gas temperature | 428.0 | 396. |

| Reverberatory Division | One Reverberatory | Two Reverberatories |
|---|---|---|
| direct charge per furnace-day | 402.5 | 262.7 |
| fettling per furnace-day (ores) | 84.9 | 65.7 |
| fettling per furnace-day (byproduct) | 25.0 | 38.7 |
| total solid charge smelted per furnace-day | 512.4 | 367.1 |
| of fettling to total solid charge | 21.45 | 28.3 |
| of direct charge: | | |
| er cent | 19.8 | 21.7 |
| per cent | 5.4 | 4.9 |
| cent | 32.3 | 26.6 |
| er cent | 2.3 | 7.7 |
| cent | 13,7 | 13.6 |
| cent | 15.55 | 13.24 |
| ratio | 1.32 | 1.41 |
| of fettling (ores and byproducts): | | |
| er cent | 38.8 | 39.5 |
| per cent | 7.6 | 8.0 |
| cent | 23.4 | 25.5 |
| er cent | 1.7 | 4.0 |
| cent | 9.1 | 3.0 |
| cent | 6.65 | 5.52 |
| ratio | 3.36 | 2.93 |
| s hot converter slag to reverberatories | 2,419.0 | 2,862.0 |
| of reverberatory slag: | | |
| er cent | 35.6 | 38.0 |
| per cent | 9.7 | 9.0 |
| er cent | 49.7 | 39.4 |
| er cent | 2.4 | 8.4 |
| per cent | 0.6 | 1.5 |
| cent | 0.51 | 0.45 |
| ratio | 2.04 | 2.06 |
| of matte, per cent. Cu | 37.43 | 37.70 |
| matte fall | 31.68 | 27.1 |
| sulphur volatilized in furnace | 24.14 | 25.7 |
| urned per ton solid charge smelted | 0.653 | 0.778 |
| urned chargeable to steam | 0.315 | 0.349 |
| urned account smelting | 0.338 | 0.429 |

accompanying 2 months' results were selected for comparison, they show the extreme variation in the material treated in this ent. Limited storage capacity at the smelter required two fur- be operated a part of the time to handle receipts. Additional eeded for the month that two reverberatories were operated was ed of siliceous oxidized ores and limerock, and represents a tem- condition. The concentrator additions, now almost finished, will uture supply sufficient concentrates to make the proportions of

total material treated fall somewhere within the percentage limits g in the examples.

| Converter Division | One Reverberatory | Two verbera |
|---|---|---|
| Tons reverberatory net matte treated | 5,293.0 | 5,90 |
| Per cent. Cu in reverberatory net matte treated | 37.43 | 3 |
| Tons siliceous ore used for flux | 820.0 | 1,35 |
| Analysis of siliceous ore used: | | |
| $SiO_2$, per cent. | 64.5 | 6 |
| $Al_2O_3$, per cent. | 10.6 | 1 |
| Fe, per cent. | 5.2 | |
| CaO, per cent. | 0.2 | |
| S, per cent. | 0.5 | |
| Cu, per cent. | 6.45 | |
| Oxygen ratio | 25.30 | 2 |
| Analysis of converter slag produced, per cent. $SiO_2$ | 19.6 | 1 |
| Tons copper bullion produced | 1,800.0 | 2,45 |
| Cu in bullion, per cent. | 99.60 | 9 |
| Tons copper produced per converter-day running time | 43.9 | 5 |
| Minutes blowing per ton copper | 33.0 | 2 |
| Average tons bullion per charge | 6.429 | |
| Average time of blowing a charge | 3 hr. 54 min. | 3 hr. 5 |
| Tons cold new material treated per ton bullion | 0.462 | |
| Tons cold byproduct re-treated at converters per ton bullion | 0.849 | |
| Blast, pounds pressure | 13.0 | 1 |
| Cu. ft. air per converter per minute | 6,738.0 | 5,44 |
| Cu. ft. air per ton bullion produced | 220,949.0 | 152,49 |
| Cu. ft. air per ton iron and sulphur eliminated | 135,043.0 | 96,14 |
| Oxygen efficiency, calculated from blower displacement and requirements of Fe and S eliminated | 60.63 per cent. | 84.0 cen |
| *Total Material Treated—All Divisions:* | | |
| Dry tons concentrates | 13,227.0 | 13,36 |
| Dry tons ores | 4,069.0 | 7,30 |
| Dry tons limerock | 391.0 | 2,13 |
| Dry tons new material, total | 17,687.0 | 22,79 |
| Dry tons byproducts treated at reverberatories | 924.0 | *2,31 |
| Concentrates, per cent. of total new material treated | 74.78 | 5 |
| Ores, per cent. of total new material treated | 23.01 | 3 |
| Limerock, per cent. of total new material treated | 2.21 | |

* Including reverb. slag skulls.

### *Boiler Plant*

There are 10 Stirling boilers, three 384-hp. direct-fired, and s 712-hp. waste-heat. Water is measured in total and also separ to the two sets of boilers. Oil is measured to the direct-fired bo

All boilers contain Foster superheaters, and deliver steam to the mains at 175 lb. pressure. The temperature of steam delivered at the power house is about 475° F. The temperature of the waste gases leaving the boilers varies between 471° to 588° F., depending upon the furnace practice and the number of boilers in service. The gases from one furnace practice are passed through five boilers. When two furnaces are in commission, six or seven boilers are used.

When smelting in one furnace, three direct-fired boilers are in continuous service to supply the power requirements, but the night load is light, therefore the boiler efficiency is low. The pounds of water evaporated per pound of oil averaged 11.98 over a period of 8 months. The oil averages around 15° Bé. and 18,200 B.t.u.

With two furnaces in operation, there is sufficient waste heat to generate all steam requirements for the night load, but oil must be used to carry the day load. Under these conditions, rather than keep the oil-fired boilers under steam over night, the three oil-fired boilers are not used. Instead, oil is burned under the waste-heat boilers. The oil so burned is measured and credited with evaporating 11.98 lb. of water, in the same manner as under oil-fired boilers on a one-furnace basis.

The net steam delivered to the power-house engines is credited to the boiler plant. Steam used in heating and atomizing oil and for other miscellaneous purposes, as well as steam wasted, is absorbed in the furnace-oil costs.

Under one-furnace conditions, the waste-heat boilers averaged over a period of 8 months, 7.71 lb. of water evaporated per pound of oil burned in the furnace.

The furnace-boiler division receives credit from the power house varying from 36 to 56 per cent. of the oil burned in the smelting furnace.

The indicated horsepower from the waste-heat boilers, per ton of solid charge smelted on a one-furnace basis over a period of 9 months averaged 98.71 i.hp.-hr. whereas on a two-furnace basis, over a period of 7 months, the average was 114.34 i.hp.-hr. from waste heat only.

*Smelter Power*

| Divisions | Kilowatt-hours | | Kilowatt-hou per Ton New Ma | |
|---|---|---|---|---|
| | One Furnace | Two Furnaces | One Furnace | Fu |
| *Sampling:* | | | | |
| Receiving, high-pressure air converted to kw.-hr | 1,813 | 2,590 | 0.10 | [illegible] |
| Crushing, electric | 10,501 | 21,548 | 0.59 | [illegible] |
| Sampling, electric | 1,146 | 2,351 | 0.06 | [illegible] |
| Bedding, electric | 3,819 | 7,836 | 0.22 | [illegible] |
| Reclaiming, electric | 22,576 | 22,843 | 1.28 | [illegible] |
| Total | 39,855 | 57,168 | 2.25 | [illegible] |
| *Roasting:* | | | | |
| Roaster furnaces, electric | 7,692 | 7,495 | 0.43 | [illegible] |
| Calcine cars, electric | 3,140 | 5,093 | 0.18 | [illegible] |
| Total | 10,832 | 12,588 | 0.61 | [illegible] |
| *Reverberatories:* | | | | |
| Reverb. furnaces, electric | 3,013 | 3,303 | 0.17 | [illegible] |
| Reverb. furnaces, high-pressure air converted to kw.-hr | 453 | 647 | 0.03 | [illegible] |
| Reverb. furnaces, low-pressure air converted to kw.-hr | 37,339 | 72,774 | 2.11 | [illegible] |
| Slag railway, electric | 4,420 | 10,451 | 0.25 | [illegible] |
| Boilers, high-pressure air converted to kw.-hr | 1,813 | 2,590 | 0.10 | [illegible] |
| Boiler feed-water pumps, electric | 31,220 | 20,553 | 1.77 | [illegible] |
| Oil pumps, electric | 4,030 | 2,009 | 0.23 | [illegible] |
| Total | 82,288 | 112,327 | 4.66 | [illegible] |
| *Converting:* | | | | |
| Converters, electric | 5,263 | 5,674 | 0.30 | [illegible] |
| Converters, low-pressure air converted to kw.-hr | 218,235 | 206,802 | 12.34 | [illegible] |
| Cranes, electric | 22,790 | 19,635 | 1.29 | [illegible] |
| Bullion casting machines, electric | 2,340 | 2,920 | 0.13 | [illegible] |
| Total | 248,628 | 235,031 | 14.06 | 10 |
| *General Works:* | | | | |
| River well, electric | 6,760 | 8,001 | 0.38 | [illegible] |
| Boiler and machine shops, electric | 2,900 | 2,730 | 0.16 | [illegible] |
| Boiler and machine shops, high-pressure air converted to kw.-hr. | 4,986 | 7,122 | 0.28 | [illegible] |
| Yards, electric | 1,313 | 1,439 | 0.07 | [illegible] |
| Laboratory, electric | 2,807 | 3,077 | 0.16 | [illegible] |
| Total | 18,766 | 22,369 | 1.05 | [illegible] |
| Total, all divisions | 400,369 | 439,483 | 22.63 | 19 |
| Dry tons new material treated | 17,687 | 22,793 | | |

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Leaching Tests at New Cornelia

BY H. W. MORSE, PH. D., LOS ANGELES, CAL.,* AND H. A. TOBELMANN,† AJO, ARIZONA

(Arizona Meeting, September, 1916)

## Introduction

The experimental work on the oxidized copper ore at the New Cornelia mine at Ajo, Ariz., ended on Jan. 12, 1916. On that date final decision was made on the general nature of the process to be used in the 5,000-ton leaching plant, and on many of the details, as far as experience on a40-ton scale could decide them.

With the approval of the Board of Directors and the General Manager, Mr. J. C. Greenway, we have compiled what seem to be the most interesting data on the results obtained during the experimental period. The most important part of the data resulted from the operation of a 1-ton and a 40-ton plant at Ajo.

Experimental work on Cornelia oxidized ore dates back to April, 1912, and has been going on nearly continuously since that time. A good many variations from the original idea have been tried, but the process finally decided upon is in principle and in all of its details a simple one. The history of the leaching work on this ore has been brought down to about a year ago in papers presented to this Institute by Stuart Croasdale[1] and by L. D. Ricketts.[2] The present paper will, therefore, deal principally with the results obtained during the last year of the work.

## Preliminary Tests by Mr. Croasdale

Preliminary tests were begun by Stuart Croasdale in July, 1912. A number of important points were definitely decided by his work. Among these were:

1. New Cornelia oxidized ore, not very finely crushed (to about 2-mesh) showed good extraction with 5 per cent. sulphuric acid. It seemed that better than 80 per cent. extraction could be expected.
2. The consumption of acid for this extraction was not prohibitive. It was apparently 3.5 to 4 lb. gross per pound of copper recovered.

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[1] Leaching Experiments on the Ajo Ores, *Trans.*, vol. 49, p. 610.

[2] Some Problems in Copper Leaching, *Trans.*, vol. 52, p. 737.

3. It was proposed to make cement copper. The iron consump per pound of copper recovered was a little over 1 lb.

4. A column 12 ft. deep of ore crushed to this fineness coulc successfully percolated.

During Mr. Croasdale's work the plan in view was a very simple indeed. The ore was to be leached with sulphuric acid. Cen copper was to be produced by the use of metallic iron. As Dr. Rick explained to the Institute nearly 2 years ago, the oxidized Cornelia could apparently be leached and cement copper produced at a pr if sulphuric acid and scrap iron were bought in the open market shipped to Ajo, and cement copper shipped out, all solutions being thr away after removal of the copper.

Mr. Croasdale showed also that the amount of substances other t copper, taken into solution under his conditions of crushing and leach is small. This naturally suggested the possibility of a closed leac cycle with electrolytic deposition and at least partial regeneration of acid required for solution of the copper. Early experiments showed iron, which is dissolved from the ore to a certain extent, and oxidize the anode during electrolysis, was a troublesome factor.

There seemed to be three possible ways of meeting this difficu (1) The use of a diaphragm; (2) the use of a depolarizer; (3) purifica from iron before electrolysis.

## Antisell Process

About this time there came to the notice of the manager good re that were being obtained with a special anode, the invention of F Antisell. This anode was in the form of a long, deep, narrow box wood veneer sides, containing graphite electrodes and having the s between graphite and wood filled with selected coke.

A cyclic process was tried with these anodes. Ore crushed t mesh was leached 3 days by upward percolation, with a practically stant solution containing 3 per cent. $H_2SO_4$, 2 per cent. ferrous iron per cent. ferric iron and 2 per cent. $Al_2O_3$. Sulphur dioxide gas was to keep down the ferric iron concentration.

The process gave good results. The anode was, however, ra cumbersome and the absorption of the sulphur dioxide was not s factory, presumably owing to the rather high acid content of solution.

The following is a summary of average results obtained by the Ant Process: Tons of ore leached, 75; heads, 1.68 per cent. Cu; extrac 77.6 per cent.; current density, 6.54 amp. per square foot; voltage, production of copper, 1.72 lb. per kilowatt-hour; $H_2SO_4$ in solu 2.73; sulphur per pound of copper, 1.06 lb.; total days run, 40.2; cathodes produced, 803 lb.

### Process of Pope and Hahn

ile these tests were being carried on, another process was being bed at Raritan by F. A. Pope and A. W. Hahn. The Antisell is in a sense a combination of diaphragm and depolarizer, without tempt to remove the iron. Pope and Hahn planned to remove a (and most of the other impurities as well) before sending solution lectrolyzed.

eir process was cyclic. Ore crushed to 4-mesh was leached for 3 ith dilute sulphuric acid, the counter-current leach being so dis-d as to permit of drawing off two classes of solution—one high in ie other low. The high iron solution was heated to about 200° F. ie addition of the requisite amount of finely ground (90 per cent. h 200-mesh) copper oxide in the form of roasted ore. The mixture tion and oxides was agitated for 3 or 4 hr. and then put through a ress. The result was the removal of nearly 90 per cent. of the total d 70 to 80 per cent. of the alumina from solution. An electrolyte copper and low in all impurities was produced and the precipi-s filter cake, was granular and easy to wash and handle.

e other portion of the leaching solution, low in iron, was electrolyzed urned to the cycle until its iron content was high enough to require ation.

sults of the Pope-Hahn process were very satisfactory, but the had hardly started when the war broke out and all experimental was stopped. The following summary will indicate the results ed: Ore leached, 329 tons, 10 charges; average copper heads, r cent.; copper tails, 0.30 per cent.; water, tails, 9 per cent.; copper owatt-hour, 1.0 lb. About 40 lb. of roasted Miami concentrate of ore handled took care of impurities; 92 per cent. of the copper concentrate was recovered.

rking the Pope-Hahn process would have required the installation ting plant, fine-crushing plant, agitators and filter presses and the se of high-grade sulphide ore or concentrates. It offered decided ages, among them a practically pure solution for electrolysis, er with the regeneration of a high percentage of the acid necessary ching.

### Greenway Process

ore the study of the effect of impurities had progressed very far, eresting fact was noticed. If neutral solution from the leaching was circulated through fresh ore, a large part of the iron in solu-as precipitated, presumably as a basic sulphate. This basic sul-was apparently not readily soluble in a solution containing more

acid. A possible method of purification from iron seemed to be indicated and the results of small-scale tests were so encouraging that a 1-ton plant was built. In the first tests, ore crushed to about ½ in. was leached for 6 days. The oldest charge of ore was in contact with fresh leach from the electrolytic cells. This leach carried about 3 per cent. of free $H_2SO_4$. Solution passed in counter-current through six tanks containing ore, and the acid of the leach was neutralized by the time it reached the fourth or fifth tank. In the last two tanks a neutral solution was in contact with fresh ore and here the iron was deposited.

The results obtained will be evident from the data of Table 2. The ferric iron was under good control for more than 100 complete cycles.

TABLE 1.—*Analysis of Average Cornelia Oxidized Ore*

| | |
|---|---|
| | Per Cent. |
| $SiO_2$ | 65.7 |
| $Al_2O_3$ | 15.3 |
| Fe total | 4.5 |
| CaO | 0.6 |
| MgO | 1.6 |
| MnO | 0.14 |
| Cu | 1.50 |
| | Oz. per Ton |
| Au | 0.005 |
| Ag | 0.20 |
| | Per Cent. |
| NaO | 3.6 |
| $K_2O$ | 4.6 |
| $P_2O_5$ | 0.3 |
| Cl | trace |
| $CO_2$ | 1.3 |
| | 99.04 |

At this time it was already practically settled:

1. That a good extraction could be obtained with dilute sulphuric acid.

2. That the acid consumption was not prohibitive.

3. That electrolytic copper could be produced successfully.

There remained, however, some important problems to be solved. Among these were:

1. Crushing; from two points of view: (1) for percolation in a deep bed; (2) for extraction.

2. Further study of the fouling of the solution, including the general accumulation of soluble salts, but especially those which might affect extraction or power efficiency in electrolysis.

3. Circulation and general handling on a small commercial scale.

4. Any factors other than those under (2), which might affect electrolysis itself or the quality of copper produced.

| | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Tails—per cent copper... | 0.64 | 0.29 | 0.43 | 0.32 | 0.24 | 0.24 | 0.19 | 0.17 | 0.24 | 0.27 |
| Extraction—per cent..... | 62.8 | 75.9 | 72.2 | 77.1 | 81.4 | 81.5 | 85.8 | 86.8 | 80.3 | 79.7 |
| Per cent. ore on 4-mesh.. | 29.0 | 47.0 | 43.4 | 28.0 | 33.0 | 25.0 | 26.0 | 25.1 | 35.1 | 39.0 |
| Days leached............. | 6.0 | 6.0 | 6.0 | 6.8 | 7.6 | 8.0 | 7.5 | 7.5 | 7.5 | 8.0 |
| Circulation—1-1 gal. per ton per min............. | 2.0 | 1.8 | 3.3 | 3.3 | 3.0 | 2.0 | 2.0 | 2.0 | 2.0 | 2.0 |
| Advance—1-1 gal. per ton per min................ | 0.42 | 0.40 | 0.17 | 0.20 | 0.20 | 0.16 | 0.16 | 0.20 | 0.16 | 0.29 |
| Electrolytic data: | | | | | | | | | | |
| Current density, amp. per sq. ft................. | 9.8 | 12.0 | 7.0 | 7.3 | 7.1 | 6.9 | 9.2 | 6.6 | 5.3 | 8.5 |
| Volts—drop in tank....... | 2.11 | 2.36 | 2.00 | 2.15 | 2.12 | 2.29 | 2.30 | 2.08 | 2.03 | 0.97 |
| Lb. copper per kw.-hr.... | 0.98 | 0.83 | 1.08 | 1.01 | 0.99 | 0.77 | 0.76 | 1.07 | 0.95 | 2.0 |
| Ave. free $H_2SO_4$ in cells.. | 4.2 | 3.6 | 2.8 | 2.7 | 2.9 | 2.9 | 3.3 | 3.4 | 2.8 | 3.8 |
| Ave. circulation in cells... | 3.8 | 4.6 | 4.4 | 4.2 | 5.8 | 7.0 | 8.2 | 7.5 | 7.8 | 5.0 |
| Ampere efficiency........ | 83.7 | 76.8 | 83.5 | 85.0 | 83.1 | 67.2 | 67.8 | 75.0 | 74.0 | 69.5 |
| Acid consumption: | | | | | | | | | | |
| Total lb. 100 per cent. $H_2SO_4$ used........... | 1,217.0 | 1,160.0 | 729.0 | 467.0 | 3,428.0 | 1,107.0 | 492.0 | 1,002.0 | 1,113.0 | 4,240.0 |
| Acid per lb. cathode Cu.. | 5.33 | 1.76 | 1.16 | 1.62 | 1.66 | 1.46 | 1.30 | 2.01 | 2.74 | 6.65 |
| Acid per lb. Cu leached.. | 3.02 | 0.94 | 1.06 | 1.36 | 1.61 | 1.24 | 1.31 | 1.86 | 1.83 | 2.30 |

NOTES.—Charges Nos. 1 to 19 had no crushing plant, so screened rejects from sampling oxide pits were used. Ore had excess fines and contained some sulphides. No filter bottoms in tanks. Segregation, started with new solution. Leaching started Aug. 8, electrolytic end started Aug. 17 with grid lead anodes.

Charges Nos. 20 to 74, on Oct. 6, tried reduction with liquid $SO_2$ using filtros bottom to distribute gas. Test stopped Oct. 14.

Charge No. 64.—All other charges normal. On Oct. 19, decided to discard one-half of solution volume, dilute to normal volume with well No. 1 water and make special run

Charges Nos. 75 to 100.—No $SO_2$ used and water from well No. 1 (high Cl) used on these charges.

Charges Nos. 101 to 116.—Same solution, longer leach, finer crushing variable conditions—No $SO_2$.

Charges Nos. 117 to 217.—On Dec. 14, $SO_2$ reduction tests were begun under the direction of F. L. Antisell. Lead-coke anodes were used. Tests discontinued Dec. 26—between charges 117 to 127. Leaching varied from 7 to 13 days. Charges Nos. 127 to 217 were run under normal conditions.

Charges Nos. 162, 163, 164 were high-grade ore—"k-10"-av. 2.7 per cent. Cu. These have not been included in the above averages. On Feb. 27, the average current density of 7 amp. per square foot was reduced to 4 amp. per square foot.

Charge No. 218.—April 19 was started with 40 tons high ferric iron solution, other conditions normal.

Charge No. 275.—June 17, arrangements made for another $SO_2$ run with lead anodes; same solution used. Test continued until charge 300.

Charges 300 to 330 run without $SO_2$ and under normal conditions.

Charge Nos. 341 to 413.—New solution, carbon anodes, $SO_2$ reduction, air agitation.

Ore used on charges Nos. 20 to 413 was from open cut and averaged 1.307 per cent. Cu.

The 1-ton plant was therefore continued in operation and a pl
handle 40 tons of ore per day was built.

It is fortunate that this larger unit was built, for a good many
that had seemed easy to do in the 1-ton plant refused to work at
the 40-ton. Fouling by iron and aluminum salts kept on increasin
ferric-iron content of the solution rose to a point where the ampe
ciency in electrolysis was poor, and various undesirable phen
appeared.

So the 40-ton plant was kept in continuous operation for ne
year and all the factors that had been studied in the 1-ton plant
taken up, one at a time, and worked over again until finally it seeme
they could all be brought under complete control.

## Final Leaching Process

The final process decided upon was as follows: Ore crushed to
4-mesh; leached 8 full days by counter-current; washed with
counter-current wash waters (or possibly four); solution, prac
neutral during last 2 days of contact with ore, is sent through red
towers, where it meets sulphur dioxide in counter-current. The
iron is thus reduced to below 0.4 per cent. Thence it goes thro
revolving tumbler in contact with cement copper. There the ferr
is still further reduced and a corresponding amount of cement
passes into solution. From the cement-copper tumbler the so
passes to a settling pond and then into the electrolytic cells, where
of the copper is removed; then back into the leaching system, p
first through the oldest ore which has already been leaching for 7
and so on to begin the cycle again.

## Important Factors

In a cyclic process such as this, no single factor or step can be s
be the most important. We can, however, consider a list of those f
that were studied with especial attention and can offer definite
on the results to be expected.

### *Extraction*

Both in the 1-ton and the 40-ton plants, it was easy to obtain
cent. extraction, using about 3 per cent. sulphuric acid and 8 d
counter-current leaching by upward percolation on 4-mesh ma
An extraction of 83 per cent. was reached over considerable peri
time. The continuous attainment of this figure was limited by
factors than the nature of the ore or the size to which it was crushed

fouling of solution, with the result that salts carrying copper crystallized in the ore in the leaching tanks and were not ely dissolved by washing.

mplete washing was, of course, another reason for low extrac- ler otherwise normal conditions, and the possibility of improve- this point is clearly shown by the wash-water data of Table 6. ing well within the time limit for washing set by the fixed cycle, d apparently be possible to get an extraction of 82 per cent. or

### *Acid Consumption*

1-ton plant, over its whole life of 413 cycles and under all sorts of e and unfavorable conditions, showed a net consumption of 1.65 0 per cent. $H_2SO_4$ per pound of copper leached. This was some- wer than the figures given by the preliminary tests on the ore. 40-ton plant apparently did not do so well. During its 301 he average net consumption was 2.8 lb. of acid per pound of eached. The difference in favor of the 1-ton plant is probably he large leakage and general losses in the 40-ton plant.

esigning the plant which is now being built at Douglas to furnish Cornelia, 3 lb. has been taken as a conservative figure for acid otion.

### *Power Consumption for Electrolysis*

e 1-ton plant, with lead anodes, the average was 0.934 lb. copper watt-hour at 8 amp. per square foot of cathode surface. This was e conditions, good and bad, under which the plant was operated. aphite anodes, also under all sorts of conditions, the figure was oper per kilowatt-hour at 8.5 amp. per square foot.

e 40-ton plant, with lead anodes, at an average current density mp. per square foot, the average production under all conditions lb. copper per kilowatt-hour. With graphite anodes, the figure 5 lb. It should be said, however, that the ferric iron was never roper control during these tests with graphite in the 40-ton plant. series VI of the 40-ton plant, with current density 6.4 amp. per oot, the copper yield was 1.04 lb. per kilowatt-hour. This run h lead anodes and under fair conditions, so far as ferric iron was ed. In the final run with lead anodes, and with both ferric iron eral fouling under still better control, an even higher figure was d.

anodes will be used in the big plant. The reasons for this choice given later.

TABLE 3.—*Summary of All 40-Ton Tests with Lead Anodes. Feb. 1 to Nov. 6, 1915*

| Charge No. (Inclusive)<br>Date | 1 to 81<br>Feb. 1–May 21 | 91 to 130<br>June 1–July 12 | 131 to 154<br>July 13–Aug. 4 | 155 to 170<br>Aug. 5–Aug. 22 | 171 to 201<br>Aug. 23–Sept. 25 | 202 to 238<br>Sept. 26–Nov. 7 | 1 to 238<br>Feb. 1–Nov. 7 |
|---|---|---|---|---|---|---|---|
| Average: | | | | | | | |
| Heads—per cent copper | 1.248 | 1.314 | 1.225 | 1.274 | 1.286 | 1.361 | 1.281 |
| Tails—per cent copper | 0.294 | 0.226 | 0.242 | 0.265 | 0.346 | 0.223 | 0.275 |
| Extraction—per cent | 76.4 | 82.8 | 80.2 | 79.2 | 73.1 | 83.6 | 78.5 |
| Per cent. ore on 4-mesh | 25.0 | 25.3 | 35.0 | 32.0 | 40.7 | 37.9 | 36.0 |
| Days leaching | 9.3 | 8.0 | 8.0 | 8.0 | 8.0 | 8.0 | 8.3 |
| Circulation—gal. per ton per min | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 |
| Advance—gal. per ton per min | 0.20 | 0.35 | 0.35 | 0.35 | 0.35 | 0.38 | 0.30 |
| Electrolytic data: | | | | | | | |
| Current density—amp. per sq. ft | 7.4 | 7.1 | 7.1 | 7.1 | 6.0 | 6.4 | 6.9 |
| Volts—drop in tank | 2.43 | 2.20 | 2.18 | 2.04 | 2.18 | 2.07 | 2.24 |
| Lb. copper per kw.-hr | 0.72 | 0.99 | 0.85 | 0.98 | 0.86 | 1.04 | 0.87 |
| Ampere efficiency | 66.8 | 83.3 | 70.8 | 76.5 | 71.7 | 82.3 | 74.0 |
| Lb. cathode copper | 49,775.0 | 30,652.0 | 17,431.0 | 10,521.0 | 18,273.0 | 23,717.0 | 150,349.0 |
| Acid consumption: | | | | | | | |
| Total lb. used—100 per cent | 188,484.0 | 98,442.0 | 47,484.0 | 37,197.0 | 85,173.0 | 102,402.0 | 559,182.0 |
| Lb. $H_2SO_4$ per lb. cathode Cu | 3.82 | 3.21 | 2.72 | 3.53 | 4.66 | 4.31 | 3.71 |
| Lb. $H_2SO_4$ per lb. Cu. in solution | 2.88 | 2.69 | 2.39 | 2.58 | 3.67 | 2.98 | 2.89 |
| Sulphur: | | | | | | | |
| Total lb. used | None | None | 4,693.0 | 5,434.0 | 18,767.0 | 26,697.0 | 55,861.0 |
| Lb. S per lb. cathode Cu | None | None | 0.27 | 0.52 | 1.02 | 1.12 | 0.80 |
| Lb. S per lb. iron reduced | None | None | 1.50 | 0.56 | 1.20 | 0.99 | 1.08 |
| | Old solution–No $SO_2$ first run | New solution—No $SO_2$ new run | $SO_2$ small tower acid electrolyte | $SO_2$ small tower neut. electrolyte | $SO_2$ large tower solution saturated | $SO_2$ large tower solution diluted | Av. of all 40-ton tests with Pb anodes |

NOTE.—Forty-ton plant started Feb. 1, 1915, with Pope-Hahn solution, tank house started Feb. 26. Control of ferric iron lost on Mar. 12

Charge No. 26, various attempts made to change iron ratio. Low tank-house efficiency caused copper content of leaching solution to increase to such an extent as to interfere with the extractions. Troubles with pumps, talcy ore, linings, etc. Solution of these first 81 charges discarded on May 21 and a new series of tests started, May 22, 1915. Charge No. 91, new solution was made up for this run. Electrolytic plant started June 1. Iron control lost

## *Control of Ferric Iron*

early in the study of the conditions for good electrolysis it
vident that the ferric-iron content of solutions must be kept
w limit. The 1-ton plant took care of itself over a long period
far as the ferric-iron content and the general fouling were con-
thout the use of sulphur dioxide or any other outside reagent.
lumina were apparently precipitated on the ore.
-ton plant started off in the same way. But before long the
began to go up and the ampere efficiency to go down. This
from the table showing general results on the 40-ton plant.
egan with fresh acid and lasted 40 days. During this period
u per kilowatt-hour was the electrolytic yield. The current
during this period of low ferric iron and clear solution was
ent. During period III the same solution was continued in
e ferric iron was not under control, and kept increasing. The
the solution on July 22, in the middle of this run, is shown in
I of Table 4. The ferric iron content had risen rapidly from
ent. on June 20 to 1.222 per cent. on July 22. The result was
g period III, the current efficiency was only 70.8 per cent. and
as 0.85 lb. copper per kilowatt-hour. From this time until the

TABLE 4.—*Solution Analyses*

Period of 40-Ton Plant Operation Beginning with Charge 91 in 1915

| | I 6–1, Per Cent. | II 6–20, Per Cent. | III 7–22, Per Cent. | IV 8–22, Per Cent. | V 9–23, Per Cent. | VI 11–5, Per Cent. |
|---|---|---|---|---|---|---|
| ............ | 1.11 | 2.26 | 2.50 | 3.39 | 3.47 | 2.95 |
| ............ | 4.03 | 2.88 | 2.56 | 3.15 | 2.40 | 3.05 |
| ............ | 0.09 | 0.176 | 0.720 | 1.49 | 2.14 | 2.52 |
| ............ | 0.164 | 0.180 | 1.220 | 1.07 | 0.80 | 0.239 |
| ............ | 0.145 | 1.250 | 2.730 | 3.69 | 4.40 | 2.840 |
| ............ | 0.092 | 0.090 | 0.080 | 0.050 | 0.120 | 0.018 |
| ............ | 0.132 | 0.122 | 0.120 | 0.150 | 0.049 | 0.054 |
| ............ | 0.023 | 0.030 | 0.030 | 0.050 | 0.046 | 0.029 |
| ............ | 0.08 | 0.08 | ..... | ..... | 0.130 | 0.090 |
| ............ | 0.075 | 0.084 | 0.110 | 0.040 | 0.960 | 0.096 |
| rity.......... | 1.100 | 1.150 | 1.250 | 1.370 | 1.376 | 1.330 |

I gives solution analysis at beginning of a run, where fresh acid has
nd before electrolysis has been started.
III is the analysis of solution when reduction of ferric iron was begun.
V shows solution at maximum concentration. Cold nights caused the
f a large amount of $FeSO_4$, $CuSO_4$, $24H_2O$.
VI is an analysis of solution under normal conditions, after "bleeding"
about 40 days.

end of the experimental work, sulphur dioxide gas was used regularly control the ferric iron. With lead anodes a low content of ferric iron not necessary. The solution may contain 0.3 or 0.4 per cent. without a marked effect on the current efficiency.

### *Sulphur Consumption*

To maintain satisfactory control of the ferric iron was, at this poi in the work, the greatest apparent difficulty. To follow the course of t tests it will be necessary to keep in mind both Tables 3 and 4. The fi of these shows the conditions under which the 40-ton plant was operati the latter, the analysis of the solution at various times.

During period III, the acid electrolyte was circulated through a tow about 2 ft. square, in contact with sulphur dioxide. The acid electrol was taken from the electrolytic system and returned to the same syste The weight of sulphur that could be burnt and brought in contact wi solution as $SO_2$ was too small, and the ferric iron kept on increasi Under these conditions, we were burning 1.5 lb. sulphur per pound of fer iron reduced.

Series IV began with only a single change, but this proved to b very important one. The *neutral* advance from the leaching system w circulated through the $SO_2$ tower before entering the electrolytic syste Under these conditions, only 0.56 lb. of sulphur was required per pou of ferric iron reduced.

Although the small tower was not of sufficient capacity to make c trol easy, by Aug. 22 the ferric iron was on the down grade—1.07 per ce as against 1.22 on July 22.

From this time on, as long as lead anodes were in use, the ferric i did not get out of control, and on Nov. 5 the solution showed only 0 per cent. of this troublesome substance. The results with graph anodes will be considered later.

### *Fouling of Solution*

In the 40-ton plant the entire solution began to foul badly after ab 100 charges had been leached. The cooler weather was beginning at t time and a bulky double sulphate of iron and copper crystallized through the ore in the leaching tanks and plugged all the pipes a launders. To remedy this, a portion of nearly neutral solution w drawn off continuously from the leaching system and passed over sc iron, making cement-copper. It was found that treating 1 per cent. the total bulk of solution each day would maintain the fouling at a c stant low point where it could cause no trouble. Columns V and of Table 4 show the effect of this "bleeding" of solution.

The cement-copper so produced was returned to a wooden tumbler and electrolyte on the way to the cells passed through it. Ferric iron is reduced to ferrous in this way and copper passes back into solution.

Under these conditions only electrolytic copper is produced and at the same time the control of the ferric iron is aided considerably. In the 40-ton plant it was apparently necessary to send about 12 per cent. of the total copper through this side cycle and back into the main one, to control completely the fouling materials introduced from the ore during leaching.

### *Grade of Copper Produced*

Cornelia copper produced direct from the ore is the equal of any electrolytic copper produced in this country. Fortunately the orebody contains only minute amounts of either arsenic; antimony or bismuth. The analysis and the physical tests on Cornelia cathodes will be found in Table 5.

*Table 5.—Analysis of Cornelia Copper Cathode*

| | Top, Per Cent. | Bottom, Per Cent. | | Top, Per Cent. | Bottom, Per Cent. |
|---|---|---|---|---|---|
| Cu | 99.900 | 99.868 | S | 0.0369 | 0.0655 |
| Fe | 0.0186 | 0.0291 | As | 0.0013 | 0.0017 |
| Cl | 0.0012 | 0.0019 | Sb | 0.00095 | 0.0014 |

*Physical Tests on Wire Bars*

| | |
|---|---|
| Tensile test | 64,500 |
| Elongation | 1.9 |
| Bends | 36 |
| Twists | 43 |
| Conductivity | 100.7 |

### *Size of Material. Percolation*

The work at Ajo was all done on the product from a Symons fine crusher, without any screening (except for tests on sized material). It was nominally a 4-mesh product but carried an average of about 30 per cent. of oversize. Extraction on this is satisfactory and while it could be increased by finer crushing, a large percentage of the copper minerals lies in fracture planes, and no very great profit would result.

This product is especially well suited to upward percolation in a bed 12 ft. deep or more. It does not segregate nor channel badly, drains well and is easy to wash.

### *Circulation in the Leaching System*

In the 40-ton plant, each of the eight active tanks had a closed circulation, upward through the ore, of about 80 gal. per minute. This rate

is probably much higher than necessary, but for the experimental w
was made ample and retained as a fixed factor. Beside this closed c
which runs continuously, a portion of the solution from each t
continuously advanced to the next tank in the series.

Fresh electrolyte from the cells, low in copper and of maximu
content, leaves the electrolytic system and passes into the leachin
containing the oldest ore—that which has been leaching for 7
Practically neutral solution, high in copper, leaves the leaching s
at the tank containing new ore. Between these two points, and
linked to them, is the advance from tank to tank.

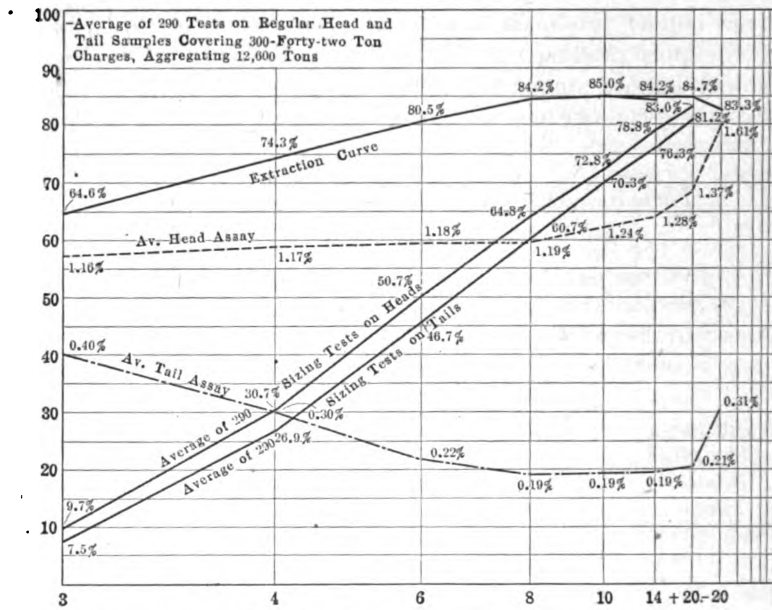


FIG. 1.—SIZING TESTS ON NEW CORNELIA OXIDIZED ORE SHOWING ALSO AV
ASSAYS ON HEADS AND TAILS AND AVERAGE EXTRACTIONS ON SIZED MATE

It is of great importance to have a practically neutral solutio
ing the last leaching tank and starting back toward the elect
system.

At the same time, we want as high an extraction as possible,
means as many tanks as possible with a fairly high acid co
To attain both these objects at the same time the advance from
to tank is carefully regulated so that about six tanks of the eight
ones are acid practically all the time. The two others are neutr
the point of neutrality of the system just balances, varying p
one tank in either direction during 24 hr., but returning to its o
position.

*Circulation in the Electrolytic System*

The electrolytic cells, together with tanks and sumps for storage, form a second closed circuit from which solution is bypassed continuously into the leaching system at the same rate as the advance from cell to cell. To produce good, coherent cathodes, the circulation through the cells must be much greater in volume than the advance from the leaching system, as the latter is rigidly fixed by the necessity for maintaining neutral leaching tanks. While the advance in the 40-ton plant was about 12 gal.

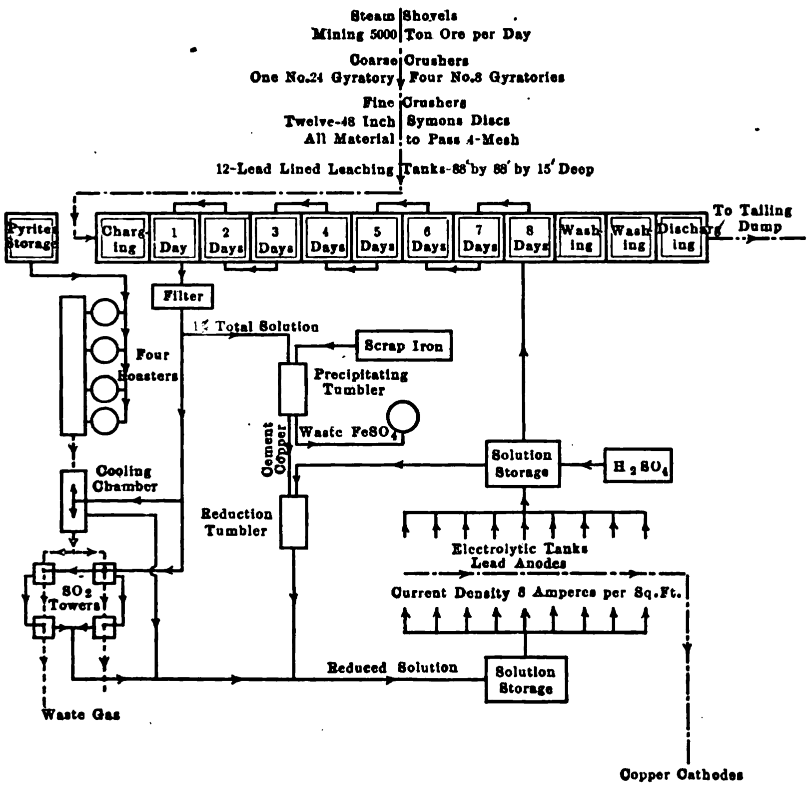


FIG. 2.—FLOW SHEET OF 5,000-TON PLANT OF NEW CORNELIA COPPER CO.

per minute, it was advantageous to circulate in the tank house at rates up to 150 gal. per minute.

This circulation gives satisfactory results with lead anodes, but when graphite is used, rapid circulation alone is not enough to give good depolarization at the anode. Violent agitation in the electrolytic tanks is necessary to attain the lowest voltage when graphite anodes are in use, and this was accomplished by blowing air through small holes in lead pipes laid along the bottom of the cell.

*Lead vs. Graphite Anodes*

In making final decision on the process to be used, there were m mechanical questions to be settled. Crushing, method of percolati circulation, pumps, structural materials, the handling of ore and taili and many other points came up for decision. Besides these, there w some questions which bear more directly on fundamental princip One of these was whether we should use lead or graphite as material anodes.

Lead was in service for over a year and its behavior was well un stood. But graphite promised much better results as far as power ciency is concerned, and also as a means of regenerating a much la part of the necessary acid for leaching.

The 1-ton plant gave most encouraging results with graphite ov period of continuous operation of nearly 3 months. With a curr density of 8.5 amp. per square foot, over 2 lb. of copper per kilowatt-h was produced, and during part of this time conditions were not go The ferric iron was not under complete control. When all conditi were right, we made 2.25 to 2.40 lb. copper per kilowatt-hour; on so days as high as 2.75 lb. This was during the warm weather at Ajo, w electrolyte temperatures held well above 105° F. The 40-ton plant using the large tower for reduction and the 1-ton plant used the sm one. The $SO_2$ used in controlling ferric iron was bypassed from main supply.

The 1-ton electrolytic plant was out of doors and while the odo $SO_2$ was strong in the immediate neighborhood of the cell, it was not enough to prevent working over it.

By the time the graphite anodes were installed in the 40-ton pla the average electrolyte temperature was only about 75° F. We co now determine accurately the efficiency of absorption and reduction $SO_2$ (which we could not do in the 1-ton plant) and we could also exam carefully the circulation conditions necessary to control ferric iron.

The problem now began to look harder. As measured by the prod tion of ferric iron from ferrous in the cell, the anodic efficiency of graph with rapid air agitation, is well over 100 per cent. With lead it is 30 35 per cent. We had therefore to combat about three times as m ferric iron per pound of copper with graphite as with lead.

Under the existing temperature conditions and with the rapid circu tion that was necessary in the electrolytic circuit, the reduction efficie of the $SO_2$ was only about 20 per cent. We could not control the fe iron without either allowing the electrolyte to stand for a good ma hours between the $SO_2$ towers and the cells, or else heating it to ab 150° F. during its passage from towers to cells.

The absorption of $SO_2$ in the towers was perfectly satisfactory. I

unless it was used to better advantage in the reduction of the iron in the solution, it was impossible to live in the tank house without either very careful ventilation or a diver's outfit. With the violent air agitation that was necessary for high power efficiency, most of the $SO_2$ which had been absorbed in the towers was blown out in the tank house.

These difficulties are by no means insuperable. Of more fundamental nature is the low current efficiency obtained with graphite and its extreme sensitiveness toward ferric iron. When ferric iron is practically completely removed by careful reduction with $SO_2$, graphite can be made to give high current efficiencies. But the very slightest rise in ferric-iron concentration results in an immediate and considerable lowering in ampere efficiency. Even in our good runs with graphite in the 1-ton plant, the current efficiency was not much above 70 per cent. With lead anodes, and without any particular attention to the ferric iron, which was usually 0.3 or 0.4 per cent., we often obtained current efficiencies of 90 per cent. or better over considerable periods.

It may be of interest to state some of the points that enter into a decision between the two kinds of anodes:

1. The first cost of an installation is about the same.
2. The life of lead is known with some accuracy. The life of graphite in this service is not known.
3. The salvage value of lead is high. With graphite it is practically nothing.
4. The probable power saving to be expected (graphite over lead) if graphite were successfully operated, is about 0.6 kw.-hr. per pound of copper.
5. More than twice as much acid is regenerated with graphite as with lead.
6. Circulation must be more rapid with graphite.
7. Agitation during electrolysis is absolutely necessary with graphite.
8. The solution must stand for some time (probably 16 to 24 hr.) or be heated to 150° F. or higher, to obtain even reasonable $SO_2$ efficiency with graphite. This is not necessary with lead (see note under 10).
9. Much larger towers are needed when graphite is used.
10. The volume to be pumped to the top of the towers is larger with graphite.

(These last two statements contain the fact that acid electrolyte must be sent through the towers. The iron content of the solution in the cells must not rise above 0.2 per cent., if good efficiency is to be reached with graphite. With lead anodes it may go to 0.4 or 0.5 per cent. without much of any effect on power consumption. With lead it is sufficient to send the neutral advance to the towers. With graphite not only this volume, but also a much larger volume from the tank-house circulation which must be kept low in ferric iron must be sent.)

11. The tank house for graphite must be carefully ventilated.

12. The tank house for graphite must be larger (about as 9 to 7
on account of the lower current efficiency.

13. Operation with graphite requires the closest attention to ever
detail of reduction, circulation and agitation. Operation with lead
practically fool-proof.

This last point is probably the most important of all. There ar
enough so-called "minor" troubles about a leaching plant.

*Control of Ferric Iron in Acid and Neutral Solution*

During the periods when acid electrolyte was being sent to the tow
for the reduction of ferric iron by sulphur dioxide, the sulphur efficien
was low. It proved to be about 20 to 25 per cent. Two causes at lea
combine to produce this result: First, the low solubility of $SO_2$ in ac
solution, and second, the slowness of the reduction reaction in ac
solution. Neutral solution was sent to the towers in later runs and
this the sulphur efficiency reached 75 to 80 per cent. That is to say,
the actual weight of sulphur burned to $SO_2$, 75 per cent. is utilized in t
reduction of ferric iron to ferrous.

*Washing the Ore*

Each charge was given three washes. The first was the second wash
the preceding charge; the second, the third wash of the preceding charg
the third, fresh water. The wash which had been used three times w
run into the electrolytic system. One complete fresh wash water w
just sufficient to balance the loss from the system in the tailings and
evaporation.

The summarized wash-water data are given in Table 6 for the sol
tion which was started in the 40-ton plant May 22, 1915. The mo
significant thing in this table is the effect of fouling as shown in the seco
column. While the solution was fresh, washing in the prescribed mann
the extraction was 81.1 per cent. and soluble copper left in the tailin
was 0.045 per cent. When the solution became foul, even though a lit
more fresh water was used, extraction dropped to 73.5 per cent. and t
soluble copper in the tailings was 0.103 per cent. Later, when fouli
was being reduced by "bleeding" and additional wash water could be us
in the same cycle, the extraction rose to 83 per cent. and the solub
copper in the tails dropped to 0.033 per cent.

It is of course possible to wash tailings with extra water, passing t
over scrap iron and making cement-copper, and this should naturally
done until the expense of operation balances the value of the recover
copper. The experiments at Cornelia were carried no further than
shown in this table.

All the experimental work in the 40-ton plant was carried out duri
continuous operation, so that the result should correspond as closely

with commercial conditions. The regular tonnage was crushed and charged into the leaching tank. Tailings were removed, and n, advance, washing, electrolysis and reduction carried on withak. Any change in operation was carried on for at least a week, lly for 10 days. If the change promised to give improved results, e a fixed part of the regular operating routine and the next point for study.

TABLE 6.—*Wash-Water Data*

| Charges Covered by Period | No. 91 to No. 170, May 22 to Aug. 20 | No. 171 to No. 205, Aug. 21 to Sept. 25 | No. 206 to No. 301, Sept. 26 to Jan. 10 |
|---|---|---|---|
| e treated | 3,376 | 1,435 | 4,051 |
| opper in the heads | 1.278 | 1.282 | 1.331 |
| opper in the tailings | 0.240 | 0.342 | 0.237 |
| xtraction | 81.1 | 73.5 | 83.0 |
| opper in washed tailings | 0.195 | 0.239 | 0.204 |
| extraction if tailings had been ean | 84.7 | 81.3 | 85.3 |
| oisture in tailings | 10.52 | 9.57 | 9.65 |
| umber of washes | 3 | 3 | 3 |
| ons new water used per ton of ore | 40.6 | 42.2 | 59.1 |
| trained in tailings per ton tailings d | 24.2 | 21.2 | 21.8 |
| accounted for per ton of ore leached, oration and leakage, etc. | 16.4 | 21.0 | 12.1 |
| solution bled per ton ore | None | None | 25.2 |
| avity of 8th day solution | 1.100 – 1.290 | 1.290 – 1.400 | 1.400 – 1.300 |
| avity of last wash | 1.050 – 1.060 | 1.060 1.150 | 1.150 – 1.050 |
| er cent. copper in first wash | 1.66 | 2.96 | 2.25 |
| er cent. copper in second wash | 1.17 | 2.17 | 1.45 |
| er cent. copper in third wash | 0.61 | 1.25 | 0.67 |
| er cent. $H_2SO_4$ in third wash | 0.35 | 0.35 | 0.35 |
| uble copper per ton tailings | 0.9 | 2.1 | 0.7 |
| xtraction if one more wash had been half of this soluble copper had been . Per cent. | 83.3 | 79.0 | 84.2 |

whole run:

| | |
|---|---|
| cent. copper in heads | 1.338 |
| cent. copper in tailings | 0.256 |
| cent. extraction | 80.7 |
| cent. extraction—according to washed tails | 84.1 |
| cent. extraction—if 4th wash had been used | 83.0 |
| cent. moisture in tailings | 10.0 |
| rage gallons of solution entrained per ton of tails | 22.0 |

Using this method, when we were through, we had a definite set conditions under which known results could be produced for any leng of time. There were no odds and ends to go back and pick up. N "minor" problems, which might turn out on investigation to be fund mental, remained to be solved. Mr. Greenway saw that this plan w strictly carried out.

During the experimental work, something over 180,000 lb. of perfe cathodes were produced, besides a good deal of cement-copper.

## Summary

So far as the Ajo tests can show what will happen in a 5,000-ton pla the following may be expected:

Extraction on 1.4 per cent. ore using closed wash cycle, 82 per ce

Extraction on 1.4 per cent. ore using extra wash water, 83 per cent

(The extraction figure used in making estimates has always be 80 per cent.).

Acid consumption (100 per cent. $H_2SO_4$), 3 lb. per pound of copp

"Bleeding" to prevent fouling of solution.

Power efficiency in electrolysis with lead anodes, 1 kw.-hr. p pound of cathode copper.

Sulphur consumption, 0.5 lb. per pound of cathode copper.

Neutral electrolyte only will need to pass through the reducti towers.

About 1 per cent. per day of the total solution volume must be se over iron to control fouling.

About 12 per cent. of the total copper produced will pass through t cement-copper side cycle.

The circulation used in the test-plant leaching tanks (2 gal. p minute per ton of ore) is undoubtedly much higher than necessary. It probable that a small part of this volume would give equally go extraction results.

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SSION OF THIS PAPER IS INVITED. It should preferably be presented in person at th
eting, September, 1916, when an abstract of the paper will be read. If this is impossible,
sion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West
New York, N. Y., for presentation by the Secretary or other representative of its author.
ial arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion
eafter should preferably be in the form of a new paper.

# Advent of Flotation in the Clifton-Morenci District, Arizona

BY DAVID COLE,* EL PASO, TEXAS

(Arizona Meeting, September, 1916)

he time flotation appeared upon the metallurgical horizon in , the writer, under the direction of Dr. Ricketts, was engaged in ing and enlarging the No. 6 Concentration Plant of the Arizona Co. at Morenci, and the work had been in progress nearly a year he Inspiration experiments with flotation had disclosed the revolu- concentration that was at that time impending.

simplified flowsheet worked out for the remodeling of the Morenci d been based upon the removal of the freed metal in a minimum of stages by treatment upon tables equipped with Butchart he latter being adapted to accomplish both classification of feed oval of the metal at one operation, substantially as described in ious paper.[1]

scheme of treatment for the slime was based upon the well- fact that after copper sulphides, such as chalcocite and chalcopy- ., are reduced to a certain extremely fine state of comminution definitely beyond the reach of separation upon any of the con- ng devices then known.

using drag-belt classifiers (which served as conveyors as well as ors), the overflow would be of the usual "slimes" class. Experi- ad shown that when these drag-belt overflows were properly (that is, to about 5 to 7 per cent. solids in the feed under treatment) fine sand, and especially the very fine but still granular sulphide s, would, if given a short distance to fall, quickly settle out in ondition to yield an excellent recovery on vanners; a further feature would be that the very fine non-separable final overflow might be discharged direct to tailing, thus conserving space nents so as to permit within the old building, without embarrass- he increase of capacity required.

reasons that are obvious, this kind of feed preparation for the slime ot be successfully accomplished in any form of pointed boxes or

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sulting Engineer.

elopment of the Butchart Riffle System at Morenci, *Trans.*, vol. 51, p. 405

spitzkasten device, therefore a further elaboration of the drag-belt idea was worked out as the best method for accomplishing the separation. This machine was known as a colloid separator and is shown in Figs. 1 and 2. It works on the premise that nearly all of what may be called

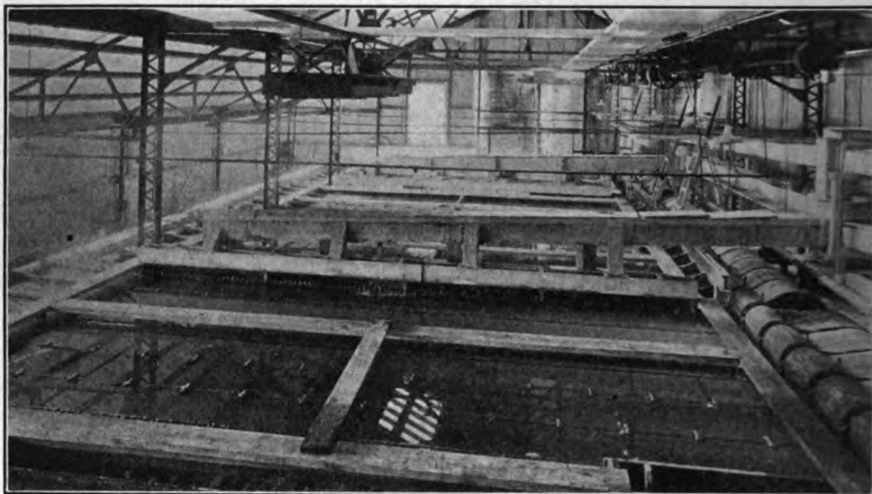


Fig. 1.—Drag-Belt Separators in Mill of Arizona Copper Co.

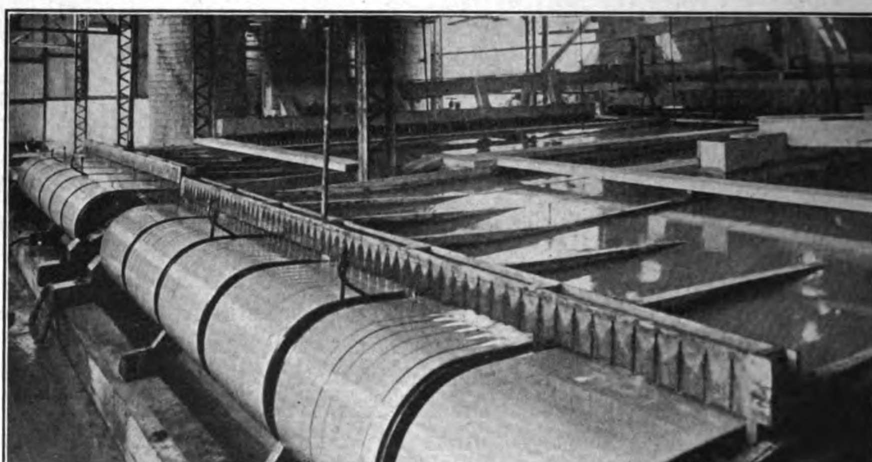


Fig. 2.—Another View of Drag-Belt Separators.

ponderable material in the thinned pulp, falling but 2 in., will lodge upon the belts and be quickly removed, while the flocculent slime material will remain in suspension and go away with the overflow. In this way a feed is prepared for the vanners containing a maximum amount of the very fine but granular sulphides and a minimum of colloidal material; at the

same time the drag-belt overflow product contains a minimum of sulphide particles and a maximum of flocculent slime or colloidal pulp. This final overflow was found in practice to be approximately two-thirds of the total tonnage handled by the belts and treatment upon vanners was, as far as copper recovery was concerned, devoid of beneficial results. The copper escaping in this overflow was in the form of extremely fine chalcocite, bornite with a little pyrite and chalcopyrite, together with oxidized, and water-soluble, copper salts. Taken together, these gave the overflow a copper tenor of from 1 to 1.3 per cent. in a ton of dry material, thereby

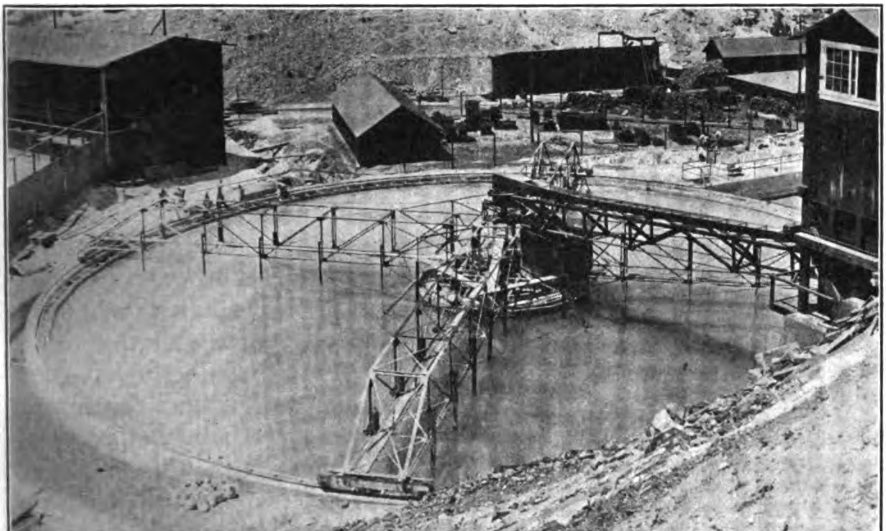


FIG. 3.—DORR THICKENER OF 130-FT. DIAMETER AT ARIZONA COPPER CO.'S PLANT.

accounting for the larger part of the tailing losses. However, this overflow, with its ultrafine and otherwise handicapped copper-bearing material, was practically beyond the reach of any concentrating machine at that time known, and could therefore go to tailing. A Dorr thickener[2] 130 ft. in diameter, the first one of such large size, was devised to recover the water from this overflow before it was allowed to go to waste. (See Fig. 3.)

By this plan, the treatable portion of the slime could be handled by the complement of vanners already installed in the company's mill. This arrangement, in conjunction with the saving in floor space, resulting from the introduction of the Butchart riffle, made it easily possible to double the capacity of the plant under practically the original roof. While it did not promise to recover a larger percentage than usual of the

[2] Described in *Engineering and Mining Journal*, Vol. 100, No. 4, p. 131 (July 24, 1915).

truly slimed copper, it did promise to give the best possible results in one-half the space.

By the time this flow sheet had been put into practical operation, the Inspiration experiments were attracting widespread attention and flotation was beginning to be taken seriously as a concentration agent for the handling of copper ores. It was, however, still regarded as an auxiliary process and was thought to be inapplicable to ores carrying an excess of talc or clay like the Clifton-Morenci ores. Some experiments with the Elmore process on these ores in former years had been unsuccessful and laboratory work which we did later on a very small scale with the meager information available seemed to corroborate this theory. Space was reserved, however, in a proper place in the mill building, for the installation of flotation equipment in case further development should prove its adaptability—at least for some appreciable portion of the tonnage.

Meanwhile, the Inspiration company had built the 600-ton "pilot" mill in which the new process was rapidly graduating from an auxiliary into the main method of separation, in the manner so fully and interestingly described by Dr. Gahl.[3] Starting with semi-mysterious compounds, Inspiration had soon found that simple flotation reagents were equally efficacious. Through the kindness of Mr. Mills, I secured a drum of cresylic acid and some pine oil with which to try a few experiments at Morenci.

The tailing from the No. 6 Concentrator was at that time discharged into Morenci Canyon and cascaded along for about 1 mile before being taken into a flume to be carried to the impounding dams. The creek bed was rough and steep, inducing great agitation of the pulp and resulting in the production of large amounts of white froth which floated down the stream. This froth carried no concentrations of copper minerals, but I thought it might be possible to change its character and produce a mineral froth by the use of flotation reagents introduced where the tailing left the mill, and thus possibly secure from the natural situation afforded some benefit at little cost.

A small can of the cresylic acid was arranged to drip into the tailings launder at a point where the tailings made the initial plunge into the creek bed. The results were instantaneous and very gratifying. Black froth began to collect in eddies and float downstream for a few yards to a second plunge where we were greatly surprised to find that it became white again on account of the instant dropping of the metallic load. Feeding the reagent into the stream immediately above the second plunge would not cause a mineral froth to rise as in the first plunge, and the reason was finally located as being the effect of a town sewer which was discharging under the surface into the creek between the two pools; the sewage effectively killed the metal-carrying capacity of the froth.

[3] *History of the Flotation Process at Inspiration,* in this *Bulletin.*

Cresylic acid was then added to the feed of a regrinding Hardinge mill which was discharging into a long drag-belt classifier. The results were again most encouraging; black mineral froth began immediately to appear and to collect in large volume upon the relatively still water in the drag-belt trough. This rough froth concentrate was found to assay over 45 per cent. copper; the product contained 35 per cent. insolubles, mostly in the form of coarse sand, mechanically suspended in the froth and easily separated by screening. It was found that 1 per cent. of the copper in the froth concentrate was in oxidized form; 22 per cent. of the concentrate was too coarse to pass a 100-mesh screen, and this portion carried only 0.87 per cent. copper, while the minus 100-mesh material carried 46 per cent. copper, 1.32 per cent. of which was oxidized, and but 16 per cent. of insolubles. The high grade of the froth concentrate was astonishing, showing that chalcocite and bornite predominated in it.

A few days later I made a bank of tube grates, consisting of six parallel 1-in. pipes made up with return bends. The pipes were drilled full of small holes and were wrapped with cotton blanket tied with spirally wound wire. This tube-grate air filter was put into the drag-belt trough as deeply as possible, without touching the belt. Coarse sand could pass through between the grates and be removed by the belt underneath. The pipes were supplied with compressed air for the purpose of creating additional froth, and it was found that the product made, without further treatment, assayed 40 per cent. copper, 1.14 per cent. of which was oxidized, and that it carried but 20.4 per cent. insolubles.

Plans for a small frothing machine of the mechanical-agitation type were immediately made, and on July 20 the apparatus was tried with the colloid separator fines as feed, with the following remarkable result: Feed, 2.32 per cent. copper, of which 0.62 per cent. was oxidized; the concentrates produced assayed 20.4 per cent. copper, of which 1.18 per cent. was oxidized; the tailing carried 0.52 per cent. copper, of which 0.38 per cent. was oxidized, leaving only 0.14 per cent. sulphide copper as the rejection of the machine. This was an extraction of more than 79 per cent. of the total copper, and more than 92 per cent. of the available (sulphide) copper. This showed clearly that much could be expected in the application of the new process to Morenci ores.

The Cananea Consolidated Copper Co., in Sonora, Mexico, had been experimenting with the use of some Flinn-Towne pneumatic flotation units in its concentrating department. The plant had been temporarily shut down on account of revolutionary troubles, and arrangements were made by Dr. Ricketts for the removal of one of these units to the No. 6 Concentrator at Morenci. The apparatus was installed under the direction of the Flinn-Towne people in the space reserved for flotation and was operated for several weeks with very gratifying results as to recoveries. These experiments demonstrated clearly that the flotation process

would be suited to the recovery of slimed copper sulphides in the Moren ores. But the Flinn-Towne units were not of a size suitable for use the equipment of a large plant, or for the handling of large tonnag except by using a great number of them. It was thought that th capacity could not be enlarged to advantage because of the difficul with the air-emitting medium used, which was in circular-disk for with central discharge. These disks could not be made larger in diamet

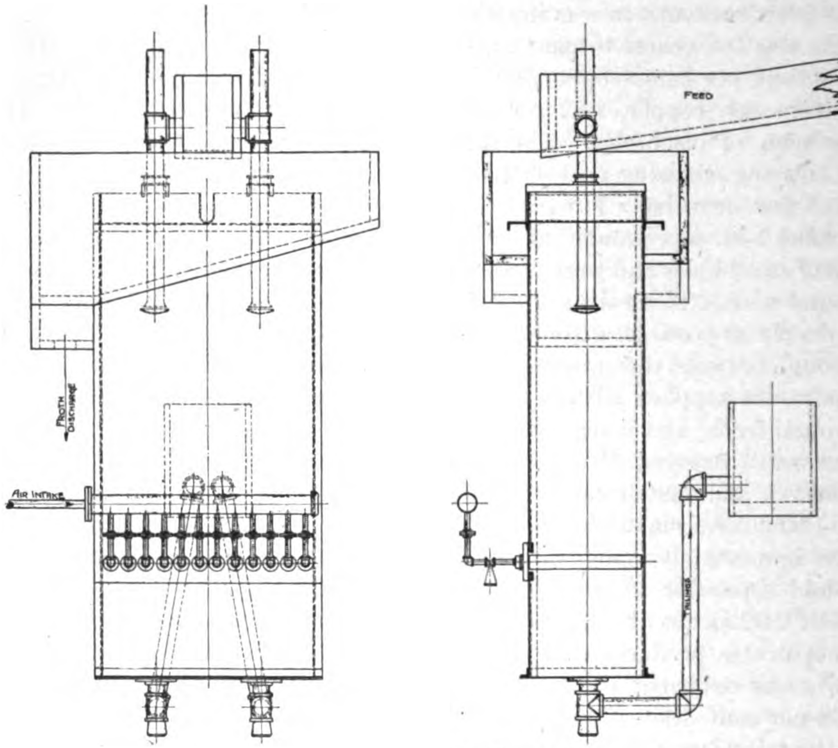



FIG. 4.—SIMPLE TUBE-GRATE CELL.

without increasing the difficulty coming from "blinding" of the a emitting surfaces, through the lodging of coarse particles upon the and from the formation of vortices by the larger volumes discharg through the single opening in the center which entrained froth with t reject.

The tube-grate idea previously tried in the drag-belt tank seemed be a better way to admit the air because nothing could lodge upon t air-emitting elements to blind them, and constriction of the passa for the pulp and water would be avoided. This tube-grate idea therefo formed a basis on which to design units of large capacity for practic

mill work and especially to solve the limited-space problem at No. 6 Concentrator. Accordingly, a tube-grate cell was installed, in January, 1915. This simple cell is shown in Fig. 4. It was used for some time to demonstrate the tube-grate idea and served as a "cleaner" in the subsequent work done with a full-sized machine.

The demonstration of the new tube-grate cell was such a success that a three-stage machine, to have a capacity of 400 tons per day, called the "C-B" machine, was designed and made. Another one of the same kind and size was made concurrently for the Inspiration Consolidated Copper Co., and both of them were started in operation early in March. The Inspiration machine, which is illustrated in Dr. Gahl's paper, gave good results, proving the design to be substantially correct. The Morenci machine was working in corrosive water, which resulted in the formation of rust on the steel tubes and the gradual closing of the openings. The air supply was found to be contaminated with grease and oil from the blower bearings, entrained muddy water in the air, etc., which closed the pores of the air filter from the inside. Some delay was experienced in overcoming these difficulties. That they were of minor importance was proved by the almost uninterrupted success and approximately perfect operation of the duplicate machine operating concurrently at Inspiration.

The C-B machine at Morenci made large volumes of very rich froth and had become immediately profitable by reason of its being able to handle a large tonnage and save copper which would be otherwise beyond the reach of concentration. It was therefore kept in operation as a profit maker, even though working under the handicap of partially clogged tubes, blower troubles, etc. The daily tonnage handled during the month of April, 1915, was from 125 to 390 tons per day, with an average of 209; the recovery made by the machine was from 35 to 79 per cent. of the sulphide copper present in the feed, with an average for the period of 65 per cent.

The blower used was an old one borrowed from the mining department, where it had been used in ventilation. It was designed for not more than 3½ lb. pressure, and the developing of 6 lb. in it deflected the shafts and caused the impellers to rub upon the sides of the machine, which had to be water-jacketed to keep down the heat developed. It was much larger than necessary, and a great excess of air was blown off from open valves. Therefore, no record of the amount of air used or power required, could be even approximately obtained.

In spite of these minor difficulties it was proved: That flotation could be applied to these ores with great advantage; that the copper in the mill tailing could be reduced to 0.50 per cent. (of which 0.25 to 0.30 per cent. was oxidized and beyond the reach even of flotation); that this result could be improved by finer grinding in the Hardinge mills; that the

TABLE 1.—*Detailed Results of Tests on C-B and Callow Flotation Machines for Month of July, 1915.*

SCREEN ANALYSIS, C-B MACHINE

| Mesh | Machine Tails | | | | | | | Machine Concentrates | | | | | | | Vanner Tails | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | |
| | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidized Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent. of Total | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidized Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum Per Cent. of Total | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidized Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent. of Total |
| + 20 | ...... | ...... | ...... | .... | ...... | ...... | ...... | ...... | ...... | ...... | .... | ...... | ...... | ...... | ...... | ...... | ...... | .... | ...... | ...... | ... |
| + 28 | 1.1 | 1.1 | 0.77 | .... | 0.85 | 1.41 | 1.41 | ...... | ...... | ...... | .... | ...... | ...... | ...... | 1.4 | 1.4 | 0.66 | .... | 0.92 | 2.11 | 2.11 |
| + 35 | 3.8 | 4.9 | 0.65 | .... | 2.47 | 5.51 | 5.51 | ...... | ...... | ...... | .... | ...... | ...... | ...... | 4.4 | 5.8 | 0.60 | .... | 2.64 | 6.04 | 8.15 |
| + 48 | 8.0 | 12.9 | 0.79 | .... | 6.32 | 10.49 | 16.00 | 0.5 | 0.5 | 2.22 | .... | 1.11 | 0.04 | 0.04 | 9.9 | 15.7 | 0.60 | .... | 5.94 | 13.61 | 21.76 |
| + 65 | 9.9 | 22.8 | 0.80 | .... | 7.92 | 13.14 | 29.14 | 0.6 | 1.1 | 14.72 | .... | 8.83 | 0.32 | 0.36 | 11.0 | 26.7 | 0.51 | .... | 5.61 | 12.85 | 34.61 |
| +100 | 14.7 | 37.5 | 0.87 | .... | 12.79 | 21.23 | 50.37 | 1.6 | 2.7 | 15.82 | .... | 25.31 | 0.90 | 1.26 | 16.6 | 43.3 | 0.48 | .... | 7.97 | 18.25 | 52.86 |
| +150 | 11.2 | 48.7 | 0.79 | .... | 8.85 | 14.69 | 65.06 | 7.9 | 10.6 | 20.16 | .... | 159.26 | 5.64 | 6.95 | 10.9 | 54.2 | 0.37 | .... | 4.03 | 9.23 | 62.09 |
| +200 | 6.9 | 55.6 | 0.67 | .... | 4.62 | 7.67 | 72.73 | 10.6 | 21.2 | 23.99 | .... | 254.29 | 9.08 | 16.03 | 6.3 | 60.5 | 0.37 | .... | 2.33 | 5.34 | 67.43 |
| −200 | 44.4 | 100.0 | 0.37 | .... | 16.43 | 27.27 | 100.00 | 78.8 | 100.00 | 29.83 | .... | 2,350.60 | 83.97 | 100.00 | 39.5 | 100.0 | 0.36 | .... | 14.22 | 32.57 | 100.00 |
| Total... | 60.0 | ...... | ...... | .... | 60.25 | 100.00 | ...... | 100.00 | ...... | ...... | .... | 2,799.40 | 100.00 | ...... | 100.0 | ...... | ...... | .... | 43.66 | 100.00 | ...... |
| | | | | | | | | | | | | Fe | Ins. | | | | | | | | |
| Check... | ...... | ...... | 0.61 | 0.24 | ...... | | ...... | ...... | ...... | 27.84 | 1.44 | 19.8 | 18.8 | ...... | ...... | ...... | 0.41 | 0.25 | ...... | ...... | ...... |

| Mesh | Vanner Concentrates | | | | | | | Table Tails | | | | | | | Table Concentrates | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | |
| | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidized Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum Per Cent. of Total | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidized Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent. of Total | Per Cent. Wt. | Cum Per Cent. Wt. | Per Cent. Total Cu. | Per Cent. Oxidized Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum Per Cent. of Total |
| + 20 | 0.2 | 0.2 | 2.02 | .... | 0.40 | 0.04 | 0.04 | ...... | ...... | ...... | .... | ...... | ...... | ...... | 0.7 | 0.7 | 1.53 | .... | 1.07 | 0.11 | 0.11 |
| + 28 | 0.6 | 0.8 | 3.63 | .... | 2.18 | 0.24 | 0.28 | 0.9 | 0.9 | 0.65 | .... | 0.59 | 1.26 | 1.26 | 1.0 | 1.7 | 4.23 | .... | 4.23 | 0.45 | 0.56 |
| + 35 | 1.6 | 2.4 | 7.26 | .... | 11.64 | 1.28 | 1.56 | 3.1 | 4.0 | 0.66 | .... | 2.05 | 4.36 | 5.62 | 2.0 | 3.7 | 7.75 | .... | 15.50 | 1.66 | 2.22 |
| + 48 | 4.7 | 7.1 | 9.08 | .... | 42.68 | 4.70 | 6.26 | 7.2 | 11.7 | 0.60 | .... | 4.32 | 9.20 | 14.82 | 6.3 | 10.0 | 9.79 | .... | 61.68 | 6.59 | 8.81 |

SCREEN ANALYSIS CALLOW MACHINE

| Mesh | Machine Tails | | | | | | | Machine Concentrates | | | | | | | Vanner Tails | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | |
| | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu. | Per Cent. Oxidised Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent. of Total | Per Cent. Wt. | Cum Per Cent. Wt. | Per Cent. Total Cu. | Per Cent. Oxidised Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent. of Total | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidised Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. Total | Cum. Per Cent of Total |
| + 20 | ...... | ...... | ...... | .... | ...... | ...... | ...... | ...... | ...... | ...... | .... | ...... | ...... | ...... | ...... | ...... | ...... | .... | ...... | ...... | ...... |
| + 28 | 0.8 | 0.8 | 0.94 | .... | 0.75 | 1.26 | 1.26 | ...... | ...... | ...... | .... | ...... | ...... | ...... | 0.4 | 0.4 | 0.61 | .... | 0.24 | 0.61 | 0.61 |
| + 35 | 1.2 | 2.0 | 0.81 | .... | 0.97 | 1.63 | 2.89 | ...... | ...... | ...... | .... | ...... | ...... | ...... | 1.8 | 2.2 | 0.53 | .... | 0.95 | 2.40 | 3.01 |
| + 48 | 5.9 | 7.9 | 0.81 | .... | 4.78 | 8.03 | 10.92 | ...... | ...... | ...... | .... | ...... | ...... | ...... | 7.1 | 9.3 | 0.49 | .... | 3.48 | 8.81 | 11.82 |
| + 65 | 10.4 | 18.3 | 0.87 | .... | 9.05 | 15.21 | 26.13 | 0.6 | 0.6 | 5.14 | .... | 3.08 | 0.48 | 0.48 | 11.4 | 20.7 | 0.49 | .... | 5.59 | 14.15 | 25.97 |
| +100 | 16.8 | 35.1 | 0.87 | .... | 14.62 | 24.57 | 50.70 | 1.7 | 2.3 | 13.20 | .... | 22.44 | 3.50 | 3.98 | 17.7 | 38.4 | 0.54 | .... | 9.56 | 24.20 | 50.17 |
| +150 | 12.5 | 47.6 | 0.80 | .... | 10.00 | 16.80 | 67.50 | 8.8 | 11.1 | 19.15 | .... | 168.52 | 26.25 | 30.23 | 12.1 | 50.5 | 0.44 | .... | 5.32 | 13.47 | 63.64 |
| +200 | 7.8 | 55.4 | 0.65 | .... | 5.07 | 8.52 | 76.02 | 10.1 | 21.2 | 24.29 | .... | 245.33 | 38.22 | 68.45 | 7.1 | 57.6 | 0.29 | .... | 2.06 | 5.22 | 68.86 |
| −200 | 44.6 | 100.0 | 0.32 | .... | 14.27 | 23.98 | 100.0 | 78.8 | 100.0 | 25.70 | .... | 2,025.16 | 31.55 | 100.00 | 42.4 | 100.0 | 0.29 | .... | 12.30 | 31.14 | 100.0 |
| Total... | 100.00 | ...... | ...... | .... | 59.51 | 100.0 | ...... | 100.0 | ...... | ...... | .... | 2,464.53 | 100.0 | ...... | 100.0 | ...... | ...... | .... | 39.50 | 100.0 | ...... |
| Check.. | ...... | ...... | 0.66 | 0.23 | ...... | ...... | ...... | ...... | ...... | 24.80 | 1.24 | Fe 19.4 | Insol. 24.6 | ...... | ...... | ...... | 0.37 | 0.15 | ...... | ...... | ...... |

| Mesh | Vanner Concentrates | | | | | | | Table Tails | | | | | | | Table Concentrates | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | | Solids | | Assay | | | Copper | |
| | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidised Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum Per Cent. of Total | Per Cent. Wt. | Cum Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidised Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent. of Total | Per Cent. Wt. | Cum. Per Cent. Wt. | Per Cent. Total Cu | Per Cent. Oxidised Cu | Per Cent. Wt. × Per Cent. Cu | Per Cent. of Total | Cum. Per Cent of Total |
| + 20 | 0.4 | 0.4 | 2.42 | .... | 0.97 | 0.15 | 0.15 | ...... | ...... | ...... | .... | ...... | ...... | ...... | ...... | ...... | ...... | .... | ...... | ...... | ...... |
| + 28 | 0.6 | 1.0 | 1.81 | .... | 1.09 | 0.17 | 0.32 | 0.5 | 0.5 | 0.88 | .... | 0.44 | 1.10 | 1.10 | 0.8 | 0.8 | 3.67 | .... | 2.94 | 0.34 | 0.34 |
| + 35 | 1.4 | 2.4 | 3.12 | .... | 4.37 | 0.66 | 0.98 | 1.3 | 1.8 | 0.64 | .... | 0.83 | 2.08 | 3.18 | 0.5 | 1.3 | 5.91 | .... | 2.96 | 0.35 | 0.69 |
| + 48 | 3.3 | 5.7 | 6.35 | .... | 20.96 | 3.18 | 4.16 | 4.2 | 6.0 | 0.57 | .... | 2.39 | 5.98 | 9.16 | 5.2 | 6.5 | 9.59 | .... | 49.87 | 5.82 | 6.51 |
| + 65 | 6.0 | 11.7 | 10.40 | .... | 67.40 | 9.48 | 13.64 | 10.4 | 16.4 | 0.47 | .... | 4.89 | 12.25 | 21.41 | 17.7 | 24.2 | 9.59 | .... | 169.74 | 19.80 | 26.31 |
| +100 | 16.0 | 27.7 | 12.81 | .... | 204.96 | 31.13 | 44.77 | 16.0 | 32.4 | 0.44 | .... | 7.04 | 17.63 | 39.04 | 40.0 | 64.2 | 8.06 | .... | 322.40 | 37.61 | 63.92 |
| +150 | 15.0 | 42.7 | 12.10 | .... | 181.50 | 27.57 | 72.34 | 11.9 | 44.3 | 0.47 | .... | 5.59 | 14.00 | 53.04 | 21.8 | 86.0 | 8.57 | .... | 186.83 | 21.80 | 85.72 |
| +200 | 12.1 | 54.8 | 6.76 | .... | 81.80 | 12.42 | 84.76 | 7.8 | 52.1 | 0.44 | .... | 3.43 | 8.59 | 61.63 | 8.2 | 94.2 | 8.87 | .... | 72.73 | 8.48 | 94.20 |
| −200 | 45.2 | 100.0 | 2.22 | .... | 100.34 | 15.24 | 100.0 | 47.9 | 100.0 | 0.32 | .... | 15.32 | 38.37 | 100.0 | 5.8 | 100.0 | 8.57 | .... | 49.71 | 5.80 | 100.0 |
| Total... | 100.0 | ...... | ...... | .... | 658.39 | 100.0 | ...... | 100.0 | ...... | ...... | .... | 39.93 | 100.0 | ...... | 100.0 | ...... | ...... | .... | 857.18 | 100.0 | ...... |
| Check.. | ...... | ...... | 6.40 | 2.13 | ...... | ...... | ...... | ...... | ...... | 0.45 | 0.25 | ...... | ...... | ...... | ...... | ...... | 7.84 | 1.44 | Fe 21.0 | ...... | ...... |

TABLE 2.—*Comparative Results Obtained in Operatian of C-B and Callow Flotation Machines at Concentrator No. 6, Arizona Copper Company, Ltd., Morenci, Ariz.*

May, 1915

| | | C-B Per Cent. | Callow, Per Cent. |
|---|---|---|---|
| Flotation machine tails | Total copper | 0.72 | 0.81 |
| | Oxidized copper | 0.38 | 0.30 |
| | Sulphide copper | 0.34 | 0.51 |
| Flotation machine concentrates | Total copper | 38.19 | 24.82 |
| | Insolubles | 25.00 | 27.00 |
| Vanner tails | Total copper | 0.53 | 0.46 |
| | Oxidized copper | * .... | 0.18 |
| | Sulphide copper | * .... | 0.28 |
| Vanner concentrates | Total copper | 8.52 | 8.89 |
| | Insolubles | * .... | 30.60 |

June, 1915

| | | C-B Per Cent. | Callow, Per Cent. |
|---|---|---|---|
| Flotation machine tails | Total copper | 0.71 | 0.69 |
| | Oxidized copper | 0.30 | 0.28 |
| | Sulphide copper | 0.41 | 0.41 |
| Flotation machine concentrates | Total copper | 35.24 | 25.41 |
| | Insolubles | 23.80 | 26.00 |
| Vanner tails | Total copper | 0.41 | 0.42 |
| | Oxidized copper | 0.23 | 0.22 |
| | Sulphide copper | 0.18 | 0.20 |
| Vanner concentrates | Total copper | 9.35 | 9.51 |
| | Insolubles | * .... | * .... |

July, 1915

| | | C-B, Per Cent. | Callow |
|---|---|---|---|
| Flotation machine tails | Total copper | 0.61 | 0.66 |
| | Oxidized copper | 0.24 | 0.23 |
| | Sulphide copper | 0.37 | 0.43 |
| Flotation machine concentrates | Total copper | 27.84 | 24.80 |
| | Insolubles | 18.80 | 24.60 |
| Vanner tails | Total copper | 0.41 | 0.37 |
| | Oxidized copper | 0.25 | 0.15 |
| | Sulphide copper | 0.16 | 0.22 |
| Vanner concentrates | Total copper | 7.96 | 6.40 |
| | Insolubles | * .... | * .... |
| Table tails | Total copper | 0.41 | 0.45 |
| | Oxidized copper | 0.24 | 0.25 |
| | Sulphide copper | 0.17 | 0.20 |
| Table concentrates | Total copper | 8.96 | 7.84 |
| | Insolubles | 43.60 | * .... |

* No assay.

| | C-B | Callow |
|---|---|---|
| Daily tonnage rate, average for the month of July, 1915 | 479 | 139 |

SEE NOTE, p. 1621.

aple reagents used elsewhere would apply. Further, a new type natic flotation machine, well adapted to the conditions at No. 6 rator, had been successfully developed and its performance rated on a full-sized unit. The new type machine would not y way embarrassed by the oversize coming from Hardinge mills.

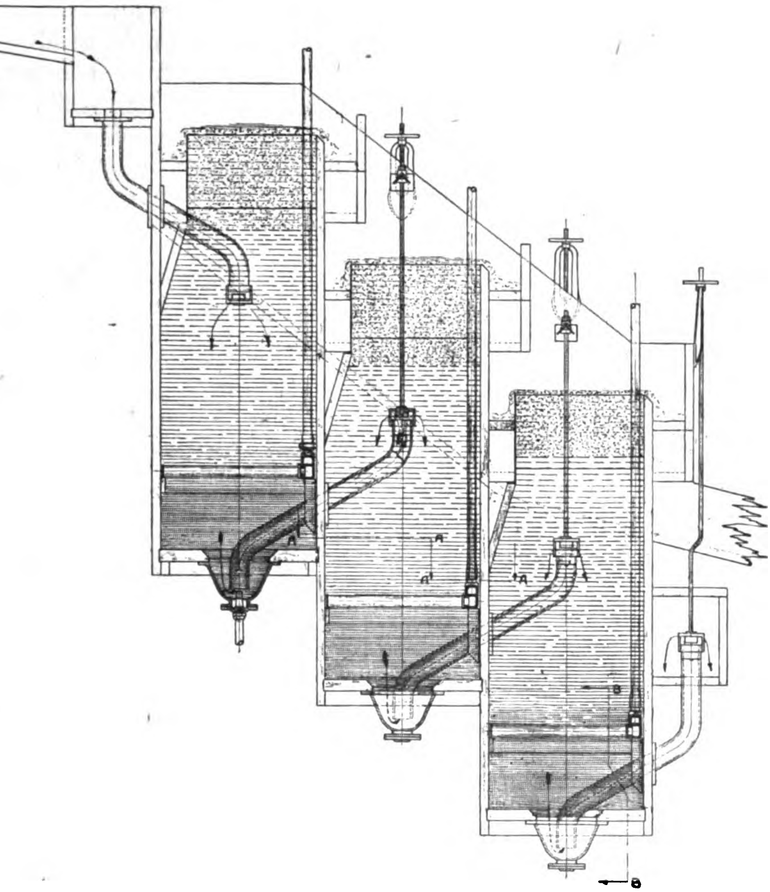


FIG. 5.—LONGITUDINAL SECTION OF C-B FLOTATION MACHINE. THREE CELLS IN SERIES.

he drag-belt overflow being handled by the colloid separator to a few of these new machines where all of the rich slimed sulpper would be taken out. Or the whole tonnage of reground

TO TABLE 2.—Without protection against "blinding" of the air-emitting sand or "oversize" the C-B machine handled considerably more than three tonnage handled by the Callow in July, and did this without detriment tallurgical work.

material produced in the Hardinge mills could go directly to the new type frother in which the slimed copper sulphides would be removed. The thoroughly frothed sands could then be treated on tables and vanners for the removal of the remaining sulphide particles which were too coarse to be separated by flotation. Since there would then be no embarassing losses in the slime part of the feed, these machines would work efficiently, and a maximum mill recovery would result.

After it became evident that flotation would apply to Morenci ores and before the value of the tube-grate idea was fully demonstrated, it was decided to install and experiment with a standard Callow flotation unit of 200 tons capacity, consisting of four rougher cells and one cleaner. This equipment was not received until after the full-sized C-B unit of 400 tons daily capacity had been installed.

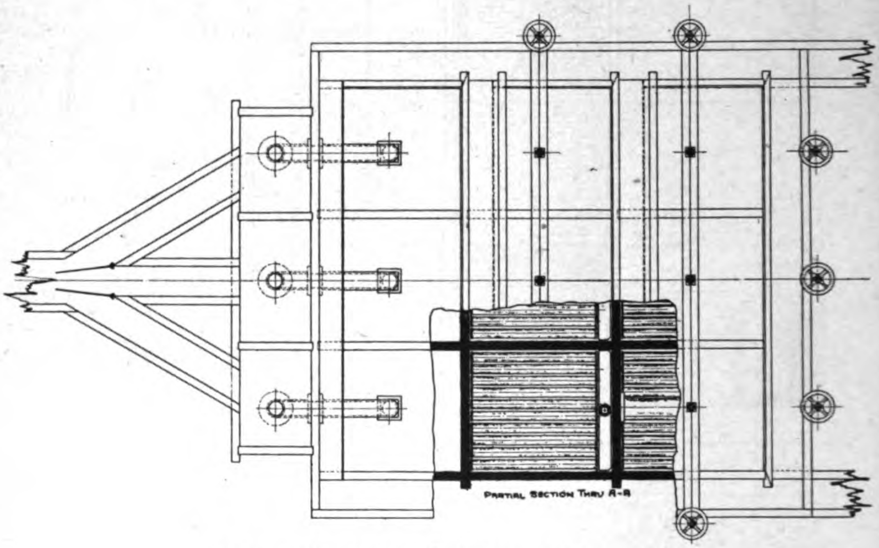



FIG. 6.—PLAN OF C-B FLOTATION MACHINE.

The Callow equipment was started in operation May 24, 1915, and competitive operation was carried on for about 3 months. The recoveries proved to be very much alike although the feed was not identical. The Callow apparatus is not adapted to handle coarse particles of feed or oversize, and had to be protected by a screen or spitzkasten. It will handle about one-half the normal tonnage of the C-B machine, occupying the same mill space. A summary of the results obtained for the months of May, June, and July, 1915, also details showing the work for the month of July, are given in Tables 1 and 2. The performance is shown to be substantially parallel as to quality of work done, but quite different as to quantity of tonnage handled.

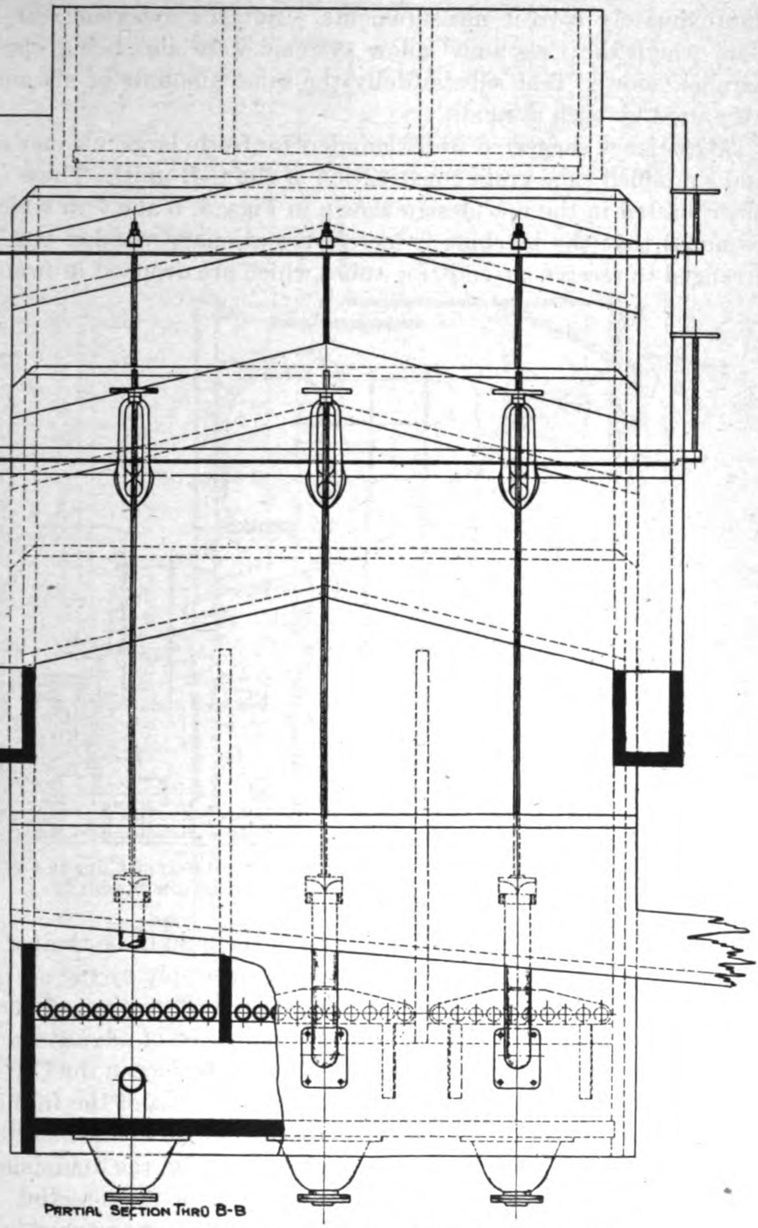



FIG. 7.—FRONT ELEVATION OF C-B FLOTATION MACHINE.

As mentioned before, the air and power consumption was not determined in the C-B installation because there was no way to take even approximately correct measurements. But the experience at Inspiration, where the C-B and Callow systems were also being operated in parallel, showed that substantially the same amounts of air and power were used by each system.

Experience suggested wider launders for froth, larger cleaner capacity, and simplified tube-grate construction of the C-B unit. These ideas are incorporated in the new design shown in Figs. 5, 6 and 7 in which it will be noted that the machine is merely a stationary wooden box suitably arranged to receive air-emitting tubes which are dropped in from the top

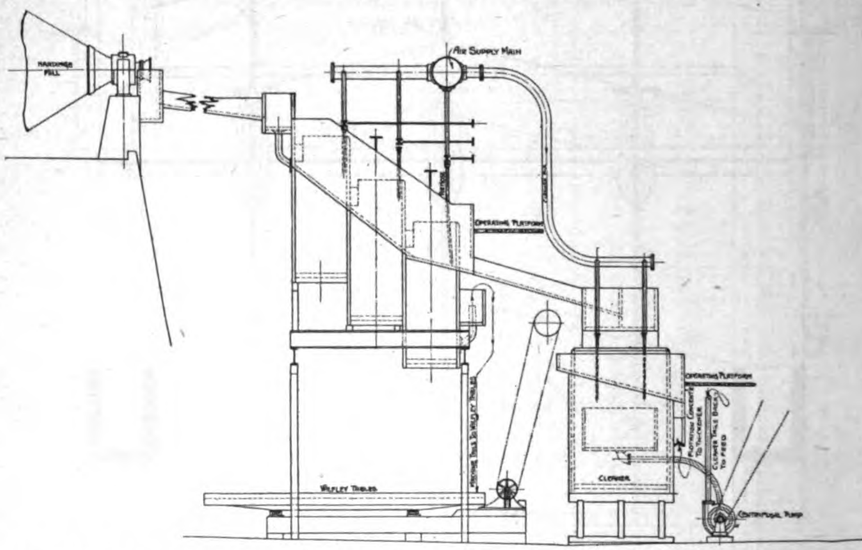



Fig. 8.—Arrangement at Mill of Cananea Consolidated Copper Co., Cananea, Mex., Using the C-B 3-Stage Flotation Machine.

and rest upon ledges at the proper level in the pulp to be treated. These air-emitting elements are connected to the air supply by the use of rubber hose. They can be taken out or put in without cutting off the feed or shutting down the machine, in case this should be of advantage. An air pressure of 5 lb. is required. The air-emitting surface in the C-B machine is more than twice the complete cross-sectional area of the frothing compartments, and even if a ridge of sand should lodge upon the extreme top of the tubes and partially cut off the air supply, the remaining unobstructed area would still be larger than the whole cross-section.

Fig. 8 shows the improved form with cleaners as arranged in the mill of the Cananea Consolidated Copper Co., and is typical of the arrangement adopted for the later models. There are eight units in this installation. They are operated under conditions varying greatly as to quality

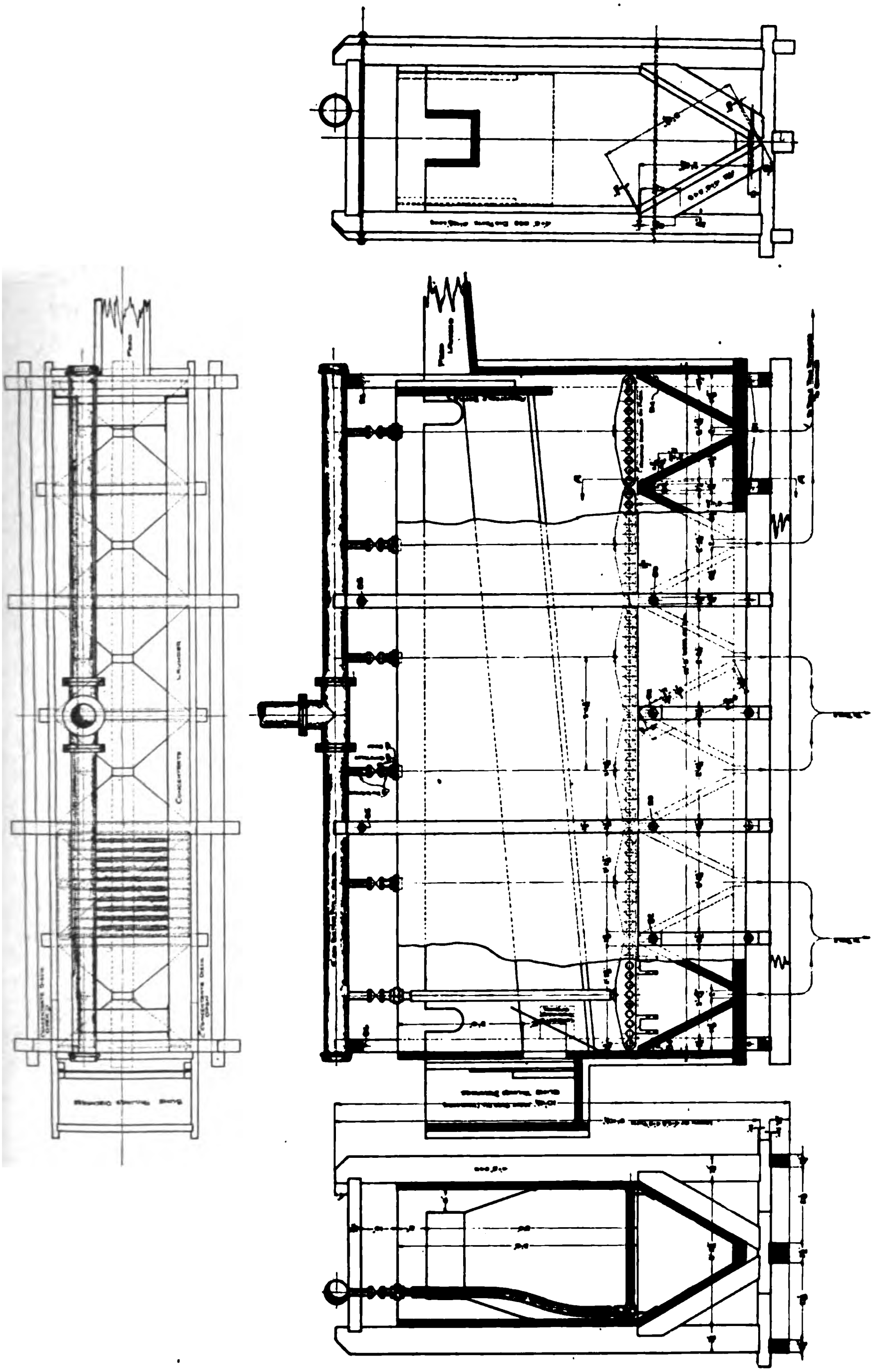


FIG. 9.—SPITZKASTEN FROTHER EQUIPPED WITH TUBE-GRATE AIR FILTER.

of ores, there being a wide range of iron and copper sulphide condition and all conditions of oxidation in the ore handled. The operation of th Cananea machines has been repeatedly interrupted by the internal strif in Mexico and no deductions of value are at present available.

Fig. 9 shows an application of the tube-grate idea to a spitzkaste type of frother. One of these machines was made and installed in No. Concentrator at Morenci but was taken out before it was tried. I seems to embody advantages of much promise in a frothing first flow sheet and will soon have a trial to determine its value in the simplifie concentration of ore that is amenable to flotation.

USSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the eeting, September, 1916, when an abstract of the paper will be read. If this is impossible, ussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 Street, New York, N. Y., for presentation by the Secretary or other representative of its Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. ussion offered thereafter should preferably be in the form of a new paper.

# History of the Flotation Process at Inspiration

BY RUDOLF GAHL,* PH. D., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

## CONTENTS



etallurgist in charge of Concentrator, Inspiration Consolidated Copper Co.

The history of flotation in America is very short, at least as far as th large-scale application of the process is concerned. It is remarkable hov many important developments have taken place in the last few years an are already being extensively utilized. What was new a year ago, seem almost commonplace now. For this reason it is with hesitation that follow the suggestion of Dr. Ricketts, President of the Institute, t describe the experiences of the Inspiration Consolidated Copper Co. wit the flotation process.

## Tests Conducted in Small Test Mill

When the Inspiration company first decided to build a concentratin plant, nothing was known about flotation, and the process was to b gravity concentration pure and simple.

### *Demonstration Tests Conducted by Minerals Separation Co.*

While plans were being prepared by H. Kenyon Burch (who had bee intrusted with the design and construction of the concentrator) th Minerals Separation Co., a concern at that time little known in Americ asked and obtained permission to demonstrate the value of its flotatio process for Inspiration ore. As a consequence, a small 50-ton flotatio machine of standard design was added to the company's test plant an started to operate in the beginning of 1913. This marks the beginning c flotation at Inspiration.

The results obtained with this machine, which was operated b members of the Minerals Separation staff, so far surpassed what th company anticipated that it was decided to continue flotation tests for th purpose, and two of the flotation experts of the Minerals Separation Co I. L. Greninger and G. A. Chapman, were retained. L. R. Wallace, no superintendent of the Miami works of the International Smelting Co was at that time metallurgist of the Inspiration company and in tha capacity took an active part in these tests. Great credit is due to hin for his quick recognition of the possibilities of flotation.

### *Flotation Tests Conducted by Inspiration Co.*

The tests led to the conclusion that it would be advisable to incor porate flotation into the concentrating process. Doubt existed only a to the extent to which this should be done. The first design brought ou by Mr. Burch only called for flotation treatment of the concentrato tailings. While the tests were progressing, it became more and mor evident, however, that flotation should form a more essential part of th milling process, and it was finally decided to leave out all the complica

vhich are usually adopted in gravity concentration plants for the e of recovering as much as possible of the mineral values of the and to rely on flotation alone for this purpose.

*Sampling of Orebody*

is decision was reached only after it had been established by ous tests that the orebody as a whole was suited for flotation treat- It was found, it is true, that a portion of the ore in the mine was able for flotation by the process in question, but the amount of e was established to be only a small fraction of the total. Besides, ts proved that the ore contained in this fraction lost its refractory ter when mixed with the rest of the ore.

. Greninger gives the following description of the manner in which ests were carried out:

ample lots, each amounting to about 10 tons and representing from 35 ft., were blasted from the back and sides of the drifts and after e crushing and grinding treated in the small flotation machine ned above.

he results were very erratic, some showing good recovery with a rade of concentrate, while others returned concentrate of a very low and also showed a low extraction.

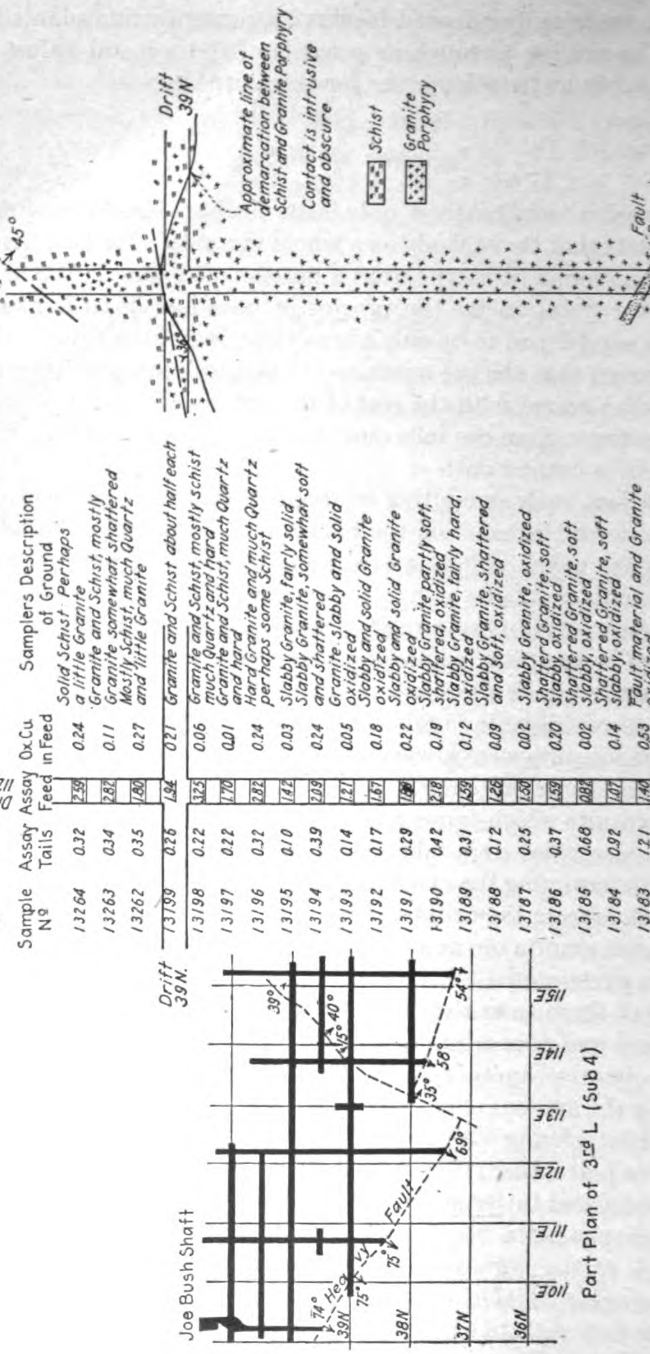
fter a considerable number of tests had been made, it was found ne good results were always obtained when treating ore of a schist , while poor results resulted when the gangue was the altered and zed granite which forms a part of the Joe Bush orebody.

en-ton samples were taken from various parts of the mine for the e of determining the extent and amount of this granite ore. These re taken from the drifts crossing the orebody from south to north, g in the granite ore and continuing along each drift until the schist was encountered.

all of these tests the results were uniformly good while treating ist ore and poor while treating the granite ore.

ubsequently, another series was inaugurated with the view of ining the amount of granite ore that could be mixed with the schist hout interfering with the treatment of this mixed ore by flotation. arious percentages of granite ore were mixed with clean schist and xture treated by flotation. It was found that 10 per cent. of granite no change in the behavior of the ore in the flotation plant and when cent. of the granite was mixed with the schist ore only a slight nce was noted, this difference consisting in a slight lowering of the of the concentrate with a corresponding falling off in extraction."

e conditions described by Mr. Greninger are well illustrated by a g made by C. E. Arnold of the mine-engineering staff of the com-




| Sample No. | Assay Tails | Assay Feed | Ox. Cu. in Feed | Samplers Description of Ground |
|---|---|---|---|---|
| 13264 | 0.32 | 2.59 | 0.24 | Solid Schist Perhaps a little Granite |
| 13263 | 0.34 | 2.82 | 0.11 | Granite and Schist, mostly Granite somewhat shattered |
| 13262 | 0.35 | 1.80 | 0.27 | Mostly Schist, much Quartz and little Granite |
| 13199 | 0.26 | 1.94 | 0.21 | Granite and Schist about half each |
| 13198 | 0.22 | 3.25 | 0.06 | Granite and Schist, mostly schist much Quartz and hard |
| 13197 | 0.22 | 1.70 | 0.01 | Granite and Schist, much Quartz and hard |
| 13196 | 0.32 | 2.82 | 0.24 | Hard Granite and much Quartz perhaps some Schist |
| 13195 | 0.10 | 1.42 | 0.03 | Slabby Granite, fairly solid |
| 13194 | 0.39 | 2.09 | 0.24 | Slabby Granite, somewhat soft and shattered |
| 13193 | 0.14 | 1.21 | 0.05 | Granite slabby and solid oxidized |
| 13192 | 0.17 | 1.67 | 0.18 | Slabby and solid Granite oxidized |
| 13191 | 0.29 | [illegible] | 0.22 | Slabby and solid Granite oxidized |
| 13190 | 0.42 | 2.18 | 0.19 | Slabby Granite partly soft, shattered, oxidized |
| 13189 | 0.31 | 1.59 | 0.12 | Slabby Granite, fairly hard, oxidized |
| 13188 | 0.12 | 1.26 | 0.09 | Slabby Granite, shattered and soft, oxidized |
| 13187 | 0.25 | 1.50 | 0.02 | Slabby Granite, oxidized |
| 13186 | 0.37 | 1.59 | 0.20 | Shatterd Granite, soft slabby, oxidized |
| 13185 | 0.68 | 0.82 | 0.02 | Shattered Granite, soft slabby, oxidized |
| 13184 | 0.92 | 1.07 | 0.14 | Shattered Granite, soft slabby, oxidized |
| 13183 | 1.21 | 1.40 | 0.53 | Fault material and Granite oxidized |

Part Plan of 3rd L (Sub 4)

vhich represents laboratory flotation results on samples taken drift in the Joe Bush orebody (Fig. 1). The results show clearly e granite by itself does not interfere with flotation, but only the aterial, evidently corresponding to what Mr. Greninger terms ed granite.

### *Other Results Obtained in Small Test Mill*

test conducted in the small test plant established in a general physical conditions under which Inspiration ores could be treated ageously; for instance, it was decided that raising the temperature improve the results obtained in proportion to the extra expense procedure.

ar as flotation oils are concerned, those in charge of the tests came onclusion that cresylic acid (98 per cent. pure) should be used as n flotation agent, and should be supplemented by crude turpen- As the most important result brought out by these tests, I con- e discovery that the flotation agents may profitably be added in ding machines, while it had formerly been the customary practice the "oils" in agitating tanks especially provided for the purpose. iscovery had an important bearing on the later developments nspiration milling practice, inasmuch as it paved the way for the nuch heavier oils; for instance, coal tar which it is impossible to nate thoroughly with the pulp in agitating tanks. Mr. Chapman, made this important discovery during this period (U. S. Patent 74).

details of the flow sheet to be followed were left to be decided by a larger scale. Only this much was settled: That no concen- of any kind, either flotation or gravity, should be attempted before was reduced to the fineness required for the flotation process. cision was brought about by the fact that excellent recoveries were d when this procedure was followed, and was, of course, also urged by considerations of simplicity in the milling process and ss of milling operations.

hese tests a 50-ton flotation machine of Minerals Separation Co. d type was used. In the general design, it was similar to the nachine later put in use. It had eight agitating compartments opellers of 12-in. diameter revolving with a peripheral speed of 200 to 1,400 ft. per minute.

## Tests in Large Test Mill

### *Flow Sheet*

rder to test on a large scale the points already settled in the small , and to decide the points left undecided by the small-scale experi-

ments, a 600-ton test plant was built and put into operation at the begi ning of January, 1914. It was my privilege to conduct those tests cooperation with Mr. Greninger, who represented the Minerals Separati Co., and with the representatives of other concerns, who in the course time decided to submit flotation machines to the consideration of t Inspiration company. The flow sheet of this 600-ton test mill was e tremely simple. It is illustrated in Fig. 2. The ore after being crush to the desired fineness by machinery, which was tested out at the sa time, was sent to tables; from these the concentrates went to the conce trate bins, while the tailings passed on to an 8-compartment flotati

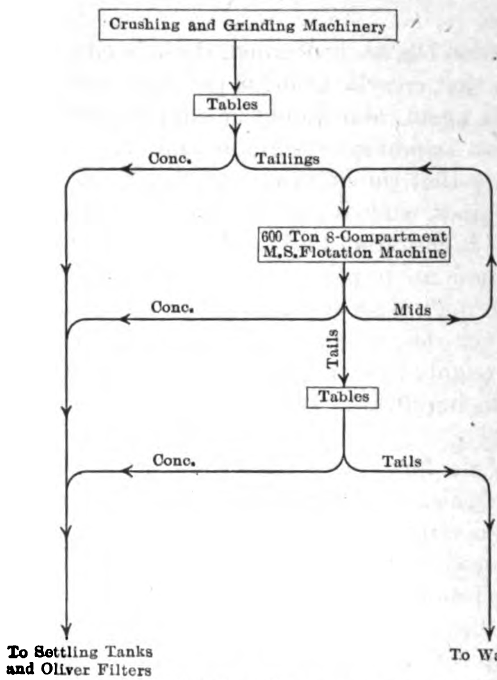



FIG. 2.—FIRST FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATE

machine of the Standard Minerals Separation type with 24-in. stirre Later a 12-compartment machine of the same type was added a operated in parallel with the 8-compartment machine. The tailings fro the flotation machines were passed on to other tables, the concentrat from which, combined with the concentrates from the upper tables a the flotation machine, went to the concentrate bins, while the tailin went to waste. The flotation machine was operated in the standa manner, the feed being introduced into the first agitating compartme passed to the first spitzkasten, drawn to the second agitating compar ment, sent from there to the second spitzkasten and drawn to the thi

agitating compartment, etc., the agitators forming the necessary suction for the transportation of the pulp from the spitzkastens to the following agitating compartments. A final concentrate was made from one or more spitzkastens, while the concentrate from those remaining was considered a middlings product and was returned to the head of the machine for retreatment.

This flow sheet was extremely simple, but after a while even this was considered too complicated, inasmuch as the necessity of table treatment ahead of flotation treatment was doubted.

### *Discussion of the Value of Preliminary Table Treatment*

The advantages pointed out in favor of the preliminary table treatment were about as follows: Since the tables would make a certain recovery, an impoverished flotation feed would result and assist the flotation machine to make a tailings product low in copper. However, the validity of this argument was doubted for the following reasons:

In the first place, if complications are to be avoided, the preliminary table treatment has to be of a comparatively rough character; the refinements of hydraulic classification have to be dispensed with, as by its use more water would be introduced into the pulp than would be advisable for the following flotation process. It must be taken into consideration that the pulp delivered to the tables already contains about 3 tons of water to 1 ton of solid matter, experience having shown that the grinding machines, consisting of either ball or pebble mills, deliver a product of the desired fineness, about 1 or 2 per cent. on 48-mesh, when the consistency of the overflow from drag-belt classifiers, working in conjunction with the grinding mills, was carried at about this figure; experience also showed that this consistency is suitable for the flotation treatment, while greater dilution of the pulp resulted in an increased copper loss. The logical way out of this difficulty would be to introduce settling tanks, for which, however, the modern mill designer has a just abhorrence; at least, when they are to be placed in the middle of the mill. Neither the liberal use of dressing water nor a low tonnage rate would be permissible for the same reason. On this account, a high recovery could not be expected from a preliminary table treatment. During our tests it averaged around 33 per cent.

Furthermore, the assumed fact, that an impoverished feed results in a better recovery of the flotation machine, was strongly doubted by some of the flotation experts, who claimed that in order to form a froth which had the necessary carrying power for mineral, it would be better to leave as much mineral as possible in the feed to the flotation machine. In other words, their advice was to leave out the tables in order to permit the flotation machines to do more efficient work.

There is no doubt, however, that tables will catch a certain amou of mineral (especially coarser grains) which will escape in the flotatic process. For this reason, tables have to be used to insure the highe recovery,[1] but it seems that tables following the flotation treatme would make up for this deficiency of the flotation machine better tha tables ahead, inasmuch as they would not work under the disadvantag known to every millman, of receiving too rich a feed. It is well know that it is possible to make a much lower table tailing working on fe containing a small percentage of copper than on a feed rich in copp From theoretical speculations, therefore, no valid reason was advanc that increased recovery should result from table treatment ahead the flotation treatment.

The next argument advanced by the supporters of a preliminary tab treatment was that it would result in an improvement of the grade the general concentrates, inasmuch as it would be possible to make a ve clean product on the tables, thereby raising the general average of t concentrates. There is evidently some sound foundation for this poin how much, could not well be investigated without resorting to actu experiments.

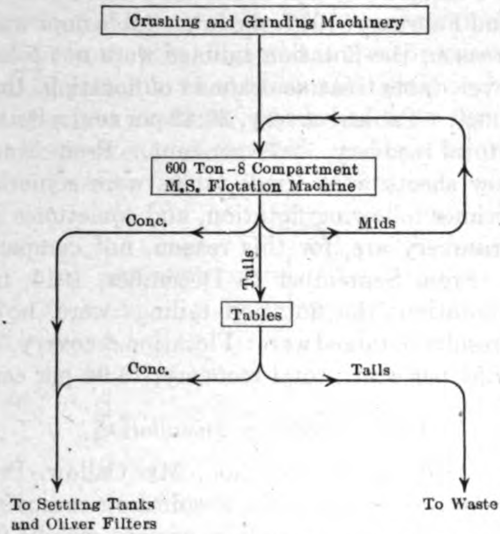
A third point seemingly in favor of preliminary table treatment this: Flotation concentrates offer more or less difficulty in mechanic handling; vacuum or pressure filters have to be resorted to for th purpose. It was pointed out, therefore, that if a certain percentage the total concentrate could be saved on the tables, and a table conce trate produced containing only a small amount of the slime, so troubl some in filter treatment, it might be a decided advantage; the quantity concentrate to be handled on filters might be materially reduced an economies effected in this manner. At the same time, experience show that the flotation concentrate resulting under those conditions, becau it contains less of the coarser sand, is more difficult for the filters to hand than it would be were the sand left in. Very likely, therefore, no decid improvement in the handling of the concentrates would result from th separation of the table concentrates from the flotation concentrates, least under the conditions prevailing here.

The same objections do not hold true for tables when used after th flotation process, as in that case no limitations are imposed regardin the hydraulic and dressing water, and the tables cannot help but mak an additional small recovery of concentrate. On account of the fact th the table feed is necessarily low in copper, the tailings will be corr spondingly low. In all cases, therefore, where flotation treatment do not make a full recovery, or practically so, of mineral values, tablin after flotation seems imperative.[2]

[1] This applies only to the flow sheet under discussion.

[2] This remark again refers only to the flow sheet under discussion.

ough such theoretical considerations were indulged in, no deci- based on them. To settle the main points in question, a series was carried out. On alternate days, the "upper" tables were by- Fig. 3) while on the remaining days, they were utilized (Fig. 2).




SECOND FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATED.

1.—*Comparison of Efficiency of Flow Sheet No. 1 (Fig. 2) and Flow Sheet No. 2 (Fig. 3)*

| Description | Flow Sheet No. 2 | Flow Sheet No. 1 |
|---|---|---|
| mill feed | 1.72 | 1.67 |
| flotation machine feed | 1.72 | 1.32 |
| flotation machine tails | 0.46 | 0.43 |
| ower table tails | 0.29 | 0.30 |
| general concentrates | 32.71 | 31.72 |
| upper tables + flotation machine | ...... | 75.30 |
| flotation machine | 74.30 | |
| l recovery on lower tables | 9.40 | 7.40 |
| overy | 83.70 | 82.70 |
| per deck on lower tables | 25.70 | 23.20 |
| per deck on upper tables | ...... | 87.00 |
| mption, cresol + turpentine in pounds per ton | 0.95 | 0.82 |

RKS.—The recoveries are calculated from the feed and tailings assays and of general concentrates.

LUSIONS.—Roughing on upper tables reduces the tailings assay of the flota- ine slightly, while it does not seem to affect the assay of the lower tables ly. Increased recovery results from the use of the lower tables.

Table 1 gives the résumé, on the strength of which it was decided use secondary tables only and to leave the preliminary table treatme (ahead of the flotation treatment) out of the flow sheet to be adopt in the concentrator under construction. That this conclusion is jus fied seems to be borne out by the following facts:

In January and February, 1914, the lower table floor was not in oper tion. For this reason, the flotation tailings were not treated on tabl There was, however, table treatment ahead of flotation, the following sults being obtained: Table recovery, 39.42 per cent.; flotation recover 33.35 per cent.; total recovery, 72.77 per cent. From March to Augu 1914, various flow sheets were tried; tables were sometimes used p ceding and sometimes following flotation, and sometimes in both plac The figures on recovery are, for this reason, not comparable with t preceding ones. From September to December, 1914, no tables we used preceding flotation; the flotation tailings were, however, treat on tables. The results obtained were: Flotation recovery, 70.83 per cen table recovery, 7.51 per cent.; total recovery, 78.34 per cent.

### *Callow Flotation Installation*

While the test mill was in operation, Mr. Callow, President of t General Engineering Co., advised the Inspiration company that he h invented and perfected a new flotation process which, in his opinio would give as good, if not better, results than the machine of the Miner Separation Co. As a consequence, arrangements were made to add unit of machines of the Callow type. The flow sheet using Call cells is illustrated in Fig. 4. It consisted of four rougher flotation cel which served for the production of a low-grade concentrate, and anoth cell of the same construction which was supplied for the purpose cleaning the concentrates made on the rougher cells. The tailings resul ing from the cleaner cells were returned and mixed with the pulp enteri the rougher cells. With the machine was furnished a Pachuca tank f the purpose of mixing flotation oil into the pulp entering the plant, shou it be more desirable to do this in addition to or in place of feeding the to the grinding machines, as had proven useful both in the small and the larger test plants. As a matter of fact, the use of the Pachuca ta was abandoned after several months' operation.

The Callow cells work on a different mechanical principle from th underlying the Minerals Separation Standard machine, inasmuch as movable parts are used for the purpose of producing the fine dissemin tion of air with the pulp, which seems to be essential for any kind flotation machine; while the Standard Minerals Separation machi depends on the operation of fast-revolving impellers, the disseminati is effected in the Callow machine by blowing air through the pores of porous blanket, which forms the bottom of each cell.

The Callow machine, illustrated in another paper, is ingeniously simple. The feed enters at the upper end of the inclined bottom, while the tailings are discharged through the float valve at the lower end, the concentrates overflowing at the top of the flotation tanks. The porous bottom of the cell is set on an incline to make possible the treatment of feed containing coarse sand. The movement of such sand particles from the feed to the discharge end is thereby accelerated.

The mechanical principle of aeration and agitation by the admission of air through a porous medium, which has been termed "pneumatic flotation," was undoubtedly discovered by Mr. Callow[3] and his asso-

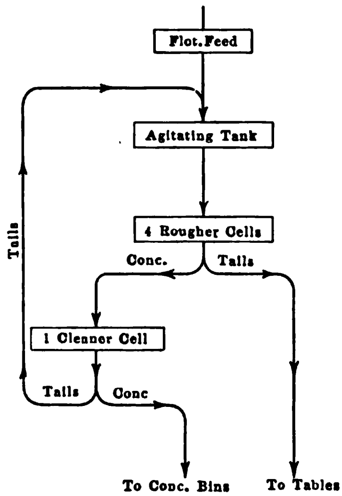



FIG. 4.—FLOW SHEET OF EXPERIMENTAL CALLOW FLOTATION PLANT.

ciates independently of other inventors, although an earlier patent had been taken out for the same thing in England by Minerals Separation (British Patent 10,929, May 3, 1910) and long before the installation of the Callow machine at this mill, Inspiration ore had been tested in New York by a pneumatic flotation machine constructed jointly by Messrs. Flinn and Towne.[4] The results of this test were so good that a small test plant of this system was installed at Inspiration.

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[3] Mr. Callow refers to his installation at the National Mill, Mullan, Idaho, April, 1914, as the first successful commercial plant utilizing the pneumatic principle.

[4] This test was made in the presence of Dr. L. D. Ricketts, Consulting Engineer, and W. D. Thornton, Vice-President of the Inspiration company. According to F. B. Flinn, a 600-ton plant of the Flinn-Towne flotation system was shipped to the Tezuitlan Copper Co. in Mexico in the spring of 1913. This could not be put into operation on account of the political conditions prevailing then. Otherwise, this would have been the first commercial pneumatic flotation plant. R. C. Canby informs me that some tests carried out by him suggested the idea to Messrs. Flinn and Towne of constructing a flotation machine to produce the desired fine dissemination of air by blowing air through a porous medium.

### *Flinn-Towne Installation*

The Flinn-Towne machine, as mentioned, utilizes the pneuma principle also, but the application is somewhat different. An illustr tion of a single Flinn-Towne machine is given in Fig. 5, while Fig. shows the outline of the installation. The cells are constructed in t

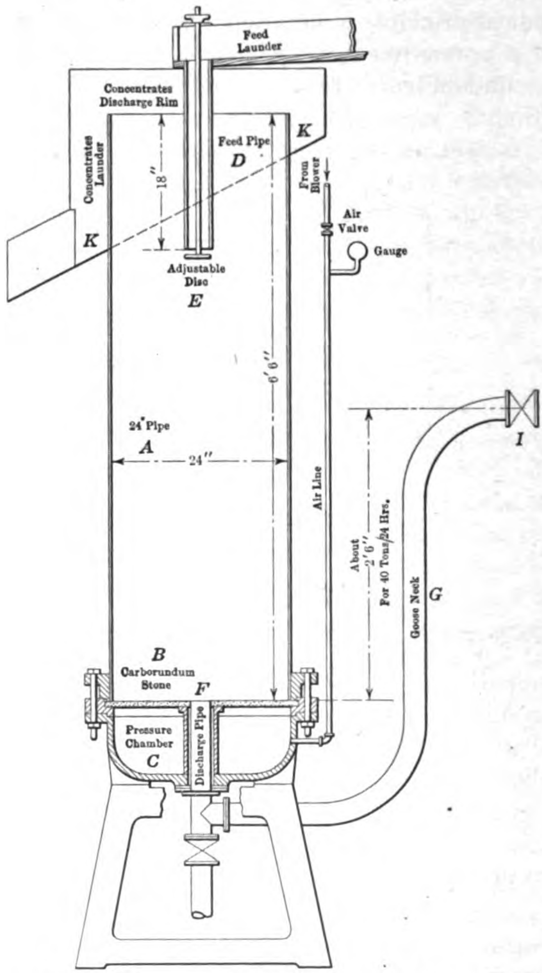



FIG. 5.—FLINN-TOWNE FLOTATION MACHINE.

shape of cylindrical tanks, the bottoms of which are formed by the poro medium. The feed enters near the top in the center of the cylinde while the tailings leave the machine through a center hole in the poro medium. In the Flinn-Towne installation, only one roughing machi was provided, while additional cells served for the purpose of cleani

both rougher tailings and rougher concentrates. The tailings produced by the concentrate cleaner cell were returned to the head of the roughing cell. In place of the canvas blanket of the Callow machine, Messrs. Flinn and Towne in the demonstration test at Inspiration used carborundum stones as the porous medium through which the air is injected into the pulp.

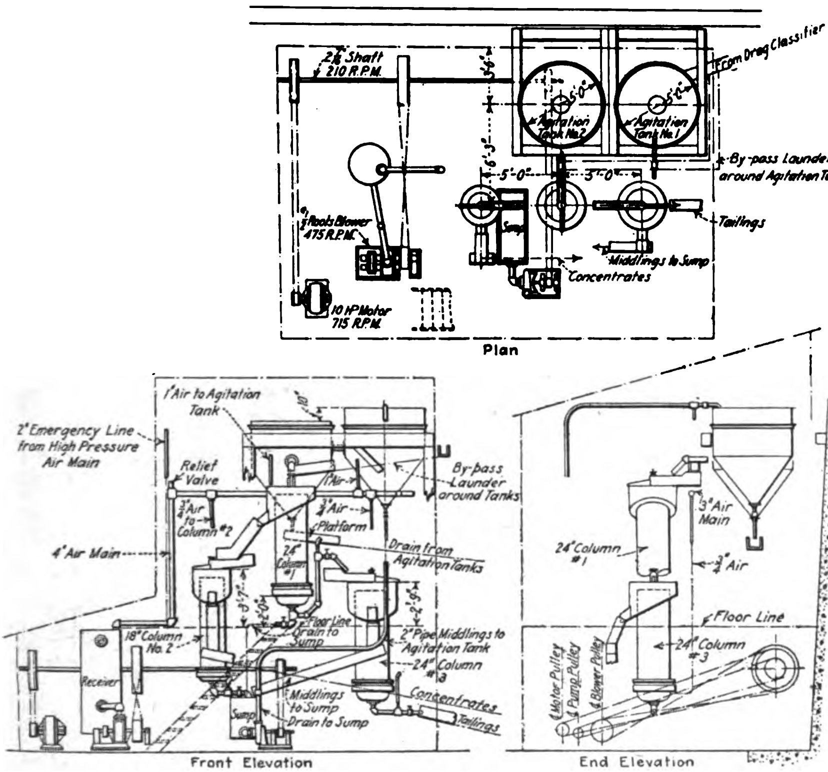



FIG. 6.—OUTLINE OF FLINN-TOWNE FLOTATION INSTALLATION.

### *Cole-Bergman Installation*

The Flinn-Towne machine was withdrawn from the contest after a competitive test between this machine and others had run for several months, although the results obtained looked very encouraging. Its place was taken by a flotation machine designed by Messrs. Cole and Bergman.

Their machine is illustrated in Fig. 7. In principle the cells are similar to those of the Flinn-Towne machine. Evidently, the designers tried to improve on this system by mechanically developing the idea underlying the machine, and by changing one point, which in their opinion formed a weak part of the Flinn-Towne machine. This is the

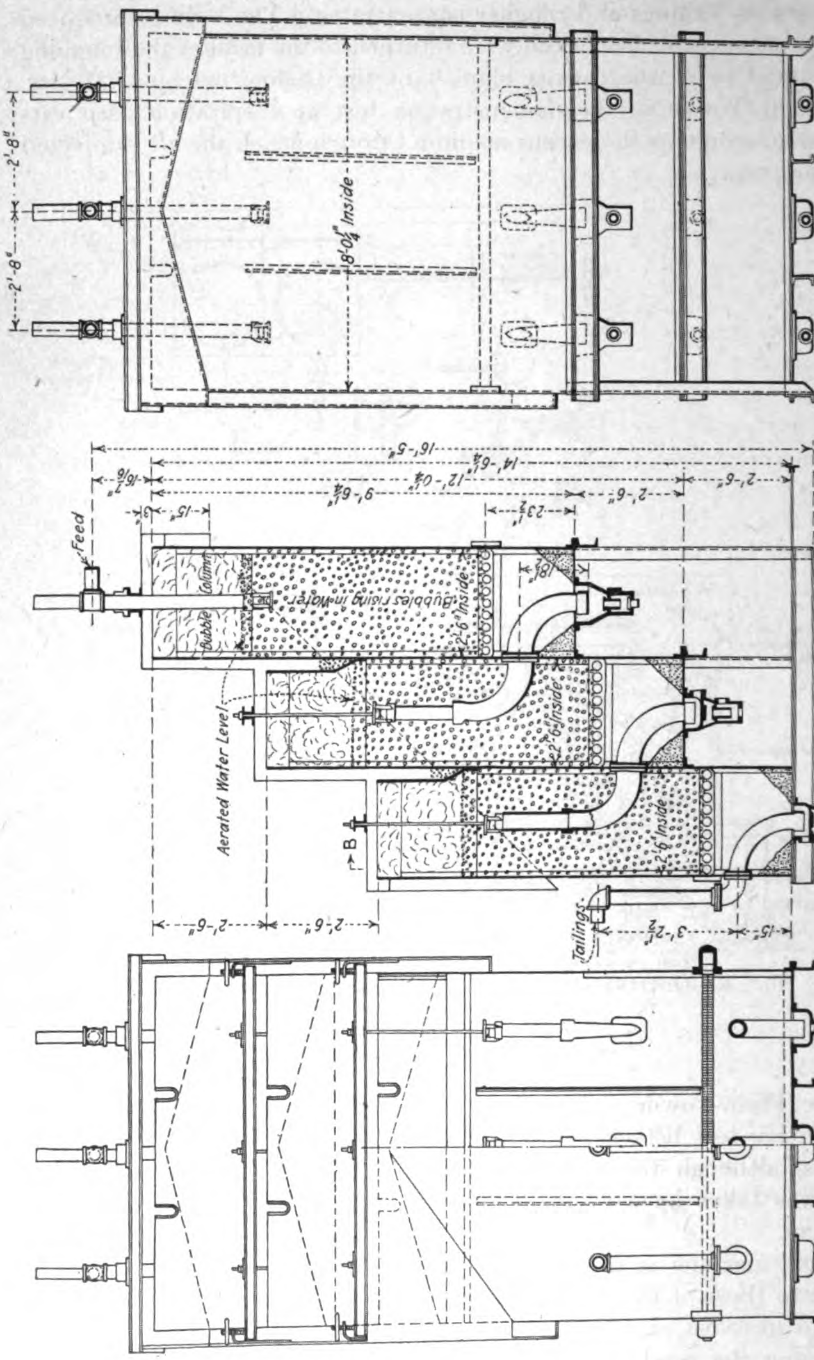



FIG. 7.—COLE-BERGMAN FROTHING CLASSIFIER.

ıction of the porous diaphragm, which, as explained below, was by a round carborundum disk. While carrying on the test of inn-Towne machine, it proved necessary occasionally to wash the ındum stones by injecting a water pipe from the top and even by ng the stones and cleaning them with water, acids, etc. Messrs. ıd Bergman ascribed this deficiency to the fact that on account of rizontal position of the porous medium, sand had a tendency to hereon and to impede or prevent the passage of air through the They substituted, therefore, a system of perforated tubes covered suitable fabric, such as canvas or flannel. This led to the construc- a porous medium in the form of a grate, as shown in the illustra- Their idea seems to be a very happy one, as it makes it possible ly the flotation process to ore pulps containing comparatively particles of sand; for instance, it seems possible to treat with this ıe ore mixtures that contain sand particles as coarse as 10-mesh and s even coarser. Using this machine, the millman is now in a posi- treat slime by flotation without the necessity of removing it from ure with sand. The Cole-Bergman machine has given good results tests, as have the rest of the machines utilizing the pneumatic prin- It was operated in conjunction with a smaller machine of the same ıstalled for the purpose of cleaning the concentrates produced on ger machine.

### *Methods Followed in Competitive Tests*

he interest of a fair contest between the flotation machines of the ıt types, each machine was provided, as nearly as possible, with ıe character of pulp as the rest. In the beginning this was ac- shed by providing each machine with one or more pebble mills h feed was sent in such quantities that a mill product of practic- e same fineness resulted in each case. When the results of the itive tests began to approach each other very closely, the equali- of the pulp furnished to the different machines was still improved y mixing the ground product discharged from all the mills, and it to a divider which permitted the division into as many parts e were competing machines, and in any proportion desired. As been decided in former experiments not to install tables ahead lotation process, no preliminary table treatment was used during npetitive tests, but each flotation system was furnished with a ables for the retreatment of the flotation tailings. It developed is was essential in trying to arrive at a fair valuation of the ma- as some of the machines produced tailings from which additional l could be extracted by the table treatment more easily than from lings of other flotation machines, the cause evidently being that

flotation machines of one type have a tendency to save more of the fin particles, while flotation machines of another type permit of a bett recovery of coarser grains. Those flotation machines that make a go recovery on the sand, but leave a larger percentage of mineral in t slime, will not benefit from a subsequent table treatment as much as tho of the other kind.

### *Advantage of Pneumatic Flotation Machines*

In the course of the tests at Inspiration, the fact became appare that, by the application of the pneumatic principle, a higher recovery the slime is made possible than without the utilization of injected a although the Standard Minerals Separation machine also gives excelle results when the tonnage is reduced to a considerably lower figure th the machine is supposed to be able to treat economically.

Therefore, in cases where high power consumption is of little imp tance, the Standard Minerals Separation machine will fill the requiremen of a flotation machine remarkably well. When, however, the power co sumed has to be seriously considered, it seems advantageous in the lig of the Inspiration experiments to make use of pneumatic flotation for t reason pointed out above, *i.e.*, that poor work of flotation machines sands can be made up by a subsequent table treatment, while poor wo on slime cannot. Within certain limits, it is more important to ins on machines effecting a good slime recovery than on machines makin good sand recovery.

### *Pneumatic Machines of Minerals Separation Co.*

The Minerals Separation Co., realizing this, offered to put in th subaeration type of machine which added the advantages resulting fr the use of injected air to the advantages which their Standard machi seemed to have in saving coarser mineral. This resulted in the constr tion and testing successively of two of their subaeration type machin one in which the old spitzkastens still were retained, and another one which they were dispensed with. As seen from the illustration, Fig. provisions were made in the former machine to admit air to the p conduits carrying the pulp from the spitzkastens to the agitating co partments next in succession, in order to make this air available to l mineral to the top of the spitzkastens where it can be skimmed off. T agitating compartments are closed at the top by covers set on an incl in such a way that the air rises to the top in the agitating compartme and is conducted to the spitzkasten, being thereby utilized for the purp of carrying the mineral values.

This construction of the Minerals Separation Co. was, however, o

an intermediate step toward the design of the second-named machine, a machine of greater simplicity, which is similar in principle to the one

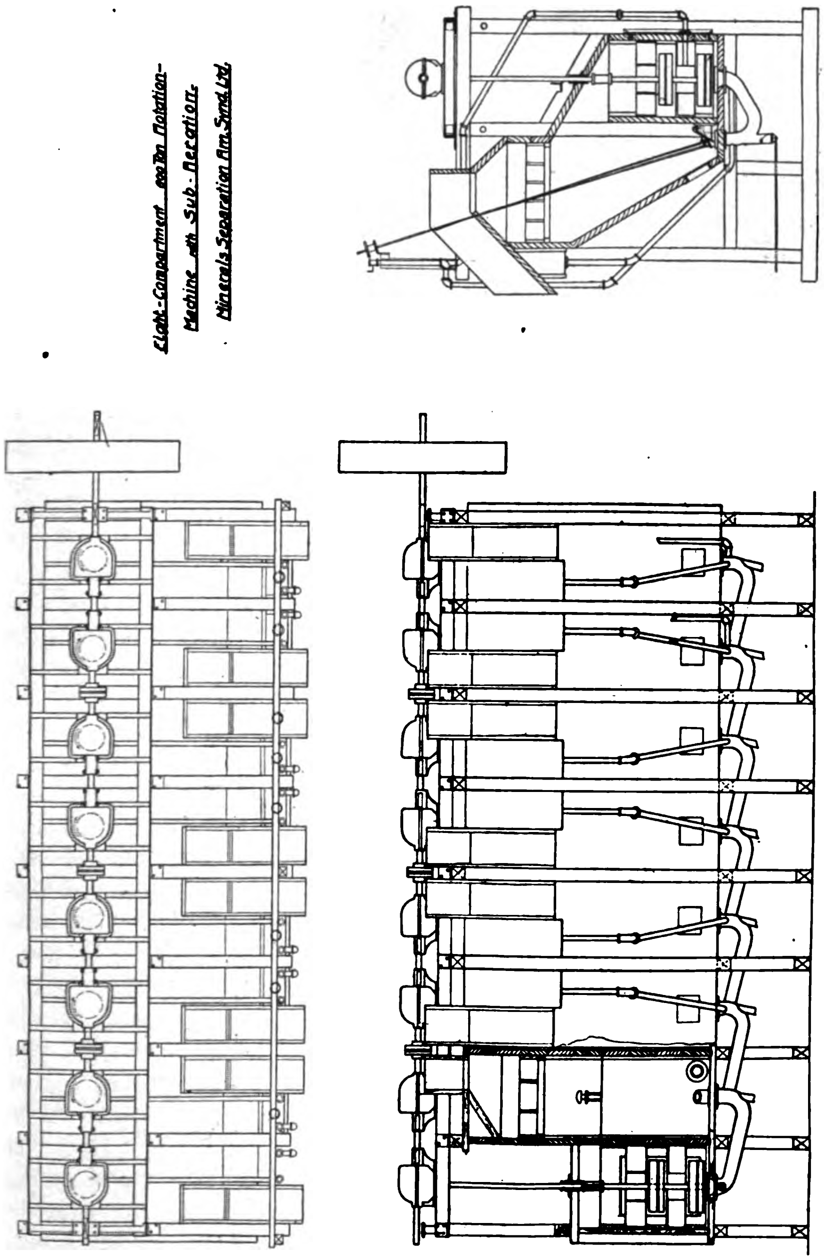



FIG. 8.—MINERALS SEPARATION SUB-AERATION TYPE OF FLOTATION MACHINE.

illustrated in Fig. 9. The machine consists of a long rectangular tank without any partitions, in which a number of agitators revolve, while air is

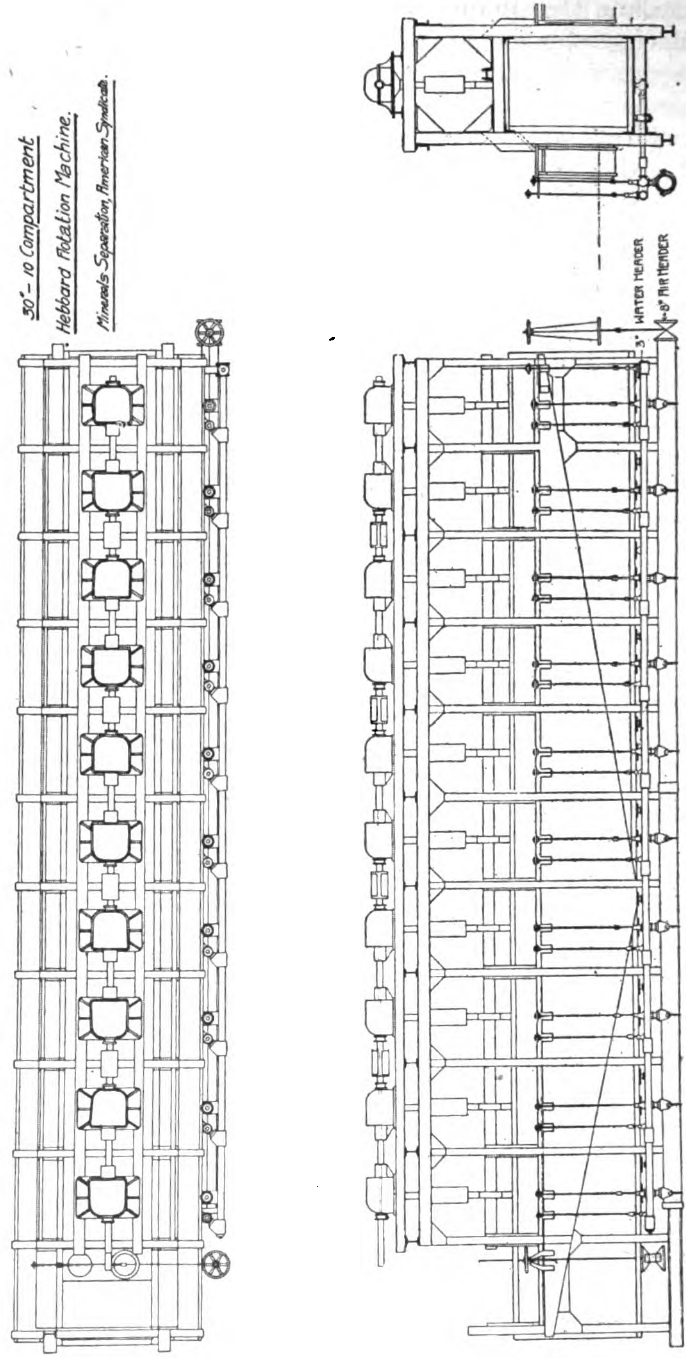



FIG. 9.—HEBBARD TYPE OF MINERALS SEPARATION MACHINE.

d underneath each impeller. We call the machine the "Hebbard Minerals Separation Machine.

rder to limit the agitation to the lower part of the machine and to an area of comparative quietness in the upper part, a system of is arranged above the agitators. As a consequence, the froth is chance to separate out in the upper portion of the rectangular nd flows off on both sides lengthwise into launders provided for pose.

s machine marks an important progress in the development of nerals Separation flotation systems toward increased recovery of at least as far as the Inspiration ores are concerned. In our tests ted a higher recovery of the fine mineral particles than the old e had accomplished. The machine also has the advantage of simplicity and relatively low power consumption. The only ck that we have found in the operation of the machine, extending number of months, is that the agitating shafts, which are suspended ove, have a tendency to bend and, if not straightened, soon cause eller blades to strike against the baffles, breaking one or the other.[5]

### *Float Skimming Device*

ile the competitive tests of the different flotation machines were onducted, it was observed that large quantities of mineral froth tly appeared in the tailings launders. The idea suggested itself ve the froth, thus increasing the recovery obtained in the flotation es and concentrating tables by an additional small amount. After xperimenting, a way was found in which this could be easily ac- hed. The tailings launder was widened and deepened for some e. The tailings stream was forced to pass underneath a baffle on g the widened space, and was also forced to travel under another oard at the end of the space, thus being made to rise through the ed area formed by the tailings end baffle and the back-wall of the ving device, which is illustrated in Fig. 10.

trary to what might be expected, tailings sand of the fineness of tion tailings will not lodge in the widened-out space unless the is greater than a certain critical distance, nor will it choke the channel in the tailings end of the machine. When conditions led the construction of the float-saving device in greater length, it nd that the accumulation of sand, which has a tendency to form niddle of the machine, can be avoided by arranging an additional ot reaching quite to the bottom in the center of the machine. If

s trouble is alleviated in the Hebbard machines turned out more recently by erals Separation Co. by the provision of suitable bearings for the impeller the bottoms of the machines.

the machine is built longer yet, more than one intermediate baffle is necessary. The explanation for this behavior of the pulp, which at first glance might seem paradoxical, is that, on account of reduced passageway between the lower part of the baffles and the bottom of the machine, the pulp is forced to travel through at speed so high that it will not permit the settling out of sand. As a consequence, a hydrostatic head will establish itself between front and back of such a baffle and has to be taken care of in the design. It is found, however, in actual practice, that the

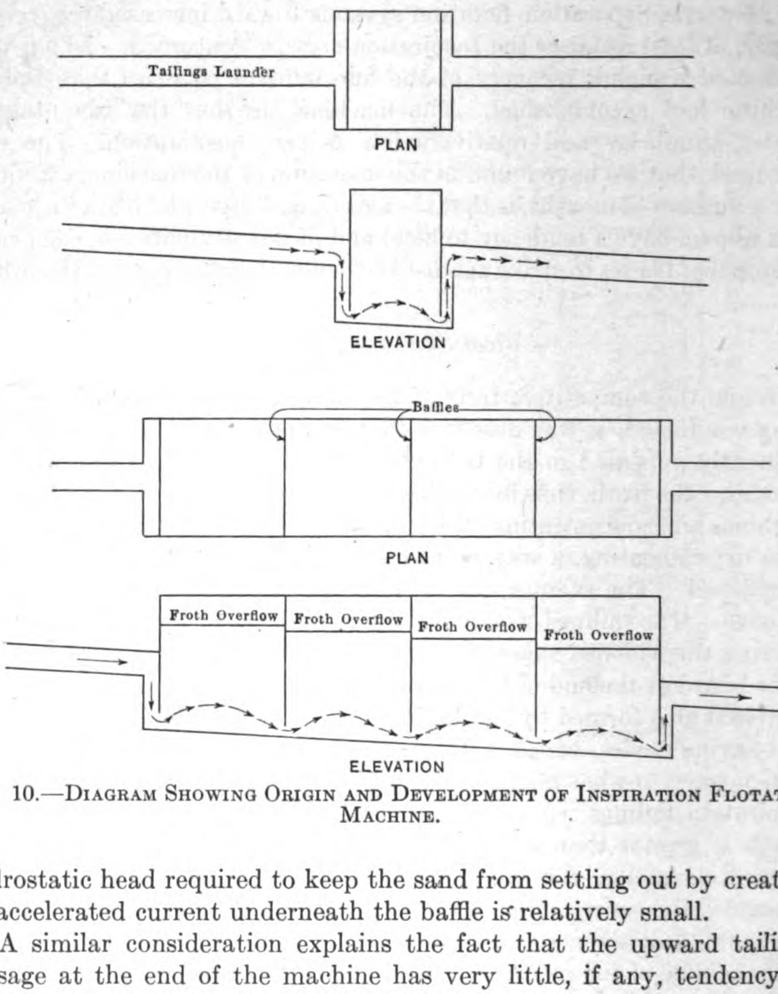



FIG. 10.—DIAGRAM SHOWING ORIGIN AND DEVELOPMENT OF INSPIRATION FLOTATION MACHINE.

hydrostatic head required to keep the sand from settling out by creating an accelerated current underneath the baffle is relatively small.

A similar consideration explains the fact that the upward tailings passage at the end of the machine has very little, if any, tendency to choke; to take care, however, of possible choke-ups that might be caused by the discharge of coarse rocks, pieces of wood, etc., into the pulp, it was found advisable to arrange some air jets in the bottom of the upward tailings passage to assist in clearing it whenever necessary.

*Inspiration Flotation Machine*

s device was in actual operation for several months at our test d made itself pay well by adding about 1 per cent. to the recovery

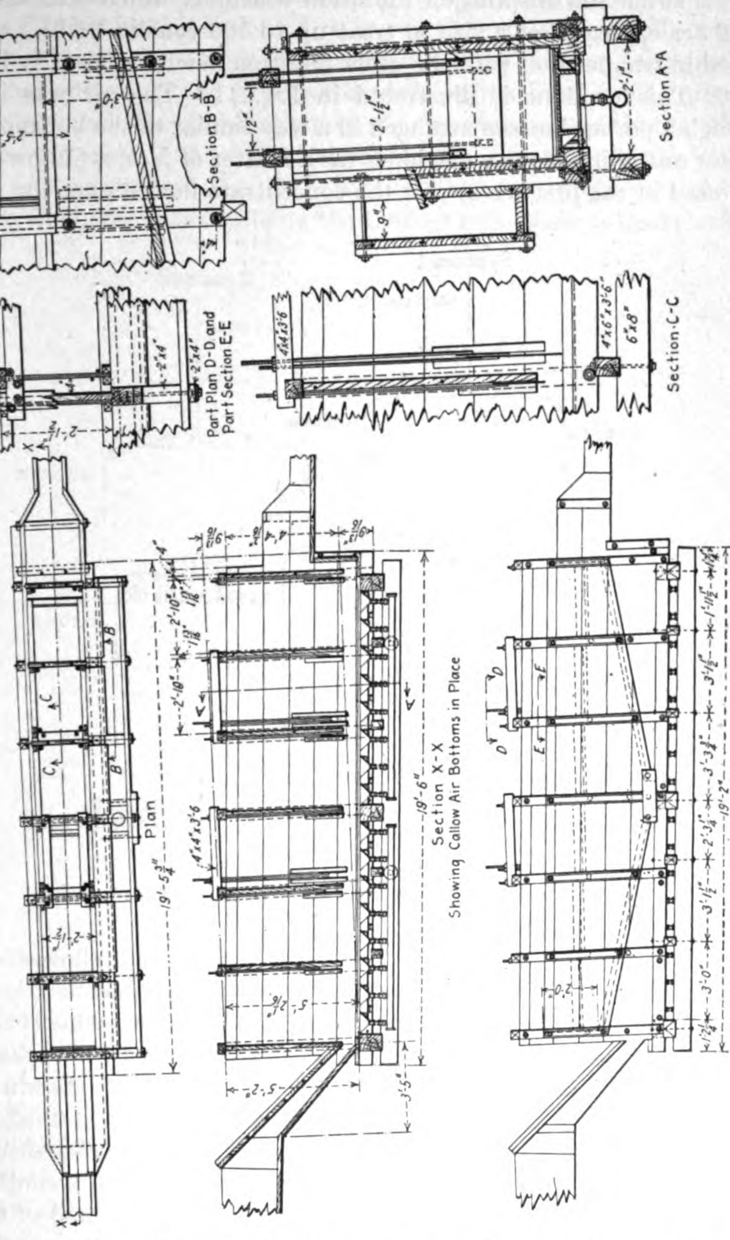



FIG. 11.—EXPERIMENTAL INSPIRATION FLOTATION MACHINE.

y obtained. The question then presented itself as to whether this which was successful in saving mineral escaping from the flotation

machines could not be transformed into a flotation machine itself b arranging for the injection of finely distributed air into the pulp passi through the machine. The development of the idea led to what we c our Experimental Inspiration Flotation Machine, which was built on small scale, that is of a size to treat up to 100 tons in 24 hr., and w placed in competition with the other flotation machines already in exis ence. This machine is illustrated in Fig. 11. The air was inject through a porous bottom arranged in a way similar to the bottoms of t Callow and Flinn-Towne machines (as a matter of fact, a Callow botto was used in the first tests) and the concentrate flowed over the edge

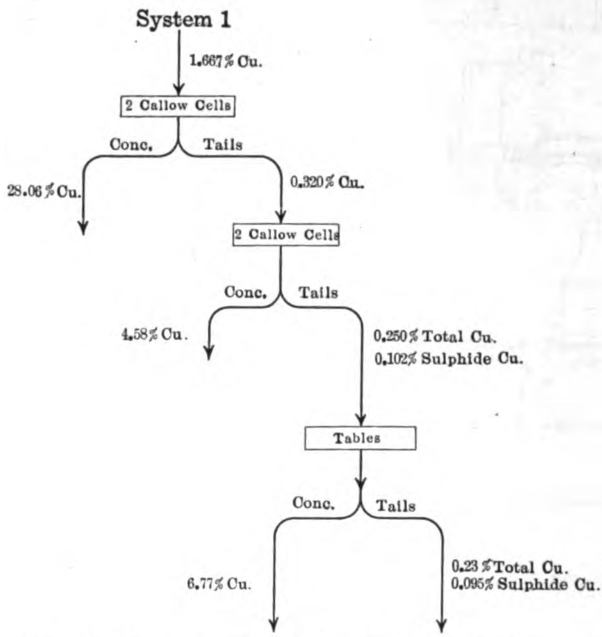
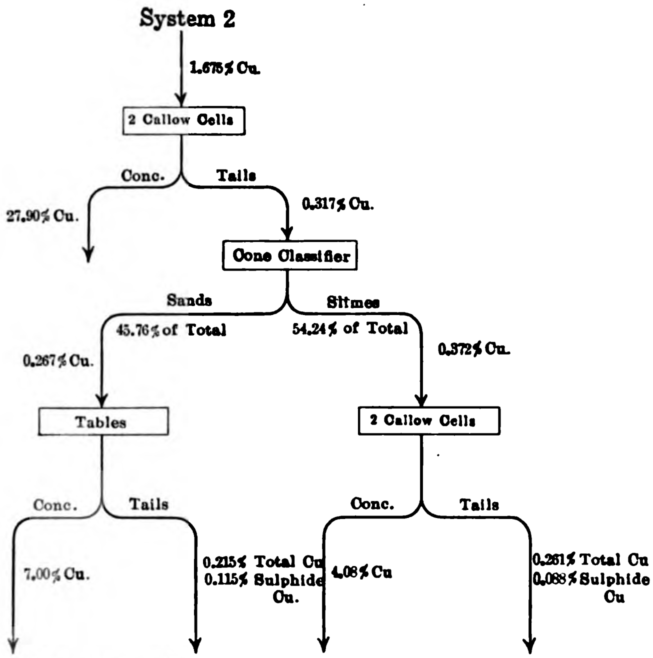



FIG. 12.—TESTS WITH CALLOW CELLS IN SERIES.

one side of the flotation tank. The mode of operation followed was al similar to the one developed for the other air machines. A low-gra concentrate was produced and retreated on a similar machine of the sa construction, called the cleaner machine, the tailings from which we returned to the head of the roughing flotation machine. As will be se from the drawing, the construction of this machine is extremely simp and comes pretty near to an ideal of Mr. Mills, our general manager, w prophesied that the flotation machine of the future would be nothing b a launder with provisions for injection of air. The method of effecti the concentration in two stages was also followed when the new types the Minerals Separation machines were put into commission.

*Arrangement of Callow Cells in Series*

principle, the Inspiration machine shows one difference from the machine; viz., that instead of splitting the pulp between a great of machines, it is forced to travel through a series of compart- n succession. This is the same policy that had also been followed onstruction of the Minerals Separation machines. The question me up, whether it might not be of advantage to arrange the Callow so in series. To test out the idea, the flow sheet of the Callow vas changed (see Fig. 12). The primary feed was sent to two of s, the other cells receiving the tailings from these primary cells.




13.—Tests with Callow Cells in Series with Classifiers Between.

s arrangement showed at least no disadvantage over the multiple ent; it seemed superior to the other system in the one respect that ed at a glance whether the cells were operated well, because in se most of the concentrate would be produced on the primary ile the secondary cells would produce a rather light-colored froth ald add only a small fraction to the recovery made on the primary Both primary and secondary concentrates were treated jointly in er cell, while the tailings from the secondary cells were conveyed mber of concentrating tables for retreatment.

ariation of the latter flow sheet was also tested (compare Fig. 13).

Between the primary and secondary cells, a cone classifier was interpos
for the purpose of making a separation between the sand and slime. T
slime was sent to the secondary flotation machines, while the sand w
passed to tables. This latter flow sheet was finally decided upon for t
Callow installation in the large concentrator. Tests made with the obj
of establishing which of the two is superior, resulted in a tie. Some of t
figures on this point are given in Table 2.

TABLE 2.—*Retreatment of Tailings from Two Callow Cells. Comparison
Results Obtained from Two Different Methods as Shown in Figs. 12 and
Screen Analysis of Feed and Tails of Secondary Callow Cells*

| Mesh | System 1 | | | | | | | | | System 2 | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Feed | | | | Tails | | | | | Feed | | | | Tails | | | |
| | Per Cent. Weight Cum. | Per Cent. Weight Indiv. | Per Cent. Cu | Cu Contents | Per Cent. Weight Cum. | Per Cent. Weight Indiv. | Per Cent. Cu | Cu Contents | Per Cent. Sol. Cu | Per Cent. Weight Cum. | Per Cent. Weight Indiv. | Per Cent. Cu | Cu Contents | Per Cent. Weight Cum. | Per Cent. Weight Indiv. | Per Cent. Cu | C t |
| 65 | 5.0 | 5.0 | 0.29 | 0.014 | 5.0 | 5.0 | 0.21 | 0.011 | 0.05 | | | | | | | | |
| 100 | 19.0 | 14.0 | 0.25 | 0.035 | 19.2 | 14.2 | 0.15 | 0.021 | 0.06 | 3.8 | 3.8 | 0.20 | 0.007 | 4.0 | 4.0 | 0.17 | 0 |
| 150 | 32.2 | 13.2 | 0.29 | 0.039 | 32.4 | 13.2 | 0.16 | 0.021 | 0.10 | 10.6 | 6.8 | 0.25 | 0.017 | 10.8 | 6.8 | 0.14 | 0 |
| 200 | 42.4 | 10.2 | 0.33 | 0.034 | 42.8 | 10.4 | 0.21 | 0.022 | 0.11 | 18.6 | 8.0 | 0.30 | 0.024 | 19.0 | 8.2 | 0.25 | 0 |
| | 100.0 | 57.6 | 0.35 | 0.201 | 100.0 | 57.2 | 0.27 | 0.155 | 0.20 | 100.0 | 81.4 | 0.40 | 0.326 | 100.0 | 81.0 | 0.30 | 0 |
| Assay calc. | | | 0.32 | 0.323 | ..... | .... | 0.23 | 0.230 | 0.15 | ..... | .... | 0.374 | ..... | ..... | .... | 0.28 | 0 |
| Assay direct. | | | 0.32 | ..... | ..... | .... | 0.23 | ..... | 0.15 | ..... | .... | 0.370 | ..... | ..... | .... | 0.26 | .. |

NOTE.—The concentrates from the flotation cells were in both cases retreate
half-size Callow cells, the retreatment tailings being returned to the head of the
mary cells.

| | Ton per Primary Cell |
|---|---|
| Tonnage rate in system 1 | 49.3 |
| Tonnage rate in system 2 | 47.5 |

CONCLUSIONS.—No advantage is apparent in favor of either system. System
be installed should be decided from mechanical points.

### *Variations of Flow Sheet*

In the way of other flow sheet variations, etc., only one thing was tr
to any extent; that is, whether it is wise to send the cleaner tailings ba
to the head of the rougher machine for retreatment. In the limited ti
at our disposal, it proved impossible to substitute something better, l
the chances are that the same holds true of flotation that is true for wa
concentration; viz., that middlings are snnt back for retreatment mai
because the designer does not know anything better to do with the
In principle it seems wrong to follow this practice, because, when hea
copper losses occur in concentrating machines which express themsel
in high tailings, there is always a doubt as to whether this is due to p

ery made in the primary treatment or to heavy losses incurred by
lings sent back for retreatment.

### *Coal Tar as Flotation Agent*

was to be expected, when the flotation process was installed in our
lant, that there would be ups and downs in the recovery because
rocess was rather new, especially in its application to chalcocite ores.
ie month of June, 1914, the recovery obtained in the test mill showed
den drop, and the serious problem confronted us of establishing the
and finding a remedy. Some of the flotation experts suggested
t might be due to the fact that a new shipment of cresylic acid which
een received and used at that time might not fill the specifications
ng 98 per cent. pure. We did not feel competent to say whether
npurities actually amounted to more than 2 per cent. We were,
ver, inclined to think that perhaps cresylic acid, which is, as is well
n, one of the products resulting from fractional distillation of coal
night not represent the fraction most suitable for the flotation of
es. Having no coal tar available in the neighborhood, we proceeded
ke some by distilling a sample of ordinary New Mexico soft coal
eparating the tar thus formed into the fractions distilling off at
ent temperatures.

ır facilities for testing oils were limited at that time. The Minerals
ation Co.'s representative did not believe in small-scale tests, and
is reason did not recommend experiments with small testing machines.
rtheless, it seemed desirable to have something with which to carry
mall-scale laboratory experiments. Dr. Ricketts, who was aware
troubles and realized the importance of such tests, was kind enough
d us a little electrically operated emulsifying machine, which served
ably for qualitative tests. We also built a testing machine based
e principle of the Standard Minerals Separation machine, with the
ence, however, that instead of sending the pulp from one agitating
artment to a spitzkasten and then into another agitating compart-
and spitzkasten, we made the pulp return from the first spitzkasten
e original agitator, forcing it to revolve in a closed circuit. Our
atory machine is illustrated in Fig. 14. Lately, a machine based on
me principle has been put on the market and is sold by the Denver
Clay Co. Thus, we had a chance to try the different fractions of
ome-made coal tar.

he chemist who conducted these tests (Mueller) hit on the idea that
ght be well, in addition to trying the different fractions, also to test
oal tar as a whole. The results were very surprising, since they
ed that by the addition of crude coal tar we could effect a greater
ery than we were able to obtain by the use of highly refined cresylic

acid. From this point dates our experience that it is better to use coal tar than soluble flotation agents like cresylic acid to save coarse mineral. Cresylic acid is an extraordinarily good agent for producing froth, but

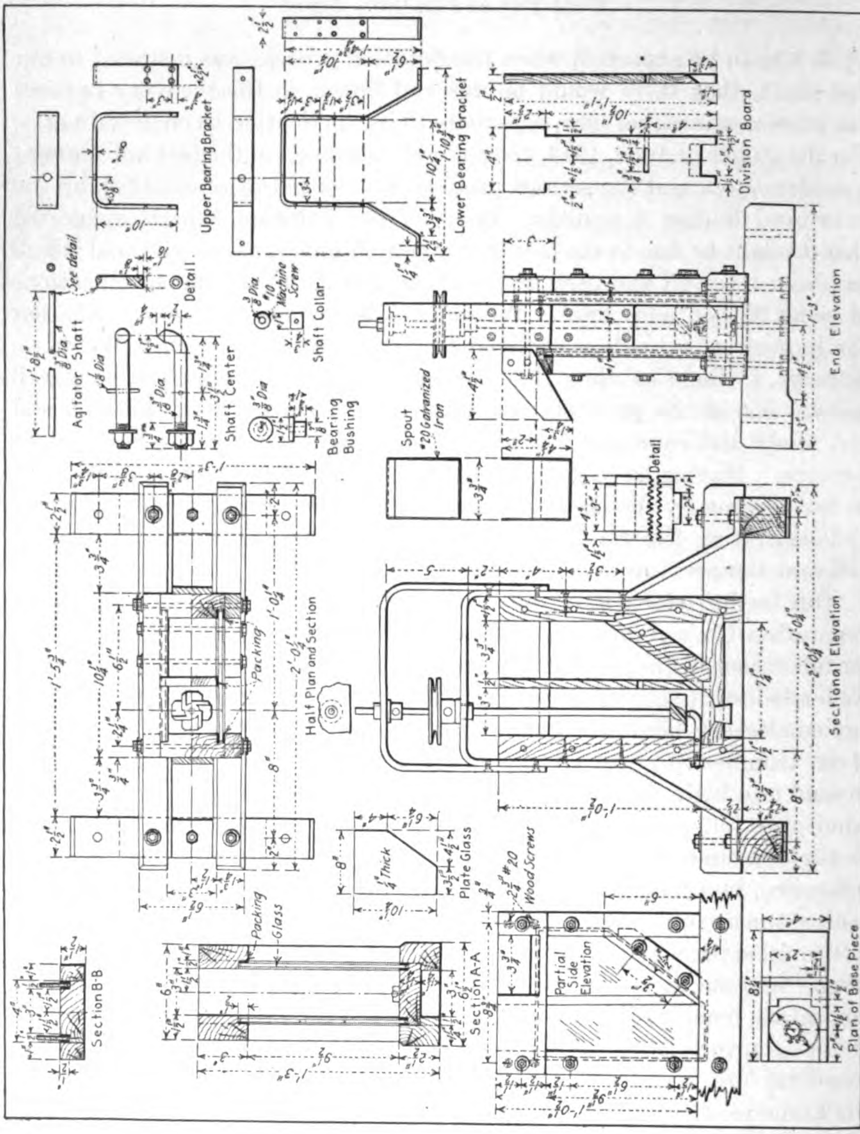



FIG. 14.—LABORATORY FLOTATION MACHINE.

the froth which it produces does not seem to have as much carrying power for coarse mineral as that produced by coal tar. Not all coal tars are equally good for this purpose. Tests in laboratory machines easily show the difference between coal tars of different origin.

The system that we used to carry out such tests has been described, although without authorization, by William A. Mueller, one of my former assistants who took part in these laboratory tests.[6]

It is difficult to utilize coal tar in plants using flotation supplementary to gravity concentration, on account of the fact that it is not easy to effect a good amalgamation of tar with the pulp in agitating tanks, and even in mechanical flotation machines. The use of coal tar lends itself very well indeed to the system of feeding tar into the grinding machines, a system that, as mentioned above, had been worked out in our small test mill and patented by Mr. Chapman.

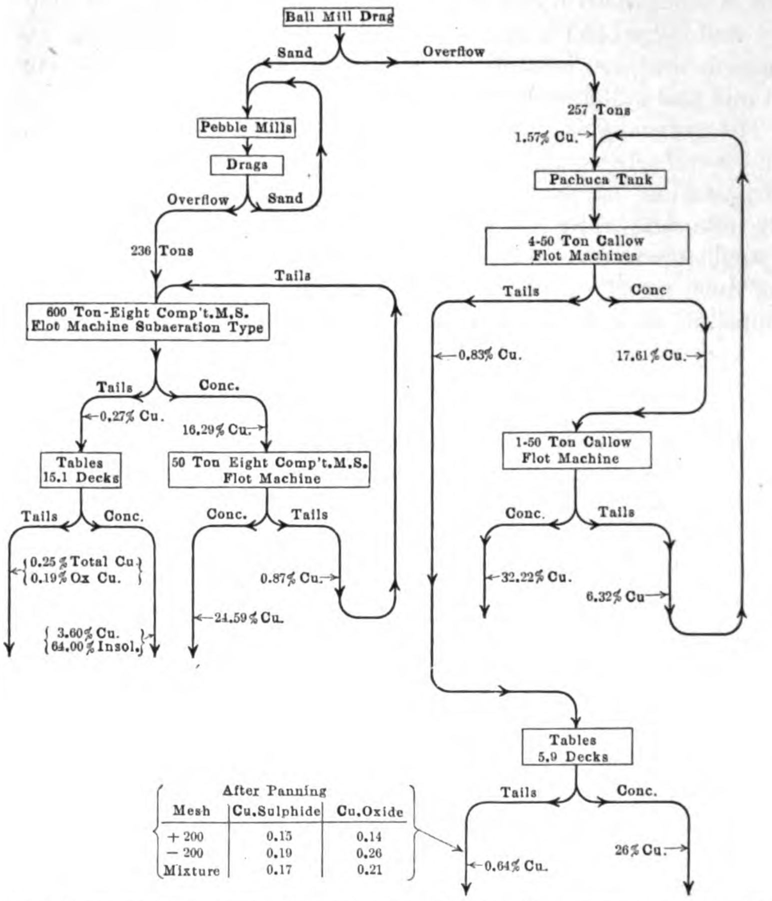
The company is indebted to Mr. Callow for proving the merits of coal-tar creosote as a flotation agent by using it in his demonstration plant at Inspiration. After we had established the value of coal tar by laboratory tests, and while efforts were being made to obtain it commercially, he applied creosote successfully. We have continued to use it for a long time, mostly in combination with coal tar, and have only recently dropped it, as we find crude coal tar cheaper and better.

### *Experience with Primary Slime*

After the first difficulty that we encountered in our large-scale tests had been solved by the introduction of coal tar into the flotation-oil mixture so far used, things went along fairly well for some time until a new difficulty was encountered. It happened that once in a while an abundant froth was produced on the flotation machines, but this froth seemed to have very little carrying power for the mineral contents of the ore and held mainly finely divided gangue. It was observed that this phenomenon occurred with special severity whenever the ores shipped to the test mill contained a large amount of fines and a small amount of coarse rock. It was attributed, therefore, to the presence of what may be called primary slime, *i.e.*, slime not formed by the crushing of ore in the mill, but originating from the mine. That the falling down of the flotation machines was caused by variation of the ores was proven by the fact that samples of the refractory ores when treated in the laboratory testing machine, gave as unsatisfactory results as the corresponding ore did in the big flotation machines. To demonstrate that the presence of the original slime caused the trouble, samples of the mill feed were separated by screening them on a 200-mesh screen. The oversize, when reduced to the proper fineness for flotation treatment, did not offer the least trouble and yielded a high-grade concentrate in the laboratory machines, while the undersize proved extremely refractory (refer to Table 3). The same experiment was repeated on a large scale,

[6] William A. Mueller: Use of Coal Tar in Flotation, *Engineering and Mining Journal*, vol. 100, No. 15, p. 591 (Oct. 9, 1915).

the products from a Marcy ball mill crushed to about 6-mesh bei treated on a drag classifier. The oversize, after being reduced in pebl mills to the necessary fineness for flotation treatment, was sent to o group of flotation machines, while the overflow from the drag classif was sent to another group. Fig. 15 illustrates the results of this expe ment. While the reground oversize yielded remarkable lean tailin



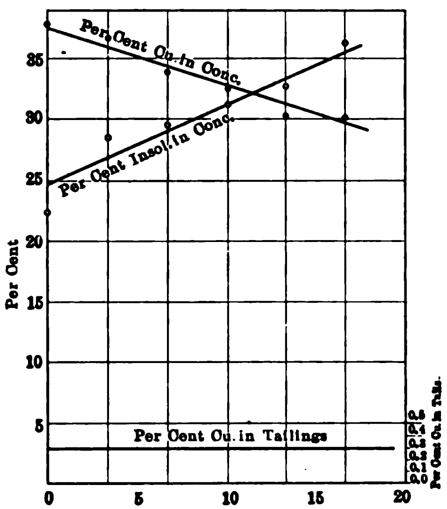

| After Panning | | |
|---|---|---|
| Mesh | Cu.Sulphide | Cu.Oxide |
| +200 | 0.15 | 0.14 |
| −200 | 0.19 | 0.26 |
| Mixture | 0.17 | 0.21 |

**Fig. 15.**—Tests to Determine Effect of Separate Treatment of Slime a Reground Sand.

the product containing the primary slime could not be treated adva tageously by flotation. We were almost on the point of concluding th in order to get the best results, a separation of the primary slime shou be made in our large mill and flotation should not be entirely relied up for the treatment of the slime. Before finally deciding on this poi however, some additional laboratory experiments were made to stu the influence of the primary slime on flotation. These tests establish

3.—*Screen Analysis of Ore Drawn from Mill Bins and Flotation Tests of Different Screen Sizes*

| Weights, Per Cent. Cumul. | Weights, Per Cent. Indiv. | Per Cent. Total Copper | Copper Contents | Per Cent. Oxidised Copper | Contents Ox. Cu | Flotation Results: Tails | Flotation Results: Midds.+Conc. | Recovery Per Cent. |
|---|---|---|---|---|---|---|---|---|
| 30.8 | 30.8 | 1.05 | 0.324 | 0.13 | 0.40 | 0.22 | 12.8 | 80.5 |
| 47.9 | 17.1 | 1.34 | 0.329 | 0.11 | 0.19 | 0.22 | 17.4 | 84.7 |
| 70.9 | 23.0 | 1.44 | 0.332 | 0.12 | 0.28 | 0.17 | 17.1 | 89.0 |
| 75.2 | 4.3 | 1.42 | 0.061 | 0.12 | 0.05 | 0.17 | 16.3 | 88.8 |
| 80.3 | 5.1 | 1.77 | 0.068 | 0.13 | 0.07 | 0.26 | 16.8 | 86.7 |
| 84.4 | 4.1 | 2.14 | 0.080 | 0.14 | 0.06 | 0.61 | 16.3 | 74.3 |
| 87.5 | 3.1 | 2.62 | 0.081 | 0.19 | 0.06 | 0.72 | 11.3 | 77.5 |
| 89.9 | 2.4 | 3.29 | 0.079 | 0.24 | 0.06 | 0.63 | 22.5 | 83.1 |
| 91.9 | 2.0 | 3.16 | 0.063 | 0.27 | 0.05 | 0.60 | 20.1 | 83.5 |
| 100.0 | 8.1 | 1.53 | 0.124 | 0.43 | 0.35 | 1.57 | 1.48 | None |
| ..... | 100.0 | 1.44 | 1.441 | 0.16 | 0.157 | | | |




IG. 16.—INFLUENCE ON FLOTATION OF MIXING PRIMARY SLIME AND REGROUND SAND.

+ 1-in. and + ½-in sizes resulting from the screen analysis were ground to mixed with varying amounts of − 200 material and the mixtures ground with l amounts of oil in the laboratory ball mills. After this preliminary treat- e samples were treated in the laboratory flotation machines.

t quite well, which is, that when refractory slime is mixed with a nt quantity of coarse ore ground to the necessary fineness, in other when the percentage of primary slime in the flotation feed is kept

low, the slime loses its refractory character. I do not mean to sa by reducing the quantity of primary slime, the deleterious influ reduced in proportion so that it cannot be detected as easily, but co from our tests that when the flotation feed contains a sufficient perc of comminuted coarse rock, the primary slime contained in it treated more advantageously than it could by itself. Some tests b on this point are illustrated in Fig. 16. They indicate that as h amount as 20 per cent. primary slime may be mixed with ground ore without causing increased tailing losses.

The concentrates, according to this set of tests, seem to have ency, however, to carry more insoluble matter with an increasi centage of slime.

*Influence of Iron on Flotation*

While these tests were in progress, we made another acciden covery which proved very helpful to us. In our tests on the mo nomical way of reducing the ore to the fineness necessary for fl we had, among other machines, a ball mill in competition with mills. In the ball mill, steel balls performed the duty that in mills was done by flint pebbles.

For a while, the ball-mill discharge was treated on one fl machine, while the pebble-mill discharge was treated on a group of While this flow sheet was being followed, we thought we noted flotation machine treating the ball-mill product showed the influe the primary slime to a lesser extent than the flotation machine t the pebble-mill product. In a discussion with Dr. Ricketts an Mills, the question was raised as to whether the iron introduced mill pulp by the attrition of the balls might not have something with the fact. The question was accordingly made the subject o laboratory experiments. The results of a series of such experime represented in Table 4, and proved conclusively that the iron beneficial influence on flotation in counteracting the harmful ef the primary slime. This discovery was one of the inducements stalling ball mills in the big concentrator, while originally pebbl had been considered for this purpose.

We have not yet reached a point where we can safely give the for the action of the iron introduced into the flotation pulp. It i from the experiments referred to, that the same results, as by gr with balls, could be obtained by introducing the iron in finely d form, say in the form of filings, into a pebble-mill pulp. We su for a while, and I am not yet certain that this supposition is inc that the metallic iron might react on the impurities contained in s in the mill water and introduced therein with the primary slime find, as a matter of fact, that our ores contain very little in the na

4.—*Effects of Iron and Other Solids on the Flotation of Refractory Copper Ores*

| Grams Ore | Per Cent. Copper | Grams Copper | Concentrates | | | Recovery Per Cent. | Remarks |
|---|---|---|---|---|---|---|---|
| | | | Grams | Per Cent. Cu | Grams Cu | | |
| 750 | 2.01 | 15.07 | 45 | 23.66 | 10.65 | 70.7 | Added 10 g. iron filings. |
| 750 | 2.01 | 15.07 | 47 | 29.52 | 11.52 | 76.4 | Added 10 g. iron filings. |
| 750 | 2.01 | 15.07 | 43 | 27.10 | 11.65 | 77.5 | Added 2 g. iron filings. |
| 750 | 2.01 | 15.07 | 47 | 26.90 | 12.64 | 84.0 | Added 2 g. iron filings. |
| 750 | 2.01 | 15.07 | 51 | 23.84 | 12.16 | 80.7 | Added 10 g. iron filings. |
| 750 | 2.01 | 15.07 | 48 | 25.60 | 12.29 | 81.5 | Added 10 g. iron filings. |
| 750 | 2.01 | 15.07 | 27 | 8.80 | 2.38 | 15.8 | Blank with no solids added. |
| 750 | 2.01 | 15.07 | 28 | 8.34 | 2.34 | 15.5 | Blank with no solids added. |
| 750 | 2.01 | 15.07 | 62 | 20.14 | 12.49 | 83.0 | Added 10 g. miscellaneous iron filings from shops. |
| 750 | 2.01 | 15.07 | 63 | 20.10 | 12.66 | 84.0 | Added 10 g. miscellaneous iron filings from shops. |
| 750 | 2.01 | 15.07 | 60 | 18.82 | 11.29 | 75.0 | Same as F49 and F50 by different observer. |
| 750 | 2.01 | 15.07 | 62 | 19.54 | 12.11 | 80.5 | Same as F49 and F50 by different observer. |
| 750 | 2.01 | 15.07 | 29 | 6.16 | 1.79 | 11.9 | Blank with no solids added. |
| 750 | 2.01 | 15.07 | 30 | 7.96 | 2.39 | 15.9 | Blank with no solids added. |
| 750 | 2.01 | 15.07 | 59 | 21.08 | 12.44 | 82.7 | Added 10 g. iron filings. |
| 750 | 2.01 | 15.07 | 56 | 25.80 | 14.45 | 96.0 | Added 10 g. iron filings. |
| 750 | 2.01 | 15.07 | 65 | 19.52 | 12.69 | 81.9 | Ground in mill with steel balls instead of pebbles. |
| 750 | 1.10 | 12.75 | 37 | 28.42 | 10.51 | 81.7 | Blank on good flotation ore. |
| 750 | 1.10 | 12.75 | 34 | 28.46 | 9.85 | 78.3 | Blank on good flotation ore. |
| 750 | 1.10 | 12.75 | 76 | 5.66 | 4.30 | 33.4 | Identical conditions as L27 and L28 but added 10 g. zinc filings. |
| 750 | 1.10 | 12.75 | 83 | 5.06 | 4.20 | 32.9 | Identical conditions as L27 and L28 but added 10 g. zinc filings. |

salts, and that whatever they do contain is mainly confined to the y slime. For this reason, in laboratory tests we have tried edly to substitute pure water for the water contained in the mill In every case, we have noted some improvements in the results ed. We have also found that when we separate the water from the ory pulp, treat it with iron filings, and add it to the original pulp we get a certain improvement in the recovery, but we have not ble to get an improvement equally as good as that obtained by ntroduction of finely divided iron into the pulp. For this reason, e often thought that the effect of iron is physical rather than al in character. The iron exists in the pulp, at least partly, in tallic form, as can be proven by the use of the magnet. If neces- e effect of the iron could be increased by removing the iron con- in the tailings pulp by means of electromagnets and returning e mills or the flotation machines. Although we have consid- e possibility of applying this fact commercially, we have not yet .

ve been told by several people that they have tested the influence

of finely divided iron on their ores and have not obtained any i
ment whatever. This shows, evidently, that metallic iron is no
versal remedy for all flotation troubles, but as far as our prima
is concerned, our experience leaves no doubt about its usefulnes
believe the figures quoted above are positive enough to bear
statement. As a curiosity I have included in Table 4 some figur
ing the influence of the addition of zinc filings to ore that with
addition does not offer any difficulty in the flotation treatment.

### *Influence of Chemicals on Primary Slime*

We also found that the harmful influence of primary slime
counteracted by the introduction of certain chemicals; for insta

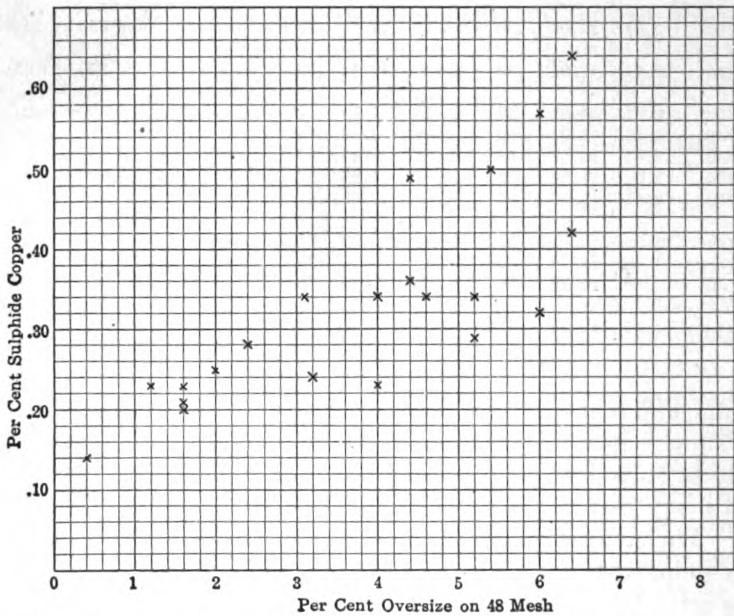



FIG. 17.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE F
COPPER SULPHIDE ASSAY OF CALLOW FLOTATION TAILING.

have used advantageously sodium hydroxide or potassium cyanid
use of sodium hydroxide was recommended to us by the Minerals
tion Co., which, I understand, applied it first in the plant of the C
Copper Co.

### *Fineness of Crushing Desired*

Tests to determine the most advantageous fineness for treat
the ore according to our flow sheet, *i.e.*, flotation machines foll

tables, naturally formed an essential part of our tests. Results of this character are plotted in Figs. 17 and 18 and seem to indicate that to get the best results grinding should be carried to such a point that not more than about 1 or 2 per cent. of the pulp will remain on a 48-mesh screen (Tyler type). The grinding at our large concentrator is at present carried closely to this point.

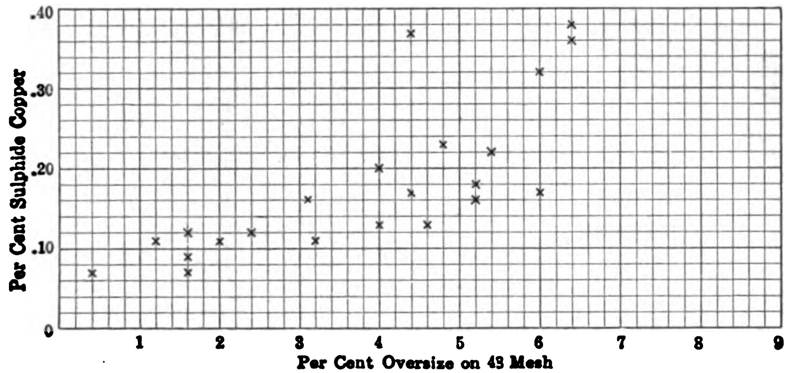



FIG. 18.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW TABLE TAILING.

## OPERATION OF LARGE CONCENTRATOR

### *Flow Sheet of Callow Sections*

The first four sections of the Commercial Concentrator of the Inspiration company were equipped with Callow flotation machines. The flow sheet followed in these sections is illustrated in Fig. 19. As will be seen from this illustration, the ore, after having been reduced to the desired fineness, and having been oiled in the Marcy ball mills, is sent to 12 (in other sections 8) Callow cells. The tailings from these primary cells are sent to a drag classifier designed by Mr. Burch, which effects a separation of sand and slime. The slime is retreated in 12 (in other sections 16) additional Callow cells, while the sand is sent to a hydraulic classifier and then to tables. The concentrates made both on the primary and the secondary Callow cells are retreated in a group of four Callow cleaning cells from which the tailings are returned to the head of the primary Callow cells. On the tables of the Deister Machine Co. a separation is made between concentrates and tailings, but no middlings are produced and no part of them is reground. This, however, is probably only a temporary arrangement, the idea being that a suitable regrinding and reconcentration process will be installed later, after having been worked out by extensive tests.

### *Flow Sheet of Inspiration Sections*

Thirteen sections of the Inspiration concentrator are equipp Inspiration flotation machines. The flow sheet of these sect represented in Fig. 20. The ore, after having been crushed an passes to the head of the flotation machines and traverses thei

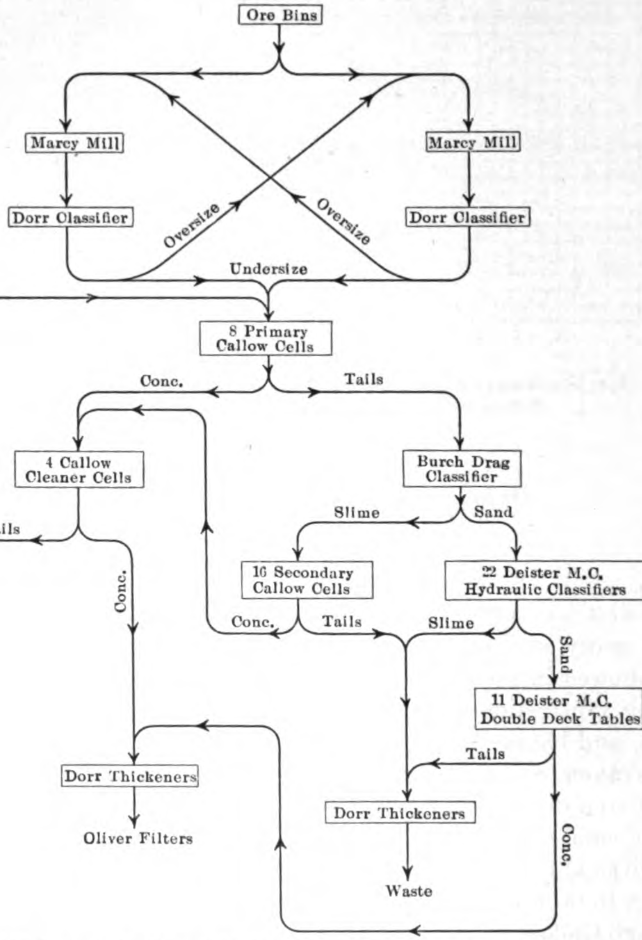



FIG. 19.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPP CALLOW FLOTATION MACHINES.

length. The resulting tailings are split, by the same kind of a mentioned before, into a sand and a slime product. The slime is run to waste. This is thought permissible because, after having through the whole length of the flotation machine, the slime h impoverished to such an extent that retreatment seems unne

The sand product undergoes the same retreatment as that described for Callow sections.

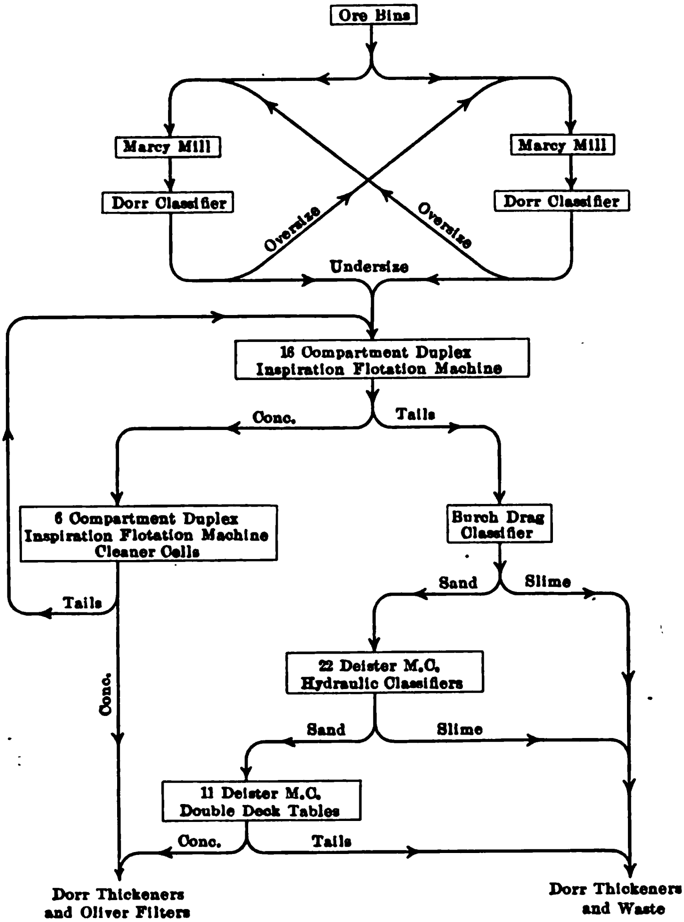
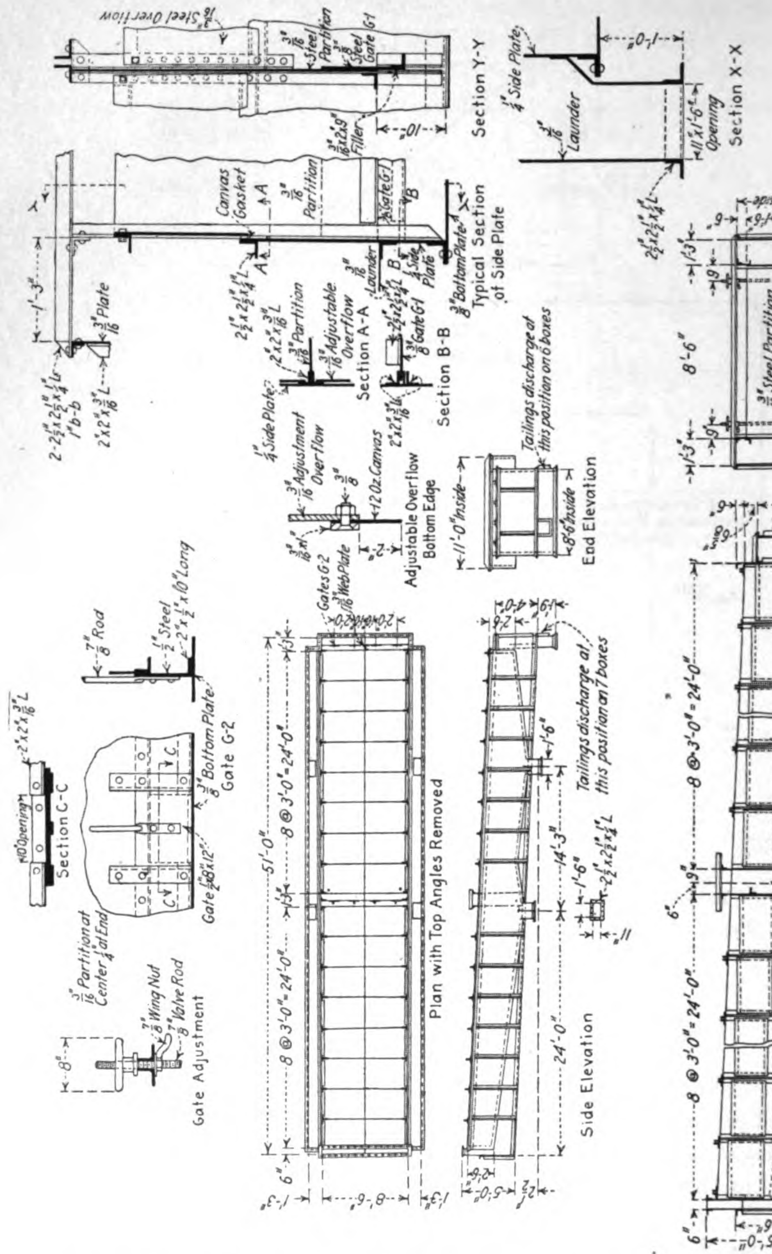



FIG. 20.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WITH INSPIRATION FLOTATION MACHINES.

*Inspiration Flotation Machine of Steel Construction*

Fig. 21 shows the drawing of an Inspiration flotation machine of steel construction as designed by Mr. Burch's office. At the end of each section of eight compartments, a pulp overflow is provided which regulates the level of the pulp in the preceding compartments of the flotation machine. The subdivision into two sections of eight is made in the interest of closer regulation of the pulp levels. It had been feared that if no such subdivisions were made, and the regulation of the levels accomplished at the tail end only, appreciable fluctuations might take place in the com-

partments near the feed end due to changes in the volume of pulp and might prove highly undesirable.




One point in the construction of the Inspiration flotation m in which we take considerable pride, is the method in which th

ced. The machines have solid bottoms; all pipe connections are om above and are at all times visible. The porous bottom rests on

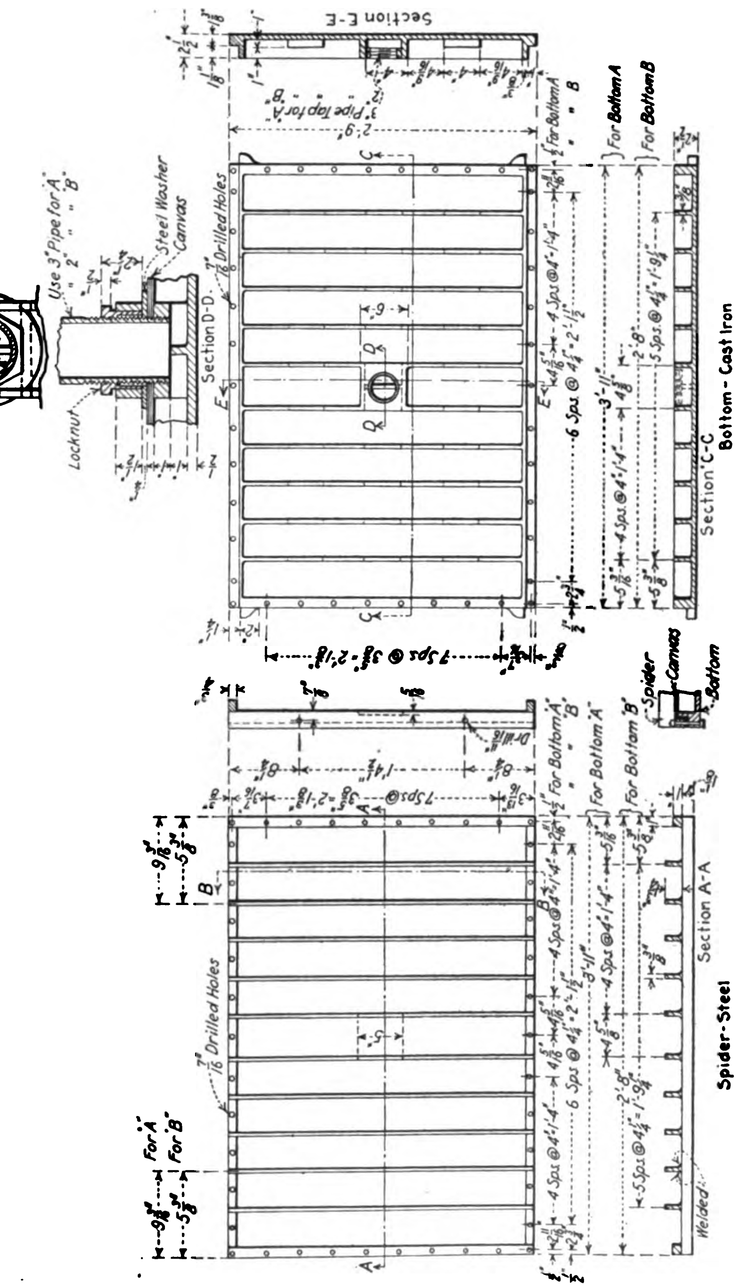



FIG. 22.—INSPIRATION FLOTATION MACHINE BOTTOMS.

the regular bottom of the machine and is introduced from above. ustrated in Fig. 22. As will be seen, it consists of a lower air

chamber having partitions running lengthwise, which, however, do n extend down to the bottom of the air chamber and, for this reason, not interfere with the interchange of air between the narrow compa ments thus formed. A grate, which forms the upper part of the por bottom, can be fastened to the lower part by a number of bolts. Bef putting them together, a porous medium (for instance, canvas) is pla between the two pieces. The air enters through a pipe from the which screws into the lower casting. Arrangements are made, of cou to secure an air-tight joint where the pipe passes through the por medium. When assembled, the bars of the upper grate form chan in the lengthwise direction of the machine. The bars are the only pa that protrude above the canvas and, on account of their arrangem longitudinally, they do not interfere with the passage of pulp through machine.

The cleaner cells consist of six compartments in series. B rougher and cleaner cells are divided by a partition in the middle, r ning the whole length of the machine. The operator is, therefore, enab to throw all the feed on one side of the machine when desired, thus p mitting repairs to be made on the compartments of the other side wh ever necessary.

In the beginning, it had been supposed that throwing all the f on one side of the flotation machine might seriously overload that s and for this reason, in the first sections installed, provisions were mad permit removing and replacing the porous bottom from any one compa ment without interrupting the flow of pulp. As far as lifting out bottom is concerned, this, of course, can be done in any case, but w one of the bottoms is withdrawn sand will pile up to a certain exten the middle of the compartment. To remove this sand before return the porous bottom, a system of perforated water pipes was installed un the air chamber. They allow the washing out of the bottom of machine by the application of water pressure.

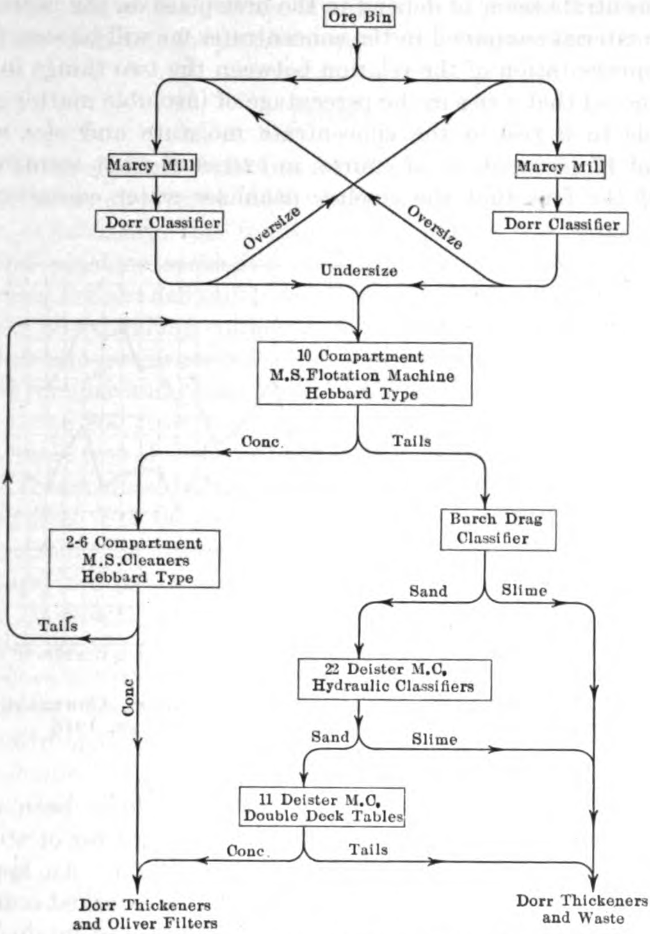
### *Minerals Separation Section*

One section of the concentrator is equipped with machines of Minerals Separation Co., Hebbard type. The flow sheet, as illustra in Fig. 23, is practically the same as the one shown in the Inspirat sections. The only difference is that two cleaner cells in parallel used in place of the one cleaner cell of the Inspiration type. The dr ings of the machine are reproduced in Fig. 9.

### *Filter Plant*

It is necessary to reduce the ore to a certain fineness before the min contents of the feed can be treated by flotation, and as fine concentr

e characteristic of retaining water with greater tenacity than oncentrates, a filter plant is a necessary adjunct of a flotation In the Inspiration mill, both the flotation and table concentrates to Dorr thickening tanks, five of them being 60 ft. in diameter e of them 80 ft., representing a total area of about 29,217 sq. ft.,




-FLOW SHEET OF INSPIRATION CONCENTRATOR SECTION EQUIPPED WITH MINERALS SEPARATION FLOTATION MACHINES.

e daily production of concentrates amounts to about 600 tons, g area of around 48.7 sq. ft. per ton of concentrate sent to the tanks. The settling tanks are each provided with a double ring baffle boards to prevent the foam that forms on the top from nating the overflow. The settling used to be carried out in two ome of the tanks serving as preliminary settling tanks, while

others resettled the overflow from the preliminary tanks. At pre we operate all the concentrate settling tanks in parallel. The sp discharges feed six Oliver filters. The ratio of solids to water is about to 1 at this point. The filters reduce the concentrate moisture t average of about 17 per cent. of the wet pulp. This figure varies, h ever, within rather wide limits from day to day. The moisture cont of the concentrate seem to depend in the first place on the percentag insoluble material contained in the concentrates, as will be seen from graphic representation of the relation between the two things in Fig It will be noted that a rise in the percentage of insoluble matter gener corresponds to a rise in the concentrate moisture and *vice versa*. moisture of 17 per cent. is, of course, in excess of what seems desir in view of the fact that the smelter penalizes water contained in

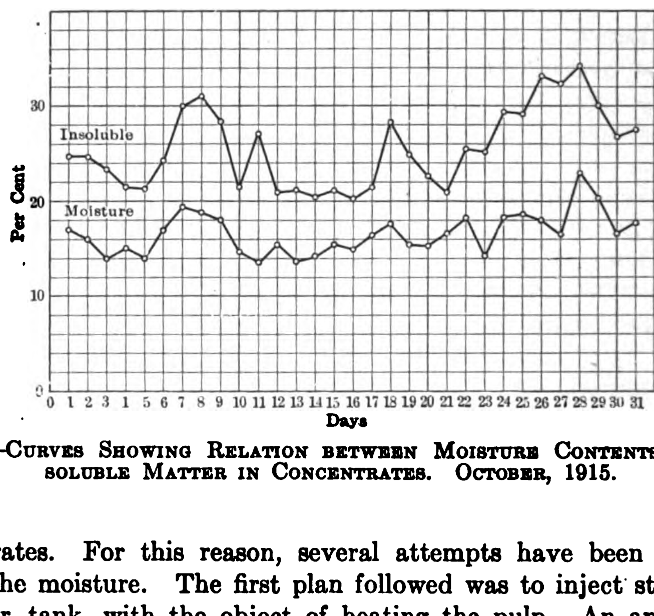



FIG. 24.—CURVES SHOWING RELATION BETWEEN MOISTURE CONTENTS AND SOLUBLE MATTER IN CONCENTRATES. OCTOBER, 1915.

concentrates. For this reason, several attempts have been mad reduce the moisture. The first plan followed was to inject steam the filter tank, with the object of heating the pulp. An appreci increase in the capacity resulted from this, while no marked reductio the moisture contents took place. The same effect was obtained by addition of slacked lime, which we tried lately in our Oliver filters. shows itself very readily in the formation of a thicker filter cake, causing a larger output of the filters. We have, however, not been to effect a reduction in moisture by the use of lime. We have ado the addition of lime to the filter pulp in regular operation on accour its decided benefit as far as the capacity is concerned. As a rule, filters can handle the whole output of the concentrator; that is, eac by 12-ft. Oliver filter has a capacity of about 120 tons in 24 hr.

## Results Obtained in Commercial Concentrator

account of the fact that the concentrator operated by the Inspira-
npany has only just left the stage of construction, it is not possi-
ive results obtained in the operation of it as a whole. As, however,
rval of several months has elapsed between the starting of the
d the 18th section, we are in a position to give at least preliminary
derived in actual operation on a fairly large scale.

### *Tonnage*

h of the sections of the Inspiration concentrator has a rated capac-
00 tons; at present most of the sections exceed 900 tons actual
y. Ordinarily, this tonnage is treated on one Standard duplex
tion machine consisting of 16 double compartments, while the
rates are cleaned on a cleaner cell with six double compartments.
e of the roughing compartments is 3 ft. by 4 ft. 3 in.; the size of
ner compartments is 3 ft. by 3 ft. Accordingly, the total area of
ed rougher and cleaner cells is 492 sq. ft. Assuming the tonnage
l to be 800 tons, this makes a tonnage treated of 1.62 tons per
square foot of surface. In the course of construction, the erection
tion machines did not always keep pace with the erection of the
g mills. For this and other reasons, it has frequently been neces-
combine the product from the two grinding sections in one flota-
tion; in other words, to throw an overload of 100 per cent. on the
n machines. Our experience is, that overloading the machines
per cent., or raising the tonnage to 3.24 tons per square foot of
does not increase the tailing losses very materially.

ollows, therefore, that the machines handle an overload without
additional copper loss, and it has been our experience that there
echanical trouble whatever, connected with overloading the flota-
chines to such an extent. We have also had to resort to over-
the Callow cells, and have likewise found that the recovery does
off seriously.

the Minerals Separation machine of the Hebbard type, which is
d in our Section No. 10, we have no figures of a similar character.

### *Air and Power Consumption*

have found that three blowers, of a capacity of 8,000 cu. ft. per
each, are sufficient to supply air for six sections of flotation ma-
Accordingly, one section requires 4,000 cu. ft. of air per minute.
flotation machines in an Inspiration Section have a surface of 492
ollows that the air consumption per square foot of porous surface

is 11.8 cu. ft. per minute; the maximum pressure required is 4½ the blowers and practically the same at the flotation machines, a is no serious loss of pressure in the air-conduits. The power co for furnishing air has been determined carefully by wattmeter r of the motors furnishing power for the blowers. The actual pow sumed is 87.5 kw. per section; or, figuring again with the rated capacity of 800 tons, 2.63 kw.-hr. per ton of ore treated. We h been able to make separate tests for the Callow and Inspiration s but assume that the air consumption is approximately the same i We have no figures yet for the Hebbard type machine. To get t amount of power consumed in connection with the flotation m a small amount of power, probably about 10 kw. or 0.30 kw.-hr. of ore, has to be added to the power consumed for the productio on account of the fact that the cleaner tailings are pumped back rougher cells and retreated, and that in the Callow sections th overflow of the drag classifiers is repumped to the secondary f cells. In the Inspiration sections, 0.425 sq. ft. of actual open surface and 0.615 sq. ft. of flotation machine area are provided treatment of 1 ton of ore in 24 hr.

*Floor Space*

The total floor space of the flotation floors amounts to 25,200 or 1.75 sq. ft. per ton of ore treated in 24 hr. (rated capacity). T floor space occupied by the concentrator proper (not including, h the crushing plant located at the mine) is 81,900 sq. ft. or 5.68 sq. ton of ore, and the total floor space, if the settling-tank installation the main mill building is also included, is 250,890 sq. ft., or 17.4 per ton of ore (rated capacity).

*Water Consumption*

The water added to the pulp where it reaches the flotation m is about 3 tons of water for each ton of solid pulp. The subs treatment, in our case, requires a further addition of nearly 3 water per ton of dry ore. However, as it is carried out on pulp t been deprived of its slime contents by passing it over drag classifi water thus entering the table tailings can be easily removed again. is accomplished in our case by settling the table tailings separate the slime tailings. The settling of the sand tailings is effected in a tank of home construction, consisting of a rectangular box with a bottom into which a number of Caldecott cones are inserted. The ing overflow is not quite clear, but is made so by resettling in thre Dorr tanks.

of the total quantity of water actually added to the ore during age through the mill, amounting approximately to 6 tons of water of ore treated, as mentioned above, nearly 3 tons, *i.e.*, the amount luring the table treatment, can, as explained above, be reclaimed asily. For the reclamation of the rest of the water, settling ponds, vell-known construction in which the reservoir is formed by build-taining wall of sand across a gulch, are resorted to in addition to ickening tanks.

*Recovery*

recovery obtainable by the application of our milling process is ned entirely by the composition of the ore, *i.e.*, by the relation 1 sulphide and oxide copper contained in the ore. Our average e copper extraction has been 90.39 per cent. for the months of April and May, 1916, the last months for which figures were avail-the writing of this paper. A certain recovery of the oxide-copper s, especially carbonates, is made in the flotation process as well as gravity concentration process. The percentage of such mineral y is low, probably about 25 per cent. For this reason, the re-obtainable on ore containing a high amount of oxide, such as ore, is correspondingly lower. We have worked in the laboratory e object in view of increasing the oxide recovery; for instance, by certain chemicals to the flotation pulp. An account of the results d will be found below, but we have not yet applied this method to ating scale, nor have we decided on using one of the other methods ole for this purpose, such as leaching.

le 5 gives average screen analyses of the feed and the general of the Inspiration concentrator for the months of March, April y. A segregation is made in the copper assay between sulphide de copper, because, considering the present stage of the art, we isfied with our mill work whenever the sulphide copper content aill tailings is low. As will be seen from the tabulations, a better y is made on the −200 material than on the coarser constituents re, which proves the point that for ores of the character of Inspi-re sliming is not to be feared since the introduction of the flota-ocess.

*Oil Consumption*

erience has shown that in the flotation treatment of our ores, we e flotation agents up to 1½ lb. per ton of ore. At present, the ture contains about 95 per cent. crude coal tar and a little less per cent. of oils derived from the dry distillation of wood.

different tars that we have tested during the operation of our mill nown greatly varying qualities as far as their flotation value is

concerned. The first tar that we tested and made use of was made from domestic coal, and happened to be a very serviceabl tion agent. Since that time, we have tested tars from several and have found some that are suitable for our purposes, while oth less so, and still others even entirely unsatisfactory. We have ol satisfactory tar products from New Mexico, Colorado, Missou Illinois. These States furnish at present as much as we need consumption. For awhile, it seemed possible that we might h import from a long distance the large quantities of tar that we During that period we tried to find substitutes, and looked esp toward the utilization of fuel oil for this purpose, but we have no able to get as good results with any kind of fuel oil as with crude c

TABLE 5.—*Average Screen Analyses for the Months of March, A and May,* 1916, *of Flotation Feed and General Tails*

| Mesh | Flotation Feed | | | | General Tails | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Per Cent. Weight | | Copper Contents | | Per Cent. Weight | | Sulphide Copper Contents | | Oxide Copper Contents | | Tota C |
| | Cum. | Indiv. | Per Cent. | Grams | Cum. | Indiv. | Per Cent. | Grams | Per Cent. | Grams | Pe Cen |
| + 65 | 9.5 | 9.5 | 0.45 | 0.042 | 9.5 | 9.5 | 0.18 | 0.017 | 0.12 | 0.011 | 0.3 |
| +100 | 21.2 | 11.7 | 0.86 | 0.101 | 21.2 | 11.7 | 0.19 | 0.023 | 0.14 | 0.016 | 0.3 |
| +150 | 33.5 | 12.3 | 1.91 | 0.235 | 33.5 | 12.3 | 0.11 | 0.014 | 0.19 | 0.023 | 0.3 |
| +200 | 39.2 | 5.7 | 2.69 | 0.154 | 39.2 | 5.7 | 0.14 | 0.008 | 0.19 | 0.011 | 0.3 |
| −200 | .... | 60.8 | 1.85 | 1.125 | .... | 60.8 | 0.06 | 0.036 | 0.47 | 0.286 | 0.5 |
| Totals...... | .... | 100.0 | .... | 1.657 | .... | 100.0 | .... | 0.098 | .... | 0.347 | .. |
| Assay direct | .... | .... | 1.62 | .... | .... | .... | 0.101 | .... | 0.318 | .... | 0.4 |
| Oxide...... | .... | .... | 0.36 | | | | | | | | |

Our experience is, that we can get along with coal tar alo is beneficial, however, to add wood-distillation products in small ties, for instance, those containing pine oil. While coal tar makes strong and heavy froth, such as appears to be required to keep mineral particles in suspension, the wood-distillation products ha characteristic of producing a multitude of froth bubbles, such a necessary to furnish the large surface required to save the ve mineral particles. Because the finer ore particles (slime), on a of their small diameter, expose a large surface, it is evidently ne to produce a correspondingly large surface of froth in order t them by flotation.

*Operating Cost for Flotation*

number of men necessary for the operation of large flotation
es is remarkably small. At the Inspiration plant, one operator
ses four sections of flotation machines. Two Mexican helpers
im in washing the bottoms, thus insuring a free passage of air
the porous medium. At the prevailing high prices of American
exican labor, this means an expense of somewhat more than 1.5 c.
of ore treated. The total expenses representing flotation proper
follows for the months of March, April and May, 1916:

| | Cents per Ton |
|---|---|
| abor | 1.62 |
| lotation oils | 1.65 |
| ther supplies | 0.35 |
| ower | 2.14 |
| Total | 5.76 |

osequent table treatment of flotation tailings, the filter treatment
concentrates and other operations connected with the process of
ration, belong more or less to flotation treatment, and their ex-
hould also be considered when the cost of the flotation process is
tablished. The total milling cost, exclusive of crushing and grind-
been for the past few months in the neighborhood of 20 c. When
t of crushing and grinding is included, the cost is about 40 c. per
re. Royalties for the use of the flotation process are not included
of these cost figures.

*Discussion of Results Obtained*

criticism has occasionally been raised against the Inspiration
y that they were slow in deciding on the design of a concentrator.
son for such slowness was, of course, that the development of the
process coincided with the period during which the fundamental
n the design of the concentrator had to be settled. The manage-
therefore, thought it best to be very conservative in installing
d concentrator equipment, which, in case the new process held
seemed to promise, might prove to be entirely superfluous. The
oint of interest, therefore, is whether it was wise for the company
instead of installing a gravity concentrating plant as had been
ly considered.

ar as the comparison of a flotation plant, as at present operated
company, with the gravity-concentration plant, as originally con-
is concerned, the recoveries actually obtained in the company's
plant are so much higher than those indicated by tests in the

original test mill as being obtainable in a gravity-concentration p
ore of the same character, that there cannot be the least doubt
system is the better.

In discussing the second question which comes up in this conn
whether, for ores of the character of Inspiration ore, gravity concen
supplemented by flotation would be preferable to the simple fl
process that has been adopted by the Inspiration company, the
point is this: Can the slime be reduced to lower copper content
combined process than by the process followed here? The reduc
the copper in the sand is, as millmen know, simply a question
grinding combined with suitable concentration.

In support of the principle followed by the Inspiration com
desire to call attention to the screen analyses of our general tailing
sented in Table 5. They show that the sulphide copper left
−200-mesh material is very low.

In addition, the floor space required for a plant like this is ma
smaller than for a plant of the combination plan, which also mea
the construction cost is much lower; and, further, the operating c
low by reason of the greater simplicity of design. I hope these fa
sufficient to decide the point at issue as far as the ore of this di
concerned.

I hope that our neighbors of the Miami company, whose mill w
before the flotation process was known, will uphold me in my stat
If this point is admitted, it will endorse the principle followed in the
of the Inspiration mill, of doing away with reduction in stages a
centration in stages.

It must be granted that the settling of flotation-concentra
requires extensive floor space, as shown by the figures referred to
and that settling and subsequent filtering absorb a certain fractio
mill operating costs. However, as a ratio of concentration in the I
tion mill is 25 into 1, only 1/25 of the ore is thus treated. The
expenses increase, of course, proportionately for ore which does n
mit concentration with such a high ratio as the ore existing in
spiration mine. Therefore, there will be a point beyond which a d
system of concentration is preferable. Just what the critical r
concentration is, must be determined by calculation and testing
on individual conditions.

## Prospects for Future Development of the Flotation P

The flotation process is in its infancy. For this reason, theref
concentrator must be necessarily in the first stages of its develo
In what direction future changes may take place, is perhaps indic
tests which have been made partly on a laboratory scale and par

somewhat larger scale, but which have not yet been incorporated into our regular milling process. Of these latent developments, I will try to give an outline in the following:

### *Porous-Bottom Experiments*

The porous bottom is, as one may imagine, the most essential part of a pneumatic-flotation machine. Our experience with the porous bottoms of the different constructions brought out very clearly the principal difficulty attached to them, which is, that the pores have a tendency to contract gradually and thereby to retard the passage of air through them. This tendency was more pronounced in the solid porous bottoms employed in the Flinn-Towne flotation machine than it was, for instance, in those of the Callow type, although the latter also show a tendency in this direction. Our first supposition was that the choking was due to the fact that the air entering below the blankets carried particles of dust, which would settle in the fine pores and reduce their area. Indeed, a canvas blanket will, after a certain length of service as a porous medium, always show some ring-shaped spots of dark color opposite the air inlets, clearly indicating that a deposition of dust particles on the blanket actually does take place. To make sure of this point we cut out round disks from a Callow blanket that had been used for some time and investigated their porosity by using them as porous bottoms in a glass tube standing in a vertical position. Air under pressure could be applied to an air chamber located underneath these disks, and the air passing through the porous blanket could be measured by a gas meter. The quantity of air discharged through the porous medium offers a measure of the porosity of the blanket, and for this reason, the velocity or speed with which the counter of the gas meter revolves, gives an indication of the porosity of the porous disks being tested. To our surprise, we found that the darkest points of the blanket were not those of lowest porosity. On the contrary, the points farthest away from the air inlet showed the greatest tendency to choke. An explanation of this paradoxical behavior seems to be offered by the fact that an air blanket is kept in a state of more or less agitation near the air inlet (in the Callow machine this happens to be a point remote from the places where it is held rigid) while farthest away from this point the blanket assumes a state of comparative rest. Incrustations, due perhaps to the presence of soluble salts in the water in conjunction with fine slime, always form to a greater or less extent in the top layer of the blanket. Evidently, the agitation counteracts the formation of the incrustation, while there is no such counteracting influence in the portions which are essentially at rest. For this reason, we concluded that a solid porous material is not suitable as a diaphragm in a flotation machine of the pneumatic type, if a bottom of long life is required. As a matter of

fact, the experience of everybody who experimented with solid b
seems to have pointed in the same direction. Mr. Cole for a while
out carborundum tubes in his machine. We tried carborundum
in the flotation machine of the Inspiration type and abandoned
and I believe that even Messrs. Flinn and Towne have, in the mea
given up the solid bottom of their original design.

A necessary condition for a serviceable flotation bottom a
therefore, that the porous medium be of a flexible nature. Th
canvas stitched every half inch or so which Mr. Callow's first cel
tained and which we have used for considerable time in the Insp
machines, seems to answer this purpose fairly well. We find, ho
that to keep it in good working condition and prevent incrustation
forming on the top, we have to clean it frequently. This is d
dipping an iron pipe connected with a water hose into the compar
and sweeping the canvas bottom with the jet of water dischargin
the lower end of the pipe. The canvas blankets seem to last for
6 months at the most. As they are inexpensive, the replacing of a
after that time is not a serious item in the operating costs. The
out of the canvas is due to the wear caused by the frequent cl
The top layer wears out first, the holes created by the stitching f
nuclei for the formation of larger holes. By the time the top lay
a number of holes the canvas blanket is generally discarded.
interest of greater economy, we intend giving up interstitching the
of canvas. We are trying to decide whether it is better to use
sheets of thicker fabric or to use canvas similar to the kind that w
been using and to put several layers on top of one another w
interstitching them. The latter has the advantage of requiring t
carding of only one layer, when it becomes defective.

There will always be some tendency to form incrustations so
canvas is used for flotation mediums. Their formation will be e
prevented only by substituting an altogether different materia
have made experiments in this direction. One of my former assi
R. H. Haskell, deserves credit for suggesting them. For instan
substituted for the canvas blankets, thin rubber sheets perforated
multitude of needle holes and obtained an excellent froth. The
tion to their use is that their life is limited. When sheets of ru
an increased thickness are used, the needle holes require too muc
sure to form openings of sufficient size for the passage of air, and t
a thick rubber sheet suitable for this purpose, slits several milli
long have to be substituted for needle holes. We have had one
rubber bottoms of this design in operation, but, just at present
not ready to substitute them for canvas blankets. We also
blanket made from a material that goes under the name of sponge
and can be produced with rather fine texture. We have not bee

er, to obtain lastingly good results from the use of this medium. rmore, we tried a woven fabric containing rubber threads in one on and threads of cotton or the like in the other direction and a ized canvas made by the Goodrich Rubber Co. We are not ed to use any of these materials on a commercial scale.

advantage of rubber should be, in the first place, that on account moothness it would have less tendency than canvas to permit the ion of incrustations. Besides, an elastic medium should have the nal advantage of avoiding the danger of catching small sand or articles in the pores of the medium, as an expansion of the medium nay be effected, for instance, by increasing the pressure) would the pores and remove such particles. We think that our experi- work in this direction is encouraging.

*Raising the Grade of Concentrates*

recovery that it is possible to effect in a flotation plant depends on the grade of concentrate desired. With a low grade of con- e, low tailings can be made, but when a high grade of concentrate ılated, increased tailing losses cannot be avoided. A question ggests itself in this connection, and which we have tried to answer oratory experiments is, "How can we raise the grade of our con- es—that is, reduce the percentage of insoluble matter contained n—without entailing additional copper losses?" We know from ory experiments that this can be done by expensive methods— tance, by heating the solutions—but such a procedure would be able from an economical standpoint. Experience has shown us ncentrate produced in the first compartments of the cleaner cells ys freer from insoluble matter than the concentrate produced in t compartments. The problem then resolves itself into finding ble cleaning process for the concentrate from the last compart- of the cleaning cells. By treating this low-grade concentrate hot, e addition of caustic soda, we have been able to separate it into a ade concentrate and fairly low tailings. This method necessitates e expense of heating a small fraction of the pulp and may be a rcial possibility.

*Recovering Carbonates by Flotation*

other subject on which we have spent considerable time in our ory is the problem of recovering copper carbonates by flotation. we started our flotation plant, we discovered, to our astonishment, e machines not only saved a high percentage of copper sulphide at they also recovered some of the carbonates. Ever since that e have tried to find means of improving the carbonate recovery.

In the first place, we studied all of the oils that seemed to hav tendency to cause the flotation of such minerals. Later on, we tr other means in addition to the variation of the oils. One way in wh copper carbonates and similar minerals might be recovered was outli by Alfred Schwartz in his United States Patent No. 807,501. process consists in first artificially producing a sulphide coating on s oxidized minerals by the introduction into the pulp of soluble sulphi and then adding suitable "oils" and effecting the flotation. If it w possible to thus chemically produce coatings of sulphide identical v the surface of the minerals formed by nature, this process would w well, as evidently the nature of the surface is the only characteri that determines whether a mineral will float or not.

The Minerals Separation Co. owns a number of patents cove this subject. Their English Patent No. 26,019, issued to Sulman Picard, describes the flotation of oxide copper minerals by similar me

I am not aware that equivalent patents have been issued in the Un States. The English patent in question is of a later date than Schwartz patent above mentioned. The representatives of Minerals Separation Co. have experimented more or less extensi with this system, while demonstrating their machine to the Inspira company. As far as I know, they have not proven its practicabil In the course of their experiments, they tried the application of sod sulphide and sodium polysulphide for this purpose. The latter produced by treating sulphur with hot caustic soda. At the time t experiments were made, I was not familiar with the chemical ac taking place, which, as much as I know now, actually results in the for tion of polysulphide mixed with thiosulphates and other oxygen-sul compounds. The failure of their experiments, I therefore ascribe the fact that perhaps a polysulphide, which they were anxious to m was not actually produced. I proceeded to make sodium polysulp by a method which I knew, that is, by the treatment of a sodium phide solution with sulphur powder. When we applied this rea to some of our carbonate ores in laboratory flotation experime we noted that a good recovery was obtained. The composition of compound was varied materially in order to find just what composi gives the best results in the flotation of carbonates. Our experie seems to indicate that sodium sulphide alone encourages the flota of carbonates, but that sodium polysulphide, or sodium sulphide w contains more sulphur than would correspond to the chemical form $Na_2S$ gives better results. The addition of caustic soda besides sodium polysulphide was found beneficial.

The question then arose as to why we succeeded in effecting flotation of oxidized copper when the experiments of the member the Minerals Separation staff failed. Tests along these lines brou

fact that the Minerals Separation compound when applied to bonate ores also worked successfully, but that it did not on our milling ore. Our own compound when added to our mill ore increased the recovery of the carbonates, but evidently interfered e sulphide extraction, and for this reason seemed to be of as little he compound of the Minerals Separation Co. When applying s of this character to tailings resulting from ordinary flotation nt, with the point in view of effecting a sufficient sulphide ex- by the regular flotation process, and using the compound in n only for the purpose of increasing the carbonate extraction, e found so far that the increase in copper-carbonate recovery over e obtained without the addition of such chemical compounds orth going after.

this is only a consequence of the fact that carbonates exist in nall amounts only in our milling ore and are partly saved by inary flotation process.

re is no real difficulty about saving carbonates by the method ned, if they exist in quantities that make it worth while to save That copper carbonates can be recovered may easily be demon- by treating a deslimed feed in a series flotation machine. If at nt of the machine, where the sulphide recovery is nearly finished, sulphide is added, the decidedly green color of the concentrates following compartments leaves no doubt on this point. The ng of the feed seems to assist in the carbonate recovery.

s of considerable (even if only theoretical) value to establish why sulphide and polysulphide tend to increase the recovery of carbonates. A coating that might be expected to form cannot be d. The concentrate resulting from the treatment of pure car- ore is decidedly green; besides, when an alkaline condition of the used there is very little, if any, tendency for any sulphide coating , and the alkaline state of the pulp is (as explained above) exactly ndition under which the best carbonate extraction results. An- point that seems to contradict the explanation of these results assumption of a sulphide coating is, that when we proceeded as suggested by Mr. Schwartz—*i.e.*, when the application of sulphide was followed by the addition of flotation agents and actual flotation—we seemed to obtain poorer results than when cedure was reversed by applying the oil first and following with plication of some soluble sulphide, although the latter method ertainly seem less favorable to the formation of a sulphide coating, rhaps for this reason has not been suggested by Mr. Schwartz. other theory that has been mentioned as an explanation of this nenon is that colloidal sulphur is formed by the solution of sodium phide in water, which, as is known, is a good flotation agent. For

instance, it is pointed out in the United States Patent No. taken out by Dr. R. F. Bacon of the Mellon Institute in Pi that by setting free colloidal sulphur, say by the reaction of sulphide with sulphur dioxide, good flotation results may be as far as the flotation of sulphides is concerned. To make th available for the flotation of carbonates and other oxidized cop erals, he suggests that a sulphide coating be first formed on the *i.e.*, to follow Mr. Schwartz's idea. Whether the colloidal su itself has a beneficial influence on the recovery of the carbonat been suggested in explanation of our observations) seems rather when it is considered that we have obtained good results in alka tions in which colloidal sulphur does not seem to separate out fr sulphide containing only a limited amount of sulphur such as wa our tests. The full theoretical explanation of these facts must be left to future investigations.

## *Recovery of Silicates*

In our experiments with the object of saving the oxidize minerals, we soon found that we could save some of these miner others were entirely refractory to the method above mentio establish which minerals could be saved and which not, we a an analytical separation into carbonates and silicates. The methods which we tried for the purpose of distinguishing bet two proved unreliable, however, and we had to resort to the se by specific gravity (panning). The carbonates of copper (mala azurite) are heavier than gangue, and the silicates (chrysoc lighter. The separation is rather difficult, owing to the small in specific gravity, and the results are therefore far from being a reliable, but they seem accurate enough to indicate that the m saving carbonate copper above referred to is of value only for the of carbonates and does not apply to silicates. This fact see another corroboration of the assumption made above, that carb copper do not float simply because of the formation of a thi coating of copper sulphide. It can easily be verified in the la that silicates can be coated with copper sulphide fully as easily carbonates. For this reason, if the filming theory is right, it s possible to float silicates just as well as carbonates. There is that they can be floated by transformation into sulphides, transformation must not be confined to the surface, but must g Our experience is, that to effect a good recovery, it is necessary t the pulp so strongly that practically all of the silicates of coppe solved and by the action of hydrogen sulphide or other soluble are transformed into the state of chemically precipitated copper

form there is no difficulty about the recovery of the copper by n, but this procedure is not entirely without objection.

ase hydrogen sulphide gas is used, the acid combined with copper ierated. This tends toward a low acid consumption and a good extraction, on account of the fact that the treatment winds up small percentage of copper in solution and free acid present, both h are desirable in the light of the law of chemical mass action. drogen sulphide is not a desirable reagent. The fact that it is a not a liquid introduces complications in the apparatus which are ated by the fact that it is poisonous and obnoxious otherwise.

er soluble sulphides used in place of hydrogen sulphide will neusome sulphuric acid with the result that the acid consumption higher and the copper extraction lower than in case of hydrogen e gas.

far as acid consumption is concerned, it is pointed out that the id lost with the pulp may be settled out in ponds and re-used. er, the re-use of acid diluted to such an extent is a more serious n than is generally realized.

treatment of concentrates that are "colloidal," to a much greater than ores which millmen have been in the habit of calling colloidal, additional problems, which, however, may prove not to be as as they look.

rything considered, I cannot see that the flotation treatment of d copper ores after previously leaching them offers better prospects raight leaching by decantation and precipitation by other methods.

### *General Theory*

y much has been published recently about the theory of the flotaocess, and very many suggestions have been made that will probrove valuable after it has been shown by critical tests how far plain the facts.

eems to me that an explanation of the qualities of the flotation not as difficult as it might appear. The problem only seems so cated because the flotation qualities of an oil or an oil mixture ot been separated into their components. In fact, it requires a ation of qualities to make a successful flotation oil. In the first he flotation oil has to coat the mineral particles. That there is a cy for the formation of such a coating can easily be seen from simple nents. For instance, if samples of copper sulphide (chalcocite), carbonate (malachite) and gangue (silica) of the same screen size ead out on watch glasses and then moistened with a drop of coalosote, it will be seen that the drop of creosote soon disappears h absorption by the copper sulphide, while it takes a much longer r it to be absorbed by the copper carbonate and a still longer time

with the gangue. On the other hand, when a drop of water is pl the same minerals, it will disappear on the gangue first, later on bonate, and finally on the copper sulphide. This evidently pro in a mixture of water and oil, the oil will attach itself with prefe the sulphide particles while the water will have the greater tend wet the gangue.

The second quality which at least is sometimes required of a f oil, is that it has to form a stable froth. In such a case, the stabil be secured by more firmly cementing together the mineral, air To accomplish this, oils are used which have a tendency to floa divided gangue particles. The action is characteristic of the pine distillates like pine tar and the lighter ones like turpentine are crude, unrefined products; in other words, when they contai of the heavier distillates. I am not quite sure, however, whet beneficial influence of oils of this group is not perhaps rather du fact that they remove colloidal material from the pulp and improve its tendency to float minerals.

A third quality demanded of a successful oil mixture is that be able to produce a sufficient volume of froth. This property i plified best by oils of the soluble type—cresol, pine oil, alcohols an substances. It can easily be proven that when oils of this type a although they may be considered insoluble, the water acquires th ing qualities of the oil. It may be demonstrated by shaking a this character with water and permitting the oil to separate ou It will be found that the water has acquired frothing qualities by going this treatment. It is perhaps even likely that the soluble of the oils belonging to this group is the only portion that is activ manner. The difference between the oils of group 1 and group 3 studied, for instance in a flotation machine of our type. It will b that the heavier mineral runs over the concentrate discharge la the first compartments forming a heavy, dark froth and the insoluble portions of the flotation oil mixtures apparently go Toward the tailings end of the flotation machines, most of th material has disappeared and the froth is lighter and of a more nature. The pulp, however, has not lost the quality of formin even after it gets to the last compartment of the flotation machines permits the conclusion that the frothing characteristics follow the pulp. The water settled out in tanks and tailing ponds has frothing qualities. Such water behaves in a similar way to alcoholic solutions with which we are used to associate this charac for instance, beer or champagne. The experience of mills using th tion process, that when the tailings water is reclaimed the qua frothing oil may be considerably reduced, further supports the assu that the formation of froth is caused by water-soluble substance

Just how the surface tension of water and air must be modified to permit the formation of froth has been made the subject of some speculations recently published by different authors. We have also devoted some thought and a few experiments in our laboratory to this question. A discussion of these matters belongs to the realm of physics, however, and is outside the scope of this paper. But I might add this remark to the discussion of the subject of flotation oils, that most flotation oils not only have the characteristics of one group, but may at the same time possess those of another one. For instance, coal tar has qualities 1 and 3. For the flotation of our ordinary milling ore we do not require much of the quality of oil classified in group No. 2, and, therefore, can get along with coal tar alone. But we find it advisable to add to the coal tar more of the qualities characteristic of the third group, and for this purpose, we add about 5 per cent. of the total in the form of crude pine oil. In many cases it is found that the flotation oil has the characteristics of group No. 2 to such an extent that it is impossible to make clean concentrates. Various chemical means, such as the addition of acid or alkali, are used to counteract this.

## Conclusion

In summing up I want to say that the fact that the Inspiration company has been able to design a commercially successful flotation plant and has found ways that hold out prospects of raising the plant to a very high state of efficiency, must be attributed to the policy followed by the company of spending great sums of money for the purpose of investigating the flotation process on a commercial scale. In carrying out these investigations, a close coöperation between laboratory and operating force helped us, I believe, more than anything else. I would like to give credit to each person who had a share in contributing toward the success of the work, but cannot do it, because the list would be too long.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 39 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# A New Flotation Oil

BY MAXWELL ADAMS, RENO, NEV.

(Arizona Meeting, September, 1916)

CONSIDERABLE interest has recently been developed in sage-brush oil because of its possible utilization as a flotation agent in the mining industry. A list of some of its physical properties, together with the method used in its extraction, may prove of interest at this time.

Something over a year ago, a study of the essential oils in desert plants was begun in the Chemical Laboratory of the University of Nevada. None of the oils so far studied possess properties of special interest to engineers, except the oil of sage, *Artemesia tridentate*, which has exceptional power as a flotation agent. This plant, known as common sage brush, also called black sage, is widely distributed over the semi-arid West, being found quite generally on most of the dry plains and mountains west of Missouri.

The method of extracting the oil followed in these experiments is very simple. The leaves, twigs and small branches are placed in an air-tight drum, having a capacity of about 27 cu. ft. Steam is admitted through a number of small openings at the bottom of the retort, and the pressure maintained at 20 to 25 lb. per sq. in. for 3 hr. The escape of the steam from the retort is regulated by allowing it to pass through a stop-cock into a condenser. The water in the receiver is drawn off from time to time and the oil, which is insoluble and floats upon the water, is thus collected. At the end of 2 hr. most of the oil has been driven out, though traces continue to come over for a much longer time. By raising the pressure, the time required could probably be shortened and the yield increased, but the lack of laboratory equipment has prevented the carrying out of this experiment.

The stock wood, bark and branches contain no oil, the distribution of the oil being limited to the leaves and young shoots. There is a seasonal variation in the amount of oil contained. Samples collected on different dates gave the following amount of oil: May 1, 0.42 per cent.; May 27, 0.6 per cent.; June 30, 0.72 per cent.; Aug. 1, 0.9 per cent.; Sept. 10, 1.0 per cent. The increase appears fairly constant from early spring, when the leaves first appear, until light frosts occur in the autumn. When the plant is air-dried there is some loss of oil, as the

following data will show: Two 100-lb. samples were collected at same time. One was distilled when green; the other was air dried 10 days before distillation. The green sample yielded 275 grams, the dried sample 248 grams of oil, showing a loss of about 10 per cen

A laboratory experiment can furnish little data useful in forming estimate of the commercial cost of production. A man working fo hr., and using a pair of common pruning shears, collected twigs wh yielded 1 lb. of oil. Since only a small percentage of the oil is los the brush is dried, the most economical method of production wo perhaps be to collect it in large quantities, by using a tractor engine a drag, in some such way as land is cleared for farming. When the br is dry, the leaves and young shoots are easily shaken from the lin Thus the amount of material to be distilled would be greatly diminis and the oil perhaps obtained at a cost and in quantity sufficient to m it available as a flotation oil, if not alone, possibly as an ingredient increase the flotative power of other oils.

The crude oil is dark in color. When redistilled with steam i water-white at first, changing gradually to a straw-yellow color u standing. It has the following physical properties: Density at 15 0.9206. Refractive index at 20°C., 1.4732. Rotation at 20°C., −4 At 98°C., a light oil, with a very sharp and pungent odor, begins to till, but only after the temperature is above 165°C. does rapid distillat take place. At 180°C., the oil turns dark and decomposition beg At a pressure of 12 mm., and below 125°C. almost all the oil can distilled.

The chemical properties of the oil are as yet undetermined. Th are small quantities of alpha and beta pinene. The main part of oil has a camphor-like odor and taste, but has failed to give the ordin tests for ketones. The fraction boiling at 175° to 180°C. has some the properties of ordinary cineol, but is acted upon by metallic sodiu which indicates that the chief ingredient is not cineol. The chem composition, which has little interest in this connection, will be work out later. The important question for the engineer is: Can the be produced in quantity and at a cost that will make it available for flotation?

SSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the eeting, September, 1916, when an abstract of the paper will be read. If this is impossible, sion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West ., New York, N. Y., for presentation by the Secretary or other representative of its author. ial arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion reafter should preferably be in the form of a new paper.

# A New Source of Flotative Agents

BY G. H. CLEVENGER, PALO ALTO, CAL.

(Arizona Meeting, September, 1916)

reagents now used in flotation consist of various acids or salts, ay be either electrolytes or non-electrolytes, dissolved in water e substance or combination of substances which function as col- r frothing agents. At times the only dissolved salts present are turally occurring in the water used. The general effect of the te is to greatly sharpen the separation between the gangue and entrate. Examples of this are: The use of sulphuric acid with s; and of sodium carbonate or calcium oxide (lime) with silver- s. Crude pyroligneous acid is also sometimes used when avail- 'arious oxidizing agents, such as permanganates, bichromates, added in the selective flotation of lead-zinc ores. Many other for performing certain specific functions, either real or imagi- ve been proposed, and a number of them have been tried upon a scale. The wild orgy of experimentation which is now going tation exceeds even that which followed the introduction of the process for the treatment of Comstock ores, when, among other age-brush tea and tobacco juice were reagents added to the pans. ll this will, of course, eventually come a more or less standard for the treatment of each class of ore.

ting a discussion of the functions of frothing and collecting he reagents used in flotation for this purpose may be classified ve general heads: (1) Essential oils; (2) fixed or fatty oils; ols and their combinations with organic acids; (4) coal tar and d products; (5) petroleum and its refined products.

tative agent of some kind is required in flotation as now prac- single reagent, as, for example, certain of the essential oils, orm the dual function of frothing and collecting agent, or a mix- ubstances may be required.

is country, the essential oils used have been wood products, been almost exclusively confined to the steam-distilled pine the present time, the supply of these is limited, and the cost rohibitive, so that their use has been dispensed with as far as The oils and tars resulting from the destructive distillation of

the various species of the coniferæ find a more general use on account the greater supply and lower cost, although even these have risen in pr and the future supply is somewhat problematical if the demand them in flotation continues to increase at the same rate as in the past

The fixed or fatty oils, with the exception of oleic acid and cru pyroligneous acid, are scarcely ever used, since they usually give p results as compared with other flotative agents.

Coal tar is a common and cheap product, but not all coal tars suitable for use in flotation. Furthermore, the marketing of coal ta coming more and more into the hands of large distributors, who contr for the output of the various gas plants. The refined products of c tar are useful flotative agents, but in the case of certain of these, for ample, carbolic acid and cresol, the price at present has become prohi tive. In general, crude coal tar yields the best results when mixed w other oils. It is a frequent practice to use a small proportion of pine in order to modify the froth. Fortunately, in many cases these chea mixtures yield very satisfactory results, and can now be obtained a reasonable cost.

It appears that only the crude petroleum oils having an aspha base are generally useful in flotation. These are invariably mixed w other oils. The refined products of petroleum are usually not sa factory, except as constituents of oil mixtures. The one exception this is perhaps the case of kerosene acid sludge, which is a byproduct petroleum refining. Kerosene acid sludge from eastern refineries g unsatisfactory results at Anaconda, but California kerosene acid sluc resulting from the refining of crude oils having an asphaltic base satisfactorily used in conjunction with a small proportion of wood cr sote for the treatment of Anaconda copper ores. While California ke sene acid sludge is a satisfactory flotative agent with many ores, available supply tends to limit its use. Formerly California refir threw it away, but now little of it is available upon the open marl since practically the whole of the present production is contracted fo long time in advance by pioneer users.

A suitable oil supply, both as regards the character of oil to give best results and its present and future availability, is a matter of seri consequence to companies operating flotation plants. It was with t in mind that I began the investigation of possible sources of flotat agents, with the particular object of affording relief to mines locatec the arid regions of the West, where none of the common flotative age are locally available, and where transportation costs upon those from outside are high.

Certain plants and shrubs have the peculiar property of secreting in the new growth, during the growing season, and particularly in leaves. Botanists appear to be uncertain regarding the function of

the metabolism of the plant. Some hold that it is a reserve food y for the plant, while others believe that it is waste product which ant fails to throw off. Be this as it may, many plants, shrubs and contain oil in the leaves and new growth. A good example of this case of the eucalyptus tree, from the leaves of which essential oils, are used for various purposes, are recovered by steam distillation. aves of the variety known as amygdalina, unique as being the tallest the world, have afforded a large proportion of the flotative agent n the concentration of complex Australian ores. This variety of fortunately very plentiful in close proximity to these ore deposits. the great arid and semi-arid mining regions of the West the most on of the few plants and shrubs native to the region are the varief sage brush known as mountain sage, pasture sage, wormwood etc.; also, in certain regions, greasewood and other shrubs of a simiture. It is, therefore, to sage brush that my attention has been ed in the search for flotative agents for concentrating Western

appears that the various varieties of wild sage were first investiby certain members of the Department of Agriculture, with regard possibility of producing from them by steam distillation essential itable for pharmaceutical use. Steam-distilled oil was prepared by k[1] from the variety *artemisia frigida* in 1905, and during the summers 7 and 1908 larger quantities of oil were prepared from specimens of ant, collected in South Dakota. In 1912, an essential oil prepared steam distillation of *Ramona stachyoides* (black sage) from southern rnia was also reported by Rabak. This oil was said to contain cent. of camphor. In 1914, Charles E. Burke and Charles C. ne[2] gave an account of an investigation of an essential oil which een prepared from the same shrub by the steam distillation of several ed pounds of leaves and twigs collected from brush growing near side, Cal. The yield of oil in this case was 0.9 per cent. of the t of the material used. This yield was somewhat higher than that ed by the Department of Agriculture. This is perhaps accounted the fact that the brush was collected later in the season. This oil orted as having the following composition:

| | Per Cent. |
|---|---|
| Pinene | 6 |
| Cineol | 30 |
| Dipentine, terpinene, etc. | 25 |
| Thujone | 8 |
| Camphor | 25 |
| Resinous material | 5 |

ank Rabak: Wild Volatile-oil Plants and Their Economic Importance, *Bulletin* 5, p. 22, *Bureau of Plant Industry.*

harles E. Burke and Charles C. Scalione: Investigations on Oil of Black Sage, *l of Industrial and Engineering Chemistry,* vol. 6, p. 804 (1914).

Camphor was separated from a portion of the oil, thus dem
the rather interesting possibility of black sage as a source
camphor. Upon request, Mr. Scalione furnished me wit
sample of the original oil. This oil is clear, with a very sligh
tinge, and has an agreeable odor. In fact, so far as appearance a
behavior goes, although the chemical composition is somewha

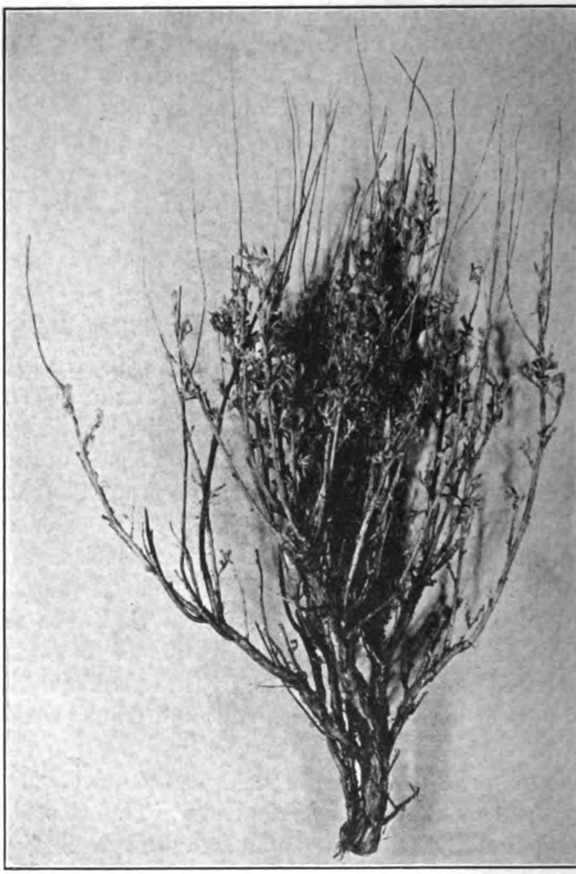
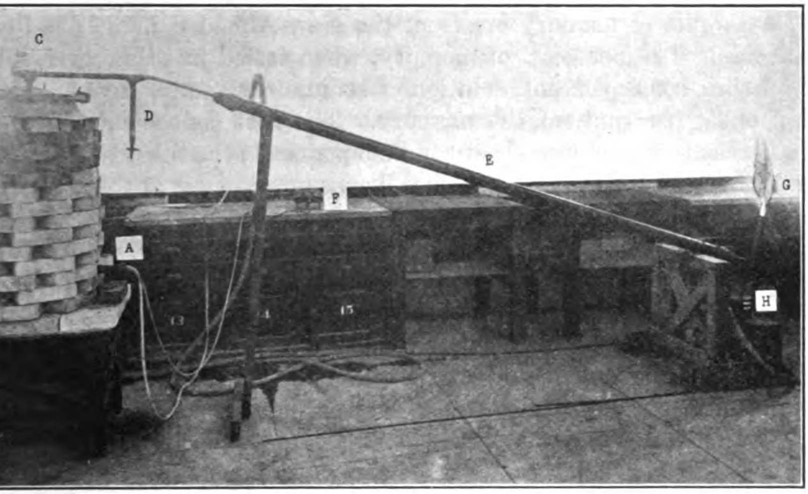


FIG. 1.—ARTEMISIA TRIDENTATA.

it very much resembles steam-distilled eucalyptus oil from the
variety, *amygdalina*. This oil is a good frothing agent, and
quite satisfactory results upon lead and zinc ores in a qualit
although the amount available did not permit of thorough inv
The idea of investigating sage brush and greasewood as possil
of flotative agents was conceived early in January, but it wa
early in March that the first 100 lb. of sage brush was forwar

gh the courtesy of G. B. Lantz. This lot was collected near Gold- Nev., and proved to be the variety *Artemisia tridentata.* "In the ern Arid Transition zone the flora consists largely of the true sage , Artemisia tridentata,"[3] therefore this would be the variety avail- in the greatest abundance near to the mines of this region (see 1).

n apparatus for destructive distillation, capable of treating 30 lb. ush at a charge, was constructed, and two charges of the brush were led (see Fig. 2). The products which first came over consisted d liquor resembling the crude pyroligneous acid obtained from wood lation, a black oil or tar, and inflammable gas. Finally these prod- ceased to come over, but, upon raising the temperature, a consider-




. 2.—Experimental Apparatus Used for the Destructive Distillation e Brush. A, burners; B, retort; C, cold junction; D, tar tap; E, condenser; vanometer; G, gas; H, distillate.

mount of gas was given off and a rather thick reddish-brown liquor, g an alkaline reaction, began to come over, and with it was a small nt of tar similar to that which came over with the acid liquor. The n liquor had the characteristic fishy odor peculiar to the amines. was also at times a distinct ammoniacal odor. The acid and ne liquors were kept separate, while the tar from both was com- . These three products were first tried qualitatively in the flotation ely ground samples of various minerals such as galena, cinnabar, e, etc.

ne acid liquor behaved very much as does ordinary pyroligneous The alkaline liquor was a good frothing agent, but the froth

[3] *Encyclopædia Britannica,* Eleventh Edition, vol. 27, p. 634.

carried up little mineral. The tar which came over with the acid liquor proved to be a splendid flotative agent.

Following this, quantitative tests were made upon a number of different ores, employing the tar produced from this lot of sage as a flotative agent. These results, a number of which are given below, were in general very satisfactory.

A sample of zinc ore from the Butte Superior mine, Butte, Mont., containing 22.39 per cent. of zinc, when tested in a Janney laboratory machine employing a solution of 0.25 per cent. of sulphuric acid, gave 97.0 per cent. extraction of the zinc. The first concentrate contained 53.8 per cent. of zinc, the second 48.9 per cent., the third 40.5 per cent., and the fourth 18.1 per cent. The oil consumption was at the rate of 0.4 lb. per ton of ore.

A sample of mercury ore from the New Almaden mine, California, containing 0.26 per cent. of mercury, when tested in a Janney machine employing a 0.2 per cent. solution of sodium carbonate, gave an extraction of 90 per cent. of the mercury. The first concentrate contained 3.6 per cent. of mercury, the second 2.5 per cent., the third 1.55 per cent., and the fourth 0.95 per cent. The oil consumption was at the rate of 1 lb. per ton of ore.

A sample of lead ore from the Coeur d'Alene region, containing 12 per cent. of lead, when tested in a Janney machine employing a 0.05 per cent. solution of sodium carbonate, gave an extraction of 92.2 per cent. of the lead. The calculated lead content of the total concentrate was 37.3 per cent. The oil consumption was at the rate of 0.67 lb. per ton of ore.

A sample of silver-gold ore from the Ophir mine, Virginia City, Nev., assaying 0.46 oz. gold and 7.4 oz. silver per ton, when tested in a Janney machine employing a 0.1 per cent. lime or sodium carbonate solution, gave an extraction of approximately 90 per cent. of the silver and 95 per cent. of the gold. The first concentrate assayed gold 5.3 oz. and silver 198 oz. per ton; the second, gold 3.75 oz. and silver 72.9 oz. per ton; and the third, gold 1.32 oz. and silver 35.3 oz. per ton. The oil consumption was at the rate of approximately 0.6 lb. per ton.

The oil consumption when employing sage tar appears to be less than with most of the other oils experimented with in treating the same ores, and the extraction was in general better. Since it is the experience of many that large-scale operation requires less oil than is indicated by small-scale tests, it is reasonable to suppose that the oil consumption in treating the ores cited would be materially lessened when working upon an operating scale, and that possibly the extraction could be bettered. I think the latter is particularly true of the Ophir ore.

Later in March, another lot of brush of the same variety was collected by Mr. Lantz from the same locality, more fully in leaf than that collected

**earlier in the season. Distillations were run on this lot, keeping the various products separate, so that they could be properly measured and weighed.**

**Temperature measurements were made at regular intervals by means of a thermocouple, and as a result there was better control of the heating than in former tests. The yield from the last of these tests, which was the most carefully conducted, and was therefore the most representative, was as follows:**

| | Pounds | Per Cent. |
|---|---|---|
| Weight of sage brush used | 30. | |
| Acid liquor | 8.4 | 28.00 |
| Tar with acid liquor | 1.1 | 3.66 |
| Alkaline liquor | 0.66 | 2.20 |
| Tar with alkaline liquor | 0.09 | 0.30 |
| Charcoal | 10.00 | 33.33 |
| Gas (by difference) | 9.7 | 32.51 |

The retort was slowly heated for 1 hr. before liquid began to condense. The temperature at the center of the charge at the end of this time, as indicated by the thermocouple, was 60° C.; the temperature at the sides of the charge was probably somewhat higher.

In the next period of 6 hr., during which the acid liquor and most of the tar came over, the temperature rose from 60° to 275° C.; the rise above 100° C. taking place during the last hour and a half.

The alkaline liquor and the last of the tar came over in the last period of 3 hr., during which the temperature rose from 275° to 611° C.

It is reasonable to assume that a yield of about 4 per cent. of the tar oil can be realized if the sage is collected at the proper season and the distillation carried on by the best methods. Then there is the acid liquor, which in certain cases could be used directly in flotation; or it might prove profitable to recover the alcohol, acetic acid, the dissolved tar oil, etc., which it contains. I have not had an opportunity to investigate the alkaline liquor thoroughly, but it would seem to present many interesting possibilities. Among other things, the first lot was found to contain 2.09 per cent., and the second lot 2.18 per cent. of nitrogen.[4] I suspect that this liquor may also contain phenolic bodies which would be useful in flotation when separated from the other constituents. The charcoal is fine, but it should be possible to utilize it for fuel in heating the retorts. One peculiarity is the high percentage of ash which it contains (10.5 per cent. in the one sample analyzed). This may be in part due to dust upon the sage brush, although the brush was chopped fine before distillation, and it would seem that a good deal of the dust would be shaken from it during this operation. If the charcoal were burned, as previously suggested, the alkaline ash might serve instead of lime or sodium carbonate in cases where flotation in alkaline solution was practiced.

[4] Analysis by Professor R. E. Swain.

The inflammable gas would, of course, be burned under the retorts. The proportion of the heat necessary for carrying on the operation which could be realized in this way is problematical; however, it is reasonable to assume that a considerable part of that required for destructive distillation could be produced by burning the gas and charcoal. Moreover, another interesting possibility in connection with the heating of the retorts is the utilization of the waste heat from various metallurgical operations.

Through the courtesy of Charles Scalione, several ounces of steam-distilled essential oil was prepared from the tip ends of a portion of the last lot of sage brush. Approximately 1 hr. was required for distilling each charge of 20 lb. The yield amounted to 0.43 per cent. of the whole plant. This oil had a greenish-yellow color when first distilled, but became yellow upon standing. It has the characteristic penetrating odor of sage, and a very decided tendency to creep up the sides of the glass containing vessel. This oil appears to have distinctly different properties from the essential oil resulting from the steam distillation of the black sage. If gives promising results with some ores, but in my opinion it is not nearly so good as the tar oil resulting from the destructive distillation of the same shrub. In fact, under ordinary circumstances, I do not think that the steam-distilled oil can be given serious consideration, since the mines that would be most benefited are generally located where fuel is high, and the yield of oil is rather small, probably less than 1 per cent. In addition, there are no other valuable products, unless the tannin extract resulting during steam distillation should have a market value or a demand should arise for "sage tea."[5]

Time has not been available for making analyses and a study of all the products resulting from the destructive distillation of sage. However, sufficient work has been done to show at least that the light tar is an efficient flotative agent for a considerable number of ores, as indicated by the tests cited.

The idea of utilizing sage brush in metallurgy is by no means new, as is shown by the following quotation,[6] referring to early development of the Washœ process, and the wild riot of experimentation accompanying it:

"The native sage brush, which everywhere covered the hills, being the bitterest, most unsavory, and nauseating shrub to be found in any part of the world, it was not long before a genius in charge of a mill conceived the idea of making a tea of this and putting it into his pans. Soon, the wonders performed by the sage-brush process, as it was called, were being heralded through the land."

[5] Patents are pending covering the use of the various sage products in flotation.

[6] Dan De Quille (William Wright): *The Big Bonanza*, pp. 138, 140, American Publishing Co., Hartford, Conn., 1877.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Antecedent Mineral Discovery Requirement

BY E. D. GARDNER, M. E., MISSOULA, MONT.

(Arizona meeting, September, 1916)

APPARENTLY the widespread agitation for the codification of our mining laws has had its effect, and it is quite possible that Congress will take up the question during this present session. The greatest objection to our present statutes pertaining to metalliferous deposits seems to be directed against the law of the apex. Those who are not in favor of changing this law are very far in the minority and they apparently realize that it is useless to urge its retention.

Next in importance to the repeal of the law of the apex, seems to be a widespread advocacy for the repeal of the law requiring discovery of mineral before a valid location of a mining claim can be made. To a lesser extent there appears to be a desire to have the law changed so as to allow patent for claims without mineral discoveries.

The purpose of this article is to review the arguments in favor of the proposed change of the discovery requirement, to show some of the evils that may be expected to result therefrom, and to consider whether a system may be devised to correct the undesirable features of the present law in this respect and at the same time not expose the mining fraternity and the general public to the evils which would result from the repeal of this fundamental requirement.

Under the law as at present framed, the validity of a mining location is dependent upon the fact of discovery of mineral within the boundaries of the land claimed.

> "* * * but no location of a mining claim shall be made until the discovery of the vein or lode within the limits of the claim located" (Sec. 2,320, U. S. Rev. Stat.).

All authorities agree that discovery is the source of the miners' title. Lindley, in his valuable work on Mines (3d Ed., Sec. 335), lays down the rule in part as follows:

> "Discovery is the initial fact. Without that no right can be acquired. * * * Such discovery must precede the location, or be in advance of intervening rights. The proof of recording and marking a claim will not authorize the court to presume a discovery."

As is well stated by the U. S. Supreme Court in Erhard v. Boaro (113 U. S. 536):

"A mere posting of a notice on a ridge of rocks cropping out of the earth or on other ground, that the poster has located thereon a mining claim, without any discovery or knowledge on his part of the existence of metal there, or in its immediate vicinity, would be justly treated as a mere speculative proceeding, and would not of itself initiate any right. There must be something beyond a mere guess on the part of the miner to authorize him to make a location which will exclude others from the ground, such as the discovery of the presence of the precious minerals in it, or in such proximity to it as to justify a reasonable belief in their existence."

Accordingly, if the miner has made a discovery and otherwise conformed to the requirements of the law, his possession of the claim located is exclusive, against all the world, even, in all probability, against the owner of the land, the Government, except on areas which Congress by special legislation has set apart and defined for paramount purposes, such as National Parks. He has an interest in the land which descends to his heirs, which he can dispose of by will, by deed, or other contract, and all that is required of him after location, until patent is secured from the Government, is to keep his claim alive by performance of the annual assessment work thereon.

In cases where two private locators claim the same ground, the one first making a discovery of mineral would probably get the area, irrespective of the relative time of beginning work or of posting notices. However, such cases are extremely rare, considering the total number of claims located each year. Strictly speaking, a prospector cannot make a valid location before he has discovered mineral, but he has the rights of possession, except against the United States; and, while there is a conflict in the decisions on this point, in many cases the rule appears to be that as long as he is working the ground and is in possession the law will protect him, and when a discovery is made his location is good.

In any attempt to remedy defects in the present system of Federal mining laws, the underlying intent and purpose of existing legislation on the subject should not be lost from view. As was well said by the Assistant Secretary of the Interior in Cataract Gold Mining Co. *et al.* (43 L. D., 248):

"The intent of the general mining laws was to encourage and promote the development of the mining resources of the United States."

Proposed amendments of existing law, not based on this broad policy but having in view the exploitation of the public domain in the interests of mere individuals without corresponding benefit to the people as a whole, should meet the condemnation of every honest and right-thinking man. Keeping in view, therefore, this beneficent purpose of the present law, let us consider whether the proposed elimination of the antecedent-discovery requirement from the law will operate to give greater effect to such purpose.

It is contended that, if the requirement of an antecedent discovery

of mineral within the lines of the claim located were dispensed with, the result would be to hasten the development of the mineral resources of the public domain and so increase the wealth of the nation; furthermore, that the present discovery requirement is an unnecessary hardship in a great many cases, and should be dispensed with in justice to those who are honestly endeavoring to develop the mineral resources of the public domain.

It is urged that, if the antecedent-discovery requirement were abolished, the one first staking out a piece of ground would be protected in his possession of it even if a later locator on the same ground were the first to discover mineral.

This raises the question, to whom should the desired reward (viz., exclusive occupation of the ground) be given? Shall it be to him who is diligent in staking out his claim, or to him who is not merely diligent in that respect but, not content with this, pursues diligence to the point of discovering the hidden mineral wealth? Which of the two is more deserving of reward? Which of the two is bending his efforts toward the consummation of the policy of the law? Certainly the one discovering mineral has performed a very essential act in the development of a mine.

In some mining districts where outcrops are few, extensive and expensive development work is necessary before valuable mineral is found. Many long and costly exploratory tunnels have been driven and shafts sunk to develop tracts of land on which occur no surface indications of veins or mineral but which were favorably situated in mineral belts. Justification for prosecution of development work under such conditions depends to a great extent on a careful study of mineral showing on adjacent lands. The successful prosecution of such exploratory work is dependent on the raising of sufficient capital for the enterprise. The raising of capital, a difficult undertaking in connection with almost any project so essentially speculative in character as that of exploring for precious metals, is made more difficult if the promoters cannot show a fee-simple title to the property to be developed. Hence, the abolishment of the discovery requirement would have the effect of overcoming this one obstacle to the development of our mineral resources.

When valuable orebodies are found on unpatented claims, there is always the menace of litigation to prove the ownership of the ground. After a strike has been made, it often happens that abandoned locations, which at one time covered the land, are resurrected or the boundary lines of prior adjoining claims are moved in such a manner as to take in the more valuable ground. The indefinite manner of describing claims in location notices, and the great number of old illegible mining corners usually scattered over the landscape in mining districts, increase the difficulty of successfully defeating such contests. The richer the ore-

bodies found in these cases, the stronger the efforts by contesting claimants to obtain the ground.

The danger of contests on unpatented ground after a strike has been made will be augmented if valid locations can be made without a discovery of mineral. Contests initiated by the locators of abandoned prior claims would be more likely to succeed if these claims had had a legal existence.

It may be well to say here, however, that it must not be taken for granted that all contests are not made in good faith. Locators of adjoining ground quite often do not know the position of each other's lines, and conflicts occur, therefore, when mineral survey is made for one or both of the claims. A contestant is usually honest in his belief in his ownership of the disputed area. Private contests against the issuance of patent for mining claims are being continually initiated by claimants for all or part of the ground.

Sometimes a locator tries to get part of another's prior location by getting his conflicting claim patented. I know of instances where valuable ground which was being held under location has been patented by a subsequent locator before the owner of the prior claim knew what was going on.

If we shut our eyes to actual conditions and assumed that all locators of mining claims act in perfect good faith in staking out ground with intent to develop it for its mineral contents, probably there would be no necessity for the antecedent-discovery requirement. But mining locations, even now, are made to cover a multitude of frauds against the public-land laws. Title to valuable water-power sites, immense quantities of timber and valuable town sites have been secured under the mining laws. How much greater and more frequent would be these frauds if even a perfunctory discovery of mineral were not required to validate the locations. By far the greater number of mining claims desired for purposes other than mining have had no discovery of mineral.

Care should be taken that the interests of all the people are not sacrificed in our consideration of the welfare of the prospectors. The discovery requirement is designed, and operates, as a check on the disposal of lands of the people under the mining laws for purposes foreign to the intent of the law, viz., the speedy and bona fide development of the mineral resources of the public domain.

If the validity of the location is to be dependent merely on taking possession of the ground without the antecedent discovery of mineral, what is to prevent the staking of claims near pay ground with intent to hold them for speculation and hold-up purposes, and not for bona fide development of mineral value, to a much greater extent than at present? What is to prevent irresponsible individuals from staking claims which can be held indefinitely by perfunctory assessment work, thereby tying up large areas of valuable ground for the purpose of levying tribute on

those desiring to obtain the ground for mining or other purposes? Surely, to abolish the antecedent-discovery requirement is to open the door to fraud and deceit incalculable, and will tend to block actual development of large areas for many years to come.

The abandonment of the mineral-discovery requirement would naturally facilitate the patenting of mining claims; but, however desirable it may be in some cases to expedite the granting of patents, I believe easy patenting, on the whole, will not benefit the mining industry, but will retard it.

Most mining men know of many promising claims and prospects for which patents have been obtained which have been, to all intents and purposes, abandoned. A large proportion of the applications for patent comes from owners who are tired of doing the annual assessment work but still desire to hold the ground. Of such cases, only a small number of claims are ever worked again.

Nearly 10,000 mineral surveys for groups of claims, from one to thirty or more each, have been made in the State of Montana alone, and this number does not include placer claims that have been taken up by legal subdivisions. While patent does not necessarily follow a mineral survey, it does so in nearly all cases, and of the patented claims only a small number have been worked since final certificates were issued. Outside of the Butte district, I will venture to say that less than 5 per cent. of the patented claims in Montana are now being worked, when metals are higher than at any time in the last 50 years.

In such active districts as Butte and the Coeur d'Alenes, where large capital is required in developing properties, the patenting of claims, on the whole, does not discourage new operations, but perhaps helps them. In cases, however, where one dominant company patents the whole surrounding territory, independent operators who would perhaps be willing to take a chance on developing some of this ground, if open, are thereby kept out.

The patenting of the ground seriously handicaps the chances of small camps and new districts for becoming important producers. Numerous cases can be pointed out where the patenting of the claims has seriously retarded the development of camps, or altogether stopped it.

The majority of those who have written about the desirability of amending the present mining laws have done so from the standpoint of the mining operator or prospector. None of the writers on the subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some

people would appear to believe. Individual claims on it are allowed only by virtue of the laws of Congress, which prescribe the performance of certain definite requirements, and it does not belong to the first comer solely on his assertion of a claim to any part of it.

Many abuses have been perpetrated by unscrupulous locators of land under the mining laws. Mining claims have been, and are now, located to control springs, water-holes, range privileges, power and reservoir sites, town sites, rights-of-way, summer-residence sites, timber and agricultural land, natural curiosities, and other surface values. While, at the present time, land more valuable for other purposes than for mining cannot be patented under the mining laws unless mineral has been found and the required expenditure made, in the past, before the Government inspected the ground before issuing patent, valuable areas were patented when the ground was in no respect mineral. At the present time, in some parts of the country, public business is seriously interfered with by unscrupulous locators of ground under the mining laws. While the Government is now embarrassed by these hold-up locations (the mining industry at large has no idea how great the number is), the main expense is the examination of such claims and the delay caused by them. If, by amending the present laws, valid locations can be made anywhere on Government land without mineral discovery, no Government or other activity on public land will be safe from extortion and blackmail. Conditions are bad enough now. For instance, in some localities, as soon as a body of Government timber is advertised for sale, the timbered area and ground over which the logs are to be moved to market is promptly located as mining claims, or old claims are revived, and heavy demands made for the use of the surface of the ground. If, by law, claims of this sort can be made valid, it will give unscrupulous individuals a monopoly of the surface and prevent legitimate business over large areas. The owner of a single valid mining claim crossing a right-of-way of a logging road could exact a sum, for the privilege of crossing his ground, large enough to take all the profit from the logging operations. As the cost of operation is directly related to what an operator can pay for the timber, the Government has a direct interest in all such cases. It has been repeatedly held by the courts that a valid location holds against the Government, even if abandoned for years. I know of areas from which the Government was contemplating selling the timber and on which there were as high as 100 abandoned mining locations. In such cases, if these locations had been valid without a mineral discovery, the locators on reasserting their rights to the ground as soon as they found the timber was valuable could have kept the Government from selling it. There is hardly a stream course or canyon in many parts of the West that has not been at some time plastered with locations.

Even if valid locations could be made without a mineral discovery

only on lands classified as mineral, it would still affect the Government, for there are bodies of white pine, which are advertised for sale from time to time, on lands in National Forests which have been classified as mineral but not covered by mining locations.

It has been said that one must actually make a discovery of mineral on land within National Forests before he is allowed possession or allowed to prospect it. Such is not the case, however. A prospector or miner is in no way disturbed by Government agents in the enjoyment of his rights within National Forests. No examination is made of any ground located for mining purposes unless it interferes with the administration of the Forest. A claim may interfere with the administration of the Forest when it conflicts with areas occupied by individuals under what are known as Special Use Permits; when it is included within a tract from which the Government has sold the timber; or where the claim is so located as to control rights of way over which it is necessary to transport Forest products. Unless the claims are actually interfering with the administration of the Forests, no examinations are undertaken until applications for patent are made. The proportion of mining claims examined prior to application for patent is very small, probably less than 1 per cent. of the total number located. There are thousands of mineral locations within the National Forests of which no examination has been made by the Forest Service, and in all probability never will be except in cases where application for patent is made.

Mining locations within the National Forests which, after an examination by a competent man, are shown to be clearly invalid, may be disregarded as far as the surface is concerned, but no action is taken to dispossess the locator of the ground and no objection whatever is made to his development of the mineral possibilities. In fact, the Assistant Secretary of the Interior has decided (Nichols-Smith case, unreported) that the Department has no jurisdiction over locations and has authority to cancel a claim only after an application for patent has been made. This decision, however, is now before the Secretary for consideration on motion for the exercise by him of his supervisory authority.

The fact that the Government has placed a large part of its domain within the National Forests shows that this land is valuable to the Government for the benefit of the people at large. This fact alone is an argument from the people's standpoint that the requirements for patenting Government land by individuals, at least within the Forests, should not be made easier.

I believe that, while sometimes the law requiring a discovery before a valid location can be made has worked hardships on prospectors, the wrong that could be worked under a law not requiring a discovery would be far greater.

Under the general mining laws, lands are not required to be chiefly

valuable for mineral before they can be entered. The Supreme Court of the United States has held in U. S. *vs.* Iron Silver Mining Co. (128 U. S. 673) that the fact that land may incidentally possess advantages other than its valuable mineral deposits will not preclude its disposition under the mining laws.

An act of Congress of Aug. 4, 1892 (24 Stat., 348) provided:

> "That any person authorized to enter lands under the mining laws of the United States may enter lands that are *chiefly* valuable for building stone under the provisions of the law in relation to placer mineral claims."

The point of difference between this act and the general mining law applicable to mineral deposits is that the Act of 1892 requires the land to be chiefly valuable for building stone.

Power or reservoir sites situated on land containing no evidence of mineral deposits have been entered as stone placers, but the applications for patent have been rejected by the General Land Office. If valid mineral locations could be made on such lands without a mineral discovery, the land could be patented as mining claims.

If patent were allowed for Government ground as mining claims without a mineral discovery there would be very few, if any, power projects free for development at the present time, for the ground would be validly held or patented as mining claims. Most of the undeveloped available power sites on public lands have been included in power withdrawals or National Forests. The ground is, therefore, not subject to entry except as mineral locations, and the only way to get a patent to such ground is under the mining laws. Nearly all power projects in the Northwest, where there is any possibility of early development, are on ground covered by mining locations. In some cases the mining claims are valid under the present law, but the majority have no mineral discovery. If the law were changed concerning mineral discoveries, there would be little new development of hydro-electric power without paying tribute to the kind of prospectors who had the foresight to locate mining claims covering the ground. Also Federal control, due to an interest in the site, could be circumvented by first obtaining patent as mining claims to all the land affected. In the Northwest there is a power project capable of developing 20,500 hp. One-half of the land affected by the project is within a National Forest, and therefore the Government under the present procedure exercises control over the development of the site. The power would be developed under a Special Use Permit issued by the Government, which requires certain conditions to be complied with. All of the public land has been located as placer mining claims, and if the claims were valid the Government under the present law would have no control whatever over the development of the power. In 1913, a permit was issued to a company to develop the power, but for some reason it was not

able to go ahead with the project. This company, however, acquired the mining locations, and if the claims could be patented the owners could prevent anyone else from developing the power site. Flour gold is found in gravel on all of these claims, but nowhere in paying quantities.

Another project with which I am familiar is capable of developing 331,000 hp. by using storage, or 178,000 hp. by utilizing only the minimum flow of the river. The lower 12 miles of this project, which is capable of developing 260,000 hp. by storage, or 140,000 hp. without, is entirely covered by mining locations. The area is within a mining district and would probably be classified as mineral, but only a few of the locations have mineral discoveries. One side of the river is within a power withdrawal, and the other within a National Forest. In this case, as in the other, a permit was granted to develop the project, but it has not been done. The permittee has acquired the mining claims and he controls the site in so far as his mining claims are valid. If all these mining claims were valid without mineral discoveries, they could be patented and the Government would lose control of the project.

On March 1, 1906, Windfield Doern and two others located the Eagle placer mining claims, alleged to be chiefly valuable for building stone, on the Stanislaus National Forest in California (41 L.D. 655–659). On Oct. 25, 1907, the claim was conveyed to the Stanislaus Electric Power Co., who filed application for patent on April 5, 1908. The entry was contested by the Government. Stone was used from the claim in the construction of a dam on the claim, from which water was conveyed to a plant 15 miles below, where power was generated. It was proposed to build a second plant on the claim and bring water in a conduit from above. The patent was denied by the General Land Office, as it was shown at a hearing that the stone had no value outside of the construction of the dam, and for that purpose had no special value. The land, however, had value as a power site.

On Jan. 2, 1907, H. V. Gates made application for patent under Act of Aug. 4, 1892, for the Excelsior placer, area 140 acres, in northern California. The patent was protested by the Northern California Power Co. At a hearing, it was disclosed that the ground contained a deposit of basalt that had only a local use for building stone, such as the construction of a power dam, and was not as suitable for that purpose as other common rock. The claim covered a reservoir and tunnel site, and the claimant had a permit from the Forest Service to use a part of the claim for power purposes. The General Land Office held that the claim was more valuable for power than for building stone, and the entry was canceled.

On a river in a western State is situated a power site capable of developing 2,490 hp. The dam site is covered by an unpatented lode claim, and above the falls are situated four patented lode claims along the river in

a row. Each of the four patented claims contains a valid mineral discovery, but no ore. Before proceeding to patent, the claimants received $5,000 from a power company for the privilege of flooding their ground.

Perhaps more fradulent mining locations have been made to acquire valuable timber than for any other purpose on non-mineral land. If the mineral-discovery requirement is repealed, large timber steals which have been prevented in the past will be successful. Many applications for timbered mining claims have been canceled after the Government has shown that no mineral discoveries have been made. It would not be possible to defeat such cases if the proposed change of the law is made.

Timber on patented mining claims can be disposed of by the owner as he sees fit. Timber on valid unpatented mining claims on proven mineral land outside of National Forests can be cut for certain described purposes as approved by Congress by Act of June 3, 1878 (20 Stat., 88). Timber, however, on an unpatented mining claim within the National Forests cannot be cut for any purpose other than the development of the claim, except by Special Use Permit or purchase from the Government.

It has been a common practice in some localities to locate mining claims on non-mineral timbered areas to give color of title to the ground while cutting the timber. Numerous suits have been instituted by the Government to recover the value of timber cut from such invalid claims.

In the case of Anderson *vs.* United States (152 Fed., 89 to 91), Anderson, Baker and Sandlin located as placer claims 1,200 acres of non-mineral Government land in the Boise land district, Idaho, from which was cut timber worth $17,751.79. Suit was brought by the Government to recover the value of the timber, and it was successful.

In the case of Powers *vs.* United States (119 Fed., 562), the Government brought suit to recover the value of 668,000 ft. b. m. of timber, worth $7,241, which was cut from non-mineral Government ground. The ground was covered by mining locations made by a third party.

About 10 years ago, a company applied for patent for 3,636.99 acres of Government land as placer claims. The magnitude of the case caused examination to be made by the General Land Office, and it was found that on one claim there was a small worked-out placer deposit of gold. There was no possibility of gold occurring in paying quantities on the rest of the group, although fine colors could be found almost anywhere in the country. This land contained an average stand of 15,000 ft. b. m. of white pine and other varieties to the acre, or about a total of 60,000,000 ft. b. m. on the group. The claims were continuous, but excluded and almost surrounded a small area from which the timber had been burned. The entry was protested by the Government and at a hearing held to determine the validity of the group, the claimants defaulted, thereby admitting the truth of the charges. The entry was finally canceled on Jan. 28, 1910. At this time, unpatented fraudulent placer claims were

being held in this district which contained a stand of between 200 and 300 million feet of timber. During the summer of 1910, forest fires that destroyed the timber swept over this country, after which all of the timber-mining locations were abandoned.

In 1913, application was made for two groups of claims in Idaho. No mineral had been found on four of the claims of the group which contained a heavy stand of white pine worth $3,800. After a hearing, the entry was held for cancellation by the Commissioner of the General Land Office.

In 1912, application was made for two claims in the Coeur d'Alene district, Idaho. This entry was also held for cancellation by the Commissioner of the General Land Office, after a hearing. These claims were completely surrounded by patented placer claims, but were situated on a small mountain. The ground was covered by a stand of white pine having a stumpage value of $3,478. One of the claimants was interested in a saw mill situated near the claims.

Along in 1906, a mining company (long since defunct) did considerable development work on lode locations in western Montana, and located a large number of claims. The company failed to find any ore, and went out of existence. The first year after the company abandoned the ground, it was located by a man who did no more work on it. He relocated some of the claims for several years, but when no one else showed any desire to obtain the ground, he also abandoned it. In 1914, a lumber company secured timber on the river above these claims. After the lumber company cleared its right-of-way, the mineral locator came along, saw the situation and immediately reasserted his right to some of the ground that controlled the right-of-way. He demanded an excessive sum for the privilege of crossing his ground. His demands were refused and logging operations shut down for about a year. An examination of the mining claims was made by a mining engineer and it was found that the claims were invalid because no mineral had been discovered upon them. After a year's delay, the company constructed the logging road and commenced operations without any further trouble and without paying tribute to the locator. If these locations were valid without mineral discoveries, the logging operations could have been held up indefinitely unless the excessive demands of the locator were met. The claims are in a general mineral belt.

If valid locations of lode claims can be made in the future without a mineral discovery, many town sites are going to be held as mining claims. The following case is now before the Land Department for settlement. In 1909, a lode claim was located in such a manner as to take in all of the level portion of the canyon at the division point of a new railroad in Idaho. On June 27, 1913, application for patent was made. The section in which the claim is situated has been classified by the United

States Geological Survey as non-mineral, and the land is in the Northern Pacific grant. The railroad has a station and some dwelling houses on the ground and part of the claim is also claimed by the heirs of a homesteader. If the mineral locator can prove a valid discovery on the claim, he is entitled to the ground. The railroad began operating about the time the claim was located. Practically a whole town, with a population of several hundred, is on the mining claim, and there is no doubt that the ground is more valuable for town-site purposes than for anything else. A large diabase dike, in which can be found traces of copper, runs through the claim, but there is no indication that paying ore will ever be found upon it. Sufficient development work has been performed. If no discovery of mineral is going to be required, many similar cases may be expected in the future.

The repeal of the mineral-discovery requirement would open the doors for blackmail of all irrigation companies situated like the one in the following case, even when there is not a trace of mineral on the land. In 1910, an application for patent was made for a placer situated in a small park in western Montana. Previous to the mineral application, an application was made to the Secretary of the Interior for the use of the park for a storage-reservoir site, for which purpose the ground is well adapted. It was proposed to impound water during the winter months and high-water periods, to be used for reclaiming about 3,000 acres of arid land. The reservoir permit was approved by the Secretary, but construction has not been begun on account of the placer claim. The Commissioner of the General Land Office decided that sufficient mineral to constitute a discovery had not been made upon the claim and it is now before the Secretary of the Interior on appeal. The park is an old lake bottom and contains sand, clay and gravel to an undetermined depth. This material all contains minute quantities of gold, but in no place, as far as developed, in paying quantities. No serious effort has been made to mine the gravel, and for the last 20 years the ground has been used by the claimant as a ranch. If it is finally decided that the claim contains sufficient mineral for a discovery, the mineral claimant will be legally entitled to the land, and, before the irrigation system can be commenced, the land will either have to be obtained from him by purchase or by condemnation proceedings.

Throughout the country, attempts have been made to obtain possession of numerous springs, mineral or otherwise, by taking up the ground as mining claims. On Aug. 26, 1886, the Margaret Mining Co. made application for the Gray Eagle Lode mining claim (11 L. D., p. 563) in King County, Washington. This claim embraced some springs known as the Hot Springs of Green River which were supposed to have great medicinal value. The entry was canceled by the Land Office because it was shown at a hearing that the land had no value for mineral but was

used for a health resort, and that the mining company was formed to acquire title to the Hot Springs.

By the Act of Jan. 31, 1901 (31 Stat., 745) Congress provided that "all unoccupied public lands of the United States containing salt springs or deposits of salt in any form and chiefly valuable therefor are hereby declared to be subject to location and purchase under the provisions of the law relating to placer mining claims." Under this Act many mineral springs having no value whatever for salt, but having supposedly medicinal properties, have been located as placer claims.

On March 20, 1905, Henry Lovely made application for patent for the Lovely placer claim in the Juneau land district in Alaska. It was shown at a hearing that the ground was used as a health resort and water from two springs was conducted to bath houses, the cost of this being applied as mining improvements. It was also shown that the ground was not mineral or saline in character. The claim was canceled Feb. 13, 1907, by the General Land Office (25 L. D., 426).

By decision of April 14, 1913, the Secretary of the Interior held that the Salt Creek placer claims were invalid (21 L. D., 745). These claims were located in February, 1911, under the Act of Jan. 31, 1901, so as to cover a group of hot springs whose waters carried in solution common salt, Epsom salts and Glaubers salts and other chemical substances. As compared to sea water, the waters of these springs were shown to contain about one-eighth as much common salt. It was held by the Secretary of the Interior, from facts determined at a hearing, that the ground had no value for the manufacture of common salt, but possessed distinct curative properties.

In central Montana is situated a mineral spring known as the Appollonaris. The ground at the small side gulch on which the spring is situated was located as a stone placer and the adjoining ground on the main creek as gold placers. No discovery of gold had been made on the gold placers, and the limestone on the stone placer was not valuable on account of its poor quality and the distance from any market or railroad. This spring has a local reputation for medicinal properties and Special Use Permits for erecting bottling works and hotel buildings on the area was desired by several people. The development of the spring was held up for 5 years, but finally the men holding the mining claims decided that they could not patent the ground and abandoned it. The spring is now being developed.

The abrogation of the mineral-discovery requirement would greatly facilitate such fraudulent attempts to obtain ground desired for purposes other than mining.

At numerous points desirable for summer residences on the shore of Lake Pend Oreille, mining locations have been posted, and in many cases there has been a pretense of doing the annual assessment work. In a

few cases, application for patent has been made. On some of these claims sufficient mineral has been found to constitute a mineral discovery, but in the majority of cases no mineral has been found.

The Frances Placer, consisting of 20 acres on a bar on the Clearwater River in central Idaho, was applied for under the mining laws. The claimant had farmed the land for at least 10 years previous to making application. The mineral entry was contested and canceled in 1912, and the land was later taken as a homestead.

Mining claims have been patented, and others are being held as locations, on favorably located scenic points along the rim of the Grand Canyon of the Colorado. Two lode claims also have been patented which cross the Bright Angel Trail. Tourists who have gone down this trail and paid their dollar each to the owner of the lode claims can realize his farsightedness in finding his mine just in this particular location. It is of interest to note that no mineral commercially valuable has been found on any of these claims.

On the public domain there are numerous conflicts of mineral claims with homesteads. In all contests, the burden of proof is on the homesteader, as he has to prove the land non-mineral in character. If valid locations can be made without the necessity of a mineral discovery, the homesteader's lot in mountain valleys is going to be still harder to bear, and the good Lord knoweth it is bad enough now.

If the mineral-discovery requirements for the location of mining claims is abrogated, no form of activity can be safely undertaken on unpatented land, and it is difficult to anticipate the many abuses that can be perpetrated.

The uncertainty of ultimate possession of unpatented land in mineral belts could be remedied, I believe, by making provision by law for clearing title to the ground against all except the United States at any time, as is now done at the time of application for patent. If, after development, mineral is found, the ground could then be patented.

Perhaps the method now used in connection with homesteads on the public domain could be adapted to the making of mineral entries. In the case of homesteads on surveyed land, an applicant, at the time of going on the land, makes a filing in the U. S. Land Office, corresponding to the posting of a location notice on a mining claim, and, if no adverse claims are proven and the land is non-mineral, his entry is allowed. This excludes all other private claims to the ground, and, after he has complied with certain prescribed regulations, patent is issued to the homesteader.

If all men were honest, or this country were a Utopia, it would be perfectly feasible to repeal the law requiring the discovery of mineral, but, unfortunately, such is not the case. Under the present law some operators may have been unjustly obliged to defend their rights, but it has been found that laws made to restrain the unjust usually cause suffer-

ing to some of the just. The repeal of this law would afford relief to some who are acting in good faith, but at the same time it would let down the bars, allowing an unlimited amount of fraud, and fostering blackmail. That the wrong application, if possible, would be made of any new mining laws, should be expected from the many frauds that have been perpetrated under the present laws. By far the greater number of the mining claims that have been desired for purposes other than mining have had no discoveries of mineral. I believe that by suitable laws it is quite possible to ameliorate the one condition without aggravating the other.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Possibility of Deep Sand Oil and Gas in the Appalachian Geo-Syncline of West Virginia

BY DAVID B. REGER,* A. B., B. S., C. E., MORGANTOWN, W. VA.

(Arizona Meeting, September, 1916)

## *Introduction*

THE exhaustion of oil and gas in the United States is proceeding at a rapid pace. This is especially true in fields where the light oils that furnish the most fuel for internal-combustion engines are found, leading to a late estimate by Arnold that the petroleum resources are 36 per cent. exhausted.[1] The further remark by Dr. David T. Day at the New York meeting of the Institute, in February, 1916, that the operators of the country are now scarcely able to supply the ever-increasing demand for gasoline, leads the writer to believe that a determined effort will soon be made to secure deeper producing horizons in those regions that now furnish high-grade oil, and in which pump stations and pipe lines offer convenient means for the economical handling of new production. A study of Arnold's table shows further that the States that produce paraffin oil exclusively are 45 per cent. exhausted, this figure being much higher than the average of paraffin and asphalt States altogether. The most extensive of all the paraffin-oil-producing areas is the Appalachian Field, extending with the Appalachian Geo-Syncline from western New York southwestward through western Pennsylvania, western West Virginia, Ohio, Kentucky, and Tennessee, where oil of high gravity is always secured. Of these States, Pennsylvania and West Virginia rank highest in the quality of their product. In West Virginia, the oil sands dip to a lower level than in any other locality in the Appalachian Geo-Syncline, the Pittsburgh Coal of the Pennsylvanian being only 50 ft. above sea level along the Nineveh Syncline at Wileyville, Wetzel County.[2] The very significant fact that the sands of the Mississippian and Upper Devonian Measures furnished the richest oil pools of the State along this

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[1] Ralph Arnold: Petroleum Resources of the United States, *Economic Geology*, vol. 10, p. 710 (December, 1915).

[2] Ray V. Hennen: Marshall-Wetzel-Tyler Report, *West Virginia Geological Survey*, p. 66. 1909.

deep basin, most of them being non-water-bearing, should give pertinency to such information as may be obtained regarding deeper sands in this region, in which there may be a possible duplication of conditions in the more shallow sands.

### *Authority for Data*

Information has been secured primarily from the published reports of the Geological Surveys of West Virginia, Ohio, Pennsylvania, and Kentucky, supplemented by various special reports and maps of the United States Geological Survey. The southwestward extension of the Chestnut Ridge (Warfield, Campton) Anticline through Kentucky has been made on the authority of a recent publication by James H. Gardner.[3] Several important well records used have never been previously published, having been lately secured by the West Virginia Geological Survey. Due credit has been given in the table of well records to the operators who furnished the various logs, in all cases where this information is available. Special acknowledgment is made to Dr. I. C. White, State Geologist of West Virginia, for many valuable suggestions from his extensive knowledge of the region under discussion.

### *Description of the Appalachian Geo-Syncline*

The Appalachian Geo-Syncline, lying west of the Appalachian Mountain System, and being roughly parallel to it, extends in a south-southwestward direction from New York State through Pennsylvania, West Virginia, Kentucky, Tennessee, and Alabama. As partially shown on Plate 1 (in those regions where detailed structure maps are available) it has a course almost straight through Pennsylvania from Brookville, via Kittanning, Pittsburgh, and Washington, to the southwestern corner of the State where it enters West Virginia and extends southwestward across Wetzel, Tyler, and into Ritchie, where the Burning Springs Anticline cuts diagonally across it shifting the line of its axis nearly 20 miles farther west, to a point near Parkersburg. From this point its course is roughly southwest through Wood, Jackson, Mason, Cabell, and Wayne Counties, to the Kentucky State line 7 miles south of Catlettsburg. Its course through Kentucky is apparently interrupted by the Chestnut Ridge (Campton) Anticline which cuts across it, separating it from its southern extension.

### *Geology of the Appalachian Basin*

The surface geology along the axis of the Appalachian Basin is principally that of the Coal Measures, but at the eastern rim these rocks rise

[3] James H. Gardner: A Stratigraphic Disturbance through the Ohio Valley, Running from the Appalachian Plateau in Pennsylvania to the Ozark Mountains in Missouri. *Bulletin of the Geological Society of America*, vol. 26, pp. 477–483 (December 5).

rapidly toward the Allegheny and other mountain ranges of the Appalachian System until they disappear above the tops and the rocks of the

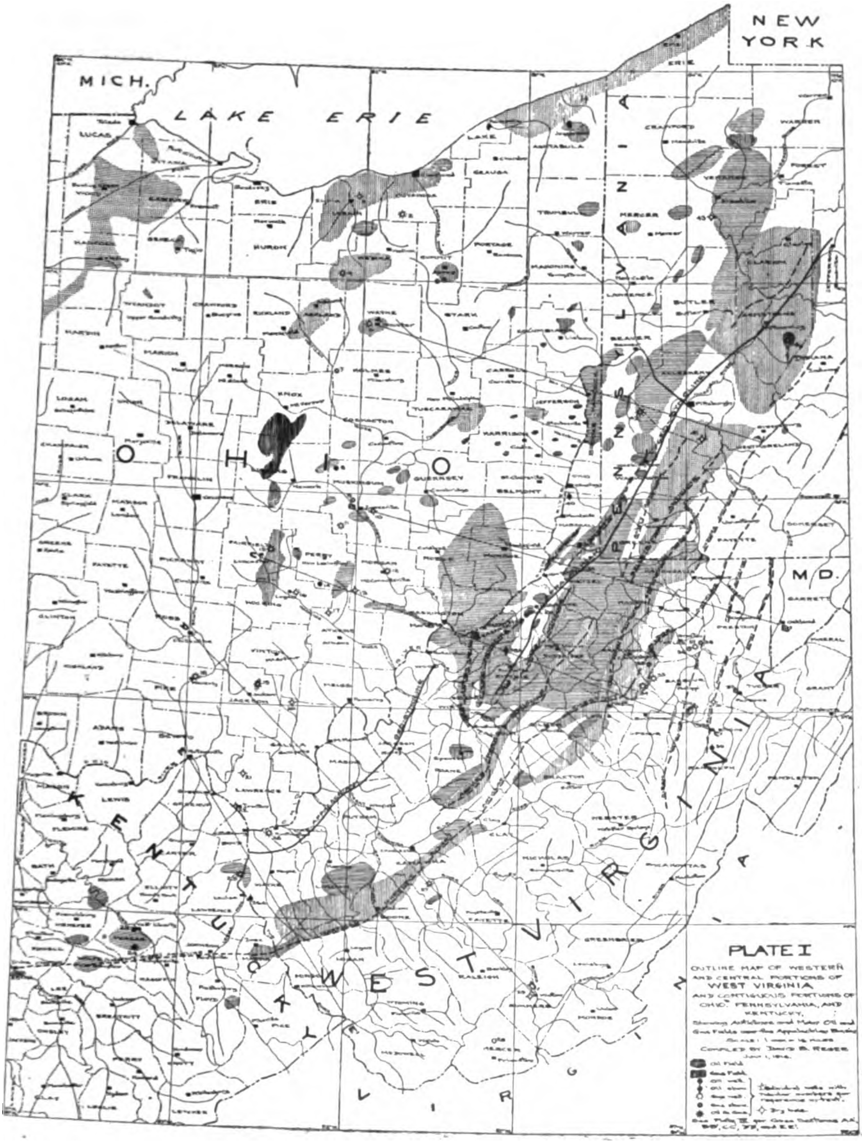



**PLATE. 1.—OUTLINE MAP SHOWING ANTICLINES AND MAIN OIL AND GAS FIELDS NEAR THE APPALACHIAN BASIN.**

Devonian, Silurian and lower formations come to the surface, the pitch in this region being abrupt and irregular. On the western side of the axis the measures rise gradually toward the Cincinnati Anticline, showing

none of the spectacular structural disturbances visible on the east. Northeastward from West Virginia, through Pennsylvania, the strata rise gradually along the axis of the basin, but the Pennsylvanian still forms the surface rocks at the New York State line, near Olean. Southwestward from West Virginia through Kentucky, the rocks rise along the axis, but very gradually, so that the Pennsylvanian still makes the surface deposits at the Tennessee line.

Plate 2 shows a columnar section made for the locality of Marion, Harrison, Wetzel, and western Monongahela Counties, West Virginia, where the rocks of the Appalachian Basin dip to their lowest level, and where the oil reservoirs of the State are the most prolific. In this region the surface rocks are principally the transitional Permo-Carboniferous type represented by the Dunkard Series. Throughout this region, numerous drillings are available that extend to the base of the Catskill Series, leaving no uncertainty as to the character of the sediments down to this level. Below the Catskill, the section is based on the deep wells drilled in Pennsylvania (Nos. 41 and 42 on Plate 1) and on numerous drillings in Morgan, Muskingum, and other Counties in Ohio where the Devonian shales have less thickness than in southern Pennsylvania, and on the fairly well-known thickness and character of these Devonian and Ordovician beds in the Alleghany Mountain region of West Virginia, farther east where these measures outcrop. This surrounding information makes the portion of the section below the Catskill more than hypothetical, the probability being that the intervals shown are fairly good while some of the individual formations may be locally absent.

*Dunkard Series.*—The Dunkard Series, composed of 1,150 ft. of sandy, red, and variegated shales, gray sandstones, with a few thin coals and limestones, is barren of oil and gas in this region and the lower 300 to 400 ft. of the series found below the main drainage channels must be drilled through without result.

*Monongahela Series.*—The Monongahela Series, composed of 400 ft. of gray sandstones, gray shales, limestones and coal beds, is also practically barren of oil and gas, and is encountered by the drill everywhere along the deeper part of the basin. At its base is the great Pittsburgh Coal bed used widely as a key rock by the drillers of Pennsylvania and West Virginia.

*Conemaugh Series.*—The Conemaugh Series, comprising 500 to 600 ft. of gray sandstones, red and sandy shales and a few coals and limestones, contains the Little Dunkard sand in the lower portion and the Big Dunkard at its base, both of which have produced oil at several localities in the State.

*Allegheny Series.*—The Allegheny Series, 250 ft. thick and composed of gray sandstones, gray shales and having a few coals and thin limestones, contains the Burning Springs and Gas sands, that produce oil in great quantity along the Burning Springs Anticline.

DAVID B. [illegible]

COLUMNAR SECTION FOR CENTER OF WEST PENNA [illegible]

([illegible])

Vertical Scale: [illegible]

| | Name of Series | [illegible] | Thickness in Feet | [illegible] |
|---|---|---|---|---|
| [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| [illegible] | [illegible] | | | |
| [illegible] | [illegible] | | | |
| [illegible] | Allegheny | | | |
| [illegible] | Pottsville | | | |
| [illegible] | [illegible] | | | |

[illegible]

*Pottsville Series.*—The Pottsville Series, composed of gray conglomeratic sandstones, gray shales, and having numerous coal beds, varies in thickness from about 200 ft. in the northern panhandle to nearly 4,000 ft. at the southern end next to Virginia. Along the axis of the Appalachian Basin, its thickness at the Pennsylvania line is about 250 ft., and this increases gradually to nearly 1,000 ft. at the Kentucky line. Oil is often found in several of its sands, the upper one being called the Second Cow Run, and the lower ones being usually termed the First, Second, and Third Salt sands. Copious flows of salt water are usually found in one or more of these horizons, giving rise to their names.

*Mauch Chunk Series.*—The Mauch Chunk Series, composed mainly of red and green shales, with a few flaggy green fine-grained sandstones and impure limestones, varies in thickness from only a few feet in some of the western counties to about 3,500 ft. in Summers and Mercer. Along the oil belt it is usually about 250 ft. thick, with the Maxton oil and gas sand near the base.

*Greenbrier Limestone.*—The Greenbrier limestone, usually a solid mass of hard limestone, varies in thickness from 50 ft. in the northern panhandle to 1,500 ft. in Greenbrier and other southern Counties. Along the Appalachian Basin, it averages about 100 ft. and is known as the "Big Lime," and is the most dependable key rock below the Pittsburgh Coal, as the intervals from it to the lower sands are fairly constant over wide areas. It is seldom a producing horizon but makes some oil and gas from sandy streaks in Lincoln, Wayne, and Mingo Counties.

*Pocono Series.*—The Pocono Series, composed of gray sandstones and gray shales, is usually about 500 ft. thick in all the counties along the Appalachian Basin, and contains the Keener, Big Injun, Squaw, Weir, and Berea sands. Of these the Big Injun is the most prolific oil and gas horizon of the State, possibly deriving its oil from the Big Lime which often rests directly upon it; the Berea sand is a close second to the Injun, being of widespread occurrence and often productive over wide areas.

*Catskill Series.*—The Catskill Series, composed of red or brown sandstones, separated by red shales, varies from 500 to 800 ft. in thickness and contains the Gantz, Fifty-foot, Thirty-foot, Gordon Stray, Gordon, Fourth, Fifth or McDonald, Sixth or Bayard, and Seventh or Elizabeth sands. Of these the Gordon is the most valuable producer, ranking with the Berea in the quantity of oil but not having such a wide occurrence. The sands of this series are the lowest that have been commercially productive in the State. Southward toward Kentucky, the series thins out completely, being scarcely represented at all south of the Great Kanawha River.

*Chemung, Portage and Hamilton Series.*—The Chemung and Portage, composed of green, gray and brown shales respectively, with lenticular sandstones, the Chemung being probably about 1,500 ft., the Portage

about 800, and the Hamilton about 700, contain no productive sands in West Virginia, although the Speechley sand of the Chemung has made shows of oil and gas in various wells. In northern Pennsylvania, there are several productive horizons in this group. Westward in Ohio and southward toward Kentucky, these three series thin out almost completely, leaving the Berea sand only 700 to 1,000 ft. above the Corniferous limestone. These shales form a great barrier that has prevented any exploitation of the lower Devonian sands in northern West Virginia.

*Marcellus Shale.*—The Marcellus shale, known as the Ohio black shale in the State of that name and as the Chattanooga shale in Tennessee, and often called simply the Devonian black shale, is probably about 300 ft. thick along the Appalachian Basin in West Virginia. It is a black, bituminous horizon, and contains a few lenticular gas sands in Ohio and Kentucky.

*Corniferous Limestone.*—The Corniferous limestone, probably not more than 50 ft. thick in West Virginia, is the Ragland oil sand of Menefee and Morgan Counties, Kentucky. It has been penetrated in West Virginia by only three wells near the Kentucky line (Nos. 26, 26A, and 27 on Plate 1), and in two or three wells in Randolph and Tucker Counties in the Alleghany Mountains, the detailed records of only one of which (No. 32) has been obtained. It is probable that the most eastern oil counties of the State do not have this horizon, as its presence is not noted in the Alleghany Mountains where it should crop, and where it should be easily recognized from its well-known flints and fossils.

*Oriskany Sandstone.*—The Oriskany sandstone, gray in color and probably 150 ft. thick, and a producer of oil and gas in southern Indiana and central New York, was found in the Geary Well (No. 42 on Plate 1) in Pennsylvania, where it was 155 ft. thick, and in the Slaughter Creek Well (No. 27 on Plate 1) in Kanawha County, West Virginia, where it was 15 ft., and in the Parsons Well (No. 32 on Plate 1), where it was 80 ft. West of the basin it is not noted in the deep wells of Ohio but on the eastern side of the Alleghany Mountains it is the well-known glass sand of Morgan County, West Virginia, and it seems likely that it is represented along the basin in West Virginia.

*Helderberg, Salina, and Niagara.*—The Helderberg, Salina, and Niagara limestones, often comprising a solid limestone mass 800 ft. or more thick, and known in Ohio drilling parlance as the "Big Lime," has been penetrated in West Virginia only by the Bartram (No. 26A) and Slaughter Creek (No. 27) Wells, and was found in the Geary (No. 42) Well in Pennsylvania. Since this great limestone mass is found in these three wells and is universally present in Ohio, and is also found in much the same great development in West Virginia east of the Alleghanies, it seems safe to conclude that it is present along the Appalachian Basin with the same or greater thickness. It is possibly the parent reser-

voir from which much of the White Medina sandstone, or "Clinton Sand" oil of Ohio is derived, so that its presence along the basin is of special interest.

*Clinton Shales.*—The Clinton variegated shales are usually found in Ohio between the "Big Lime" and the "Clinton" sand, varying in thickness from 50 to 200 ft., and probably occur in West Virginia along the Basin.

*Medina White Sandstone.*—The "Clinton" oil sand of southern Ohio, believed by I. C. White and others to represent the Medina white sandstone, is the great oil-and gas-producing stratum of Hocking, Fairfield, Perry, and other southern Ohio Counties, being usually 10 to 50 ft. thick. No drill has ever reached it in West Virginia, but it is present in the Alleghany Mountains east of the oil fields and probably occurs along the Basin.

*Medina Shales.*—The Medina variegated shales, with probable lenticular sandstone horizons, noted in many deep wells in southern Ohio, may have a thickness of 500 ft. along the Appalachian Basin.

*Martinsburg Shale.*—The Martinsburg shale, often called the Cincinnati or Hudson, is found in Ohio, Kentucky, and eastern West Virginia, and may therefore be taken for granted along the Appalachian Basin in West Virginia. According to Hoeing,[4] this series furnishes several oil and gas sands in Wolfe, Morgan and other Kentucky Counties. Its thickness is possibly 500 to 1,000 ft. along the Appalachian Basin.

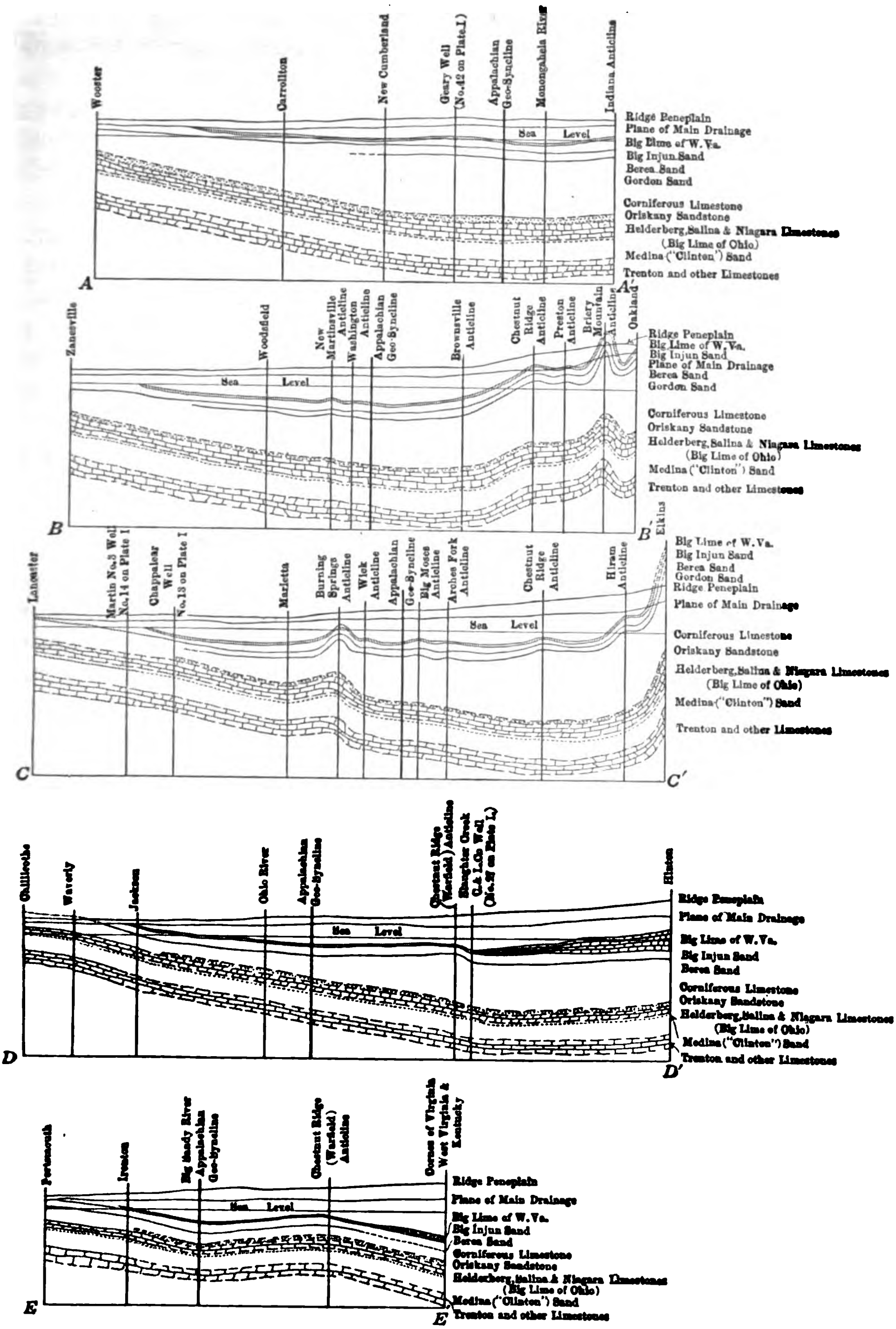
*Utica Shale.*—The Utica formation, consisting of black shales with probable sandstone lentils, may have a thickness of 300 ft. along the basin, although the information regarding it in surrounding States is vague and uncertain.

*Trenton Limestone.*—The Trenton group, composed of several thick limestones, often coalesced into one thick mass, 1,000 to 1,200 ft. thick, is the great oil and gas-producing zone of north-central Ohio and also is productive in Indiana, central New York, and in some parts of Kentucky. The fact that it occurs in these surrounding States and also crops in great thickness in West Virginia east of the Alleghanies seems to leave little doubt that it occurs also, with its usual great thickness, along the Appalachian Basin.

## *Cross-Sections*

Based on the evidence of the numbered wells shown on Plate 1, many of which were very deep tests, and on many hundreds of oil and gas wells, the records of which are published in the West Virginia State Reports, and also on the outcropping rocks around the rim of the Appalachian Basin, five cross-sections, the geographic locations of which are shown

[4] J. B. Hoeing: Oil and Gas Sands of Kentucky, *Bulletin No.* 1, *Kentucky Geological Survey*, pp. 39–40 (1905).




## CROSS SECTIONS

Horizontal Scale: $\frac{11}{32}$ = 16 Miles
Vertical Scale: $\frac{11}{32}$ = 4000 Ft.

Solid Lines Indicate Sands Determined from Well Records
Dotted Lines Indicate Supposed Position of Deep Sands as Determined from a few Deep Wells and from Intervals East and West of Appalachian Basin

PLATE 3.—CROSS-SECTIONS AS INDICATED ON PLATE 1.

on Plate 1, have been plotted across the basin in West Virginia (Plate 3). The scale of the plates does not permit the representation in detail of the surface hills and valleys but a solid line shows the ancient peneplain surface of the ridges, and another shows the plane of the main drainage channels with reference to sea level, from which the approximate interval from this latter plane to the principal sands may be determined, as all estimates for drilling must start from this drainage plane. Formations shown with solid lines are plotted from actual records but the gaps are filled in with dotted lines to represent the probable positions of the various deep sands and limestones as outlined above. Of the more shallow sands, only two or three are plotted for use as key rocks. These cross-sections reveal the very interesting probability that the well-known gradual eastward thickening of the Chemung, Portage, and Hamilton shales shifts the axis of the Appalachian Basin of the lower limestones and sands many miles farther east than the position it occupies when referred to the formations above these shales.

### *Deep-Well History*

The accompanying table, containing the abbreviated records of a large number of wells, has been compiled from the various sources indicated to show the different deep tests that have been made in West Virginia and in neighboring States to find oil in the sands below the Catskill Series. In Ohio, where hundreds of productive wells have been drilled to the "Clinton" and Trenton, only those have been selected that lie nearest to the Appalachian Basin, being located along a fairly straight line reaching from Lake Erie to the Ohio River, and in Kentucky only a few wells are listed to show the typical formations and intervals. Each well is given a tabular number that corresponds to the same number placed beside the well symbol on Plate 1.

Of the wells tabulated for West Virginia, only four (Nos. 26, 26A, 27 and 32) were drilled as deep as the Corniferous limestone, and only three of these went through the Oriskany sandstone into the Niagara and Helderberg limestone. No well in the State has ever been drilled into the Medina White sandstone ("Clinton" oil sand of Ohio) or into the Trenton limestone. All the wells drilled to the Oriskany sandstone have been somewhat distant from the main oil fields of the State and are therefore indecisive as tests of its oil and gas possibilities. Briefly, then, the deep sands not yet tested in the main West Virginia oil fields are the Corniferous limestone ("Ragland" of Kentucky), Oriskany, Medina white sandstone ("Clinton" of Ohio), and the Trenton limestone, with a possibility of another sand horizon in the Martinsburg (Hudson) group, which produces oil in Kentucky.

The well most typical of the formation and conditions likely to be

encountered in drilling for these deep sands in West Virginia is the Geary deep well (No. 42 on Plate 1) drilled near McDonald, Pa., and only about 50 miles due north of the main West Virginia fields. This well holds the unique distinction of being the deepest hole ever drilled in the United States, and its record down into the Ohio "Big Lime" was published by I. C. White.[5] Judging from the record of this well and that of the Slaughter Creek (No. 27) deep well, it seems likely that troublesome salt water may be expected in the Oriskany and Salina beds along the minor synclines that lie parallel to the main Appalachian Basin, and that oil and gas will more probably be found along the axes of the anticlines or at no great distance down their slopes.

This theory that oil may be found principally along the anticlines is contrary to the experience of operators who have exploited the sands of the Catskill Series in the same region, and who have learned through the drilling of many thousands of wells that these latter sands are usually not water-bearing and the oil is therefore almost universally found along the synclines.

Whether deep-sand oil and gas exist in commercial quantity along the Appalachian Basin in West Virginia will be known only through the ultimate test of the drill, but the fact that these sands are widely productive in Ohio, Kentucky, Indiana, and New York, in regions where structural relief is similar to that in West Virginia, and the further strong probability, and almost certainty, that these porous reservoirs actually exist in close proximity to the great limestones that, along with the Devonian black shales, are evidently their main source of oil in other localities, indicates that they will be found commercially productive along the Basin. The fact that the bituminous shales are undoubtedly much thicker in West Virginia than in Ohio, Kentucky and Tennessee indicates an even more hopeful possibility of oil.

If the doubt as to the presence of oil be dismissed as unreasonable, the problem of the operator resolves itself into the double question; first, as to the location of the pools of oil and gas; and, second, as to the mechanical possibility of drilling wells 5,000 to 10,000 ft. deep. As outlined above, the most favorable localities seem to be along or near the axes of the anticlines where the danger of finding salt water would be minimized and where the gas, if any, should most certainly collect. The search for oil would naturally be made on either side of the anticlines by making locations at increasing distances from the axes.

That drilling may be carried on successfully to the great depths indicated by the figures above seems open to little serious doubt. In 1860, when the search for oil and gas may be said to have been actively begun in the Appalachian Basin, few, if any, wells had been drilled to a

[5] I. C. White: Notes on a Very Deep Well Near McDonald, Pennsylvania, *Bulletin of the Geological Society of America*, vol. 24, p. 275 (June 10, 1913).

*Summarized Record of Deep*

| No. on Plate 1 | Farm Name and Number and Oil Company | County and State | Elevation above Tide, Feet | Big Lime of W. Va. | | Berea Sand | | Corniferous Limestone | | Oriskany Sandstone | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet |
| 1 | J. A. Giddings lot, Jefferson O. & G. Co. | Ashtabula, Ohio. | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| 2 | Garfield & Caine, East Ohio G. Co. | Cuyahoga, Ohio. | 780 | ... | ... | 10 | 80 | ... | ... | ... | ... |
| 3 | Citizens No 1. | Lorain, Ohio. | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| 4 | E. S. Albert No. 1. | Medina, Ohio. | 920 | ... | ... | 386 | 10 | ... | ... | ... | ... |
| 5 | Barberton Chem. Co. No. 1. | Summit, Ohio. | ... | ... | ... | 460 | 10 | 2,180 | ... | ... | ... |
| 6 | Wooster well, H. B. Odenkirk. | Wayne, Ohio. | 860 | ... | ... | 465 | 30 | ... | ... | ... | ... |
| 7 | Kaylor. | ... | ... | ... | ... | 615 | 10 | ... | ... | ... | ... |
| 8 | Fairall. | Muskingum, Ohio. | ... | ... | ... | 834 | 16 | ... | ... | ... | ... |
| 9 | Dillon Falls well, Chicago-Zanesville O. & G. | Muskingum, Ohio. | ... | ... | ... | 895 | 12 | ... | ... | ... | ... |
| 10 | George Handehsy, Zanesville G. & O. Co. | Muskingum, Ohio. | ... | ... | ... | 1,010 | 23 | ... | ... | ... | ... |
| 11 | W. J. Roberts. | Muskingum, Ohio. | ... | 250 | 38 | 847 | 14 | ... | ... | ... | ... |
| 12 | McConnellsville Fair Grounds, McConnellsville City. | Morgan, Ohio. | 725 | 625 | 44 | 1,279 | 66 | 3,082 | 2 | 3,125 | 8 |
| 13 | T. J. Chappalear, J. P. Fishel. | Morgan, Ohio. | 750 | ... | ... | 1,150 | 20 | ... | ... | ... | ... |
| 14 | Martin No. 3. E. S. Martin. | Perry, Ohio. | ... | ... | ... | 920 | 20 | ... | ... | ... | ... |
| 15 | Crouse No. 1. B. S. Stretton. | Fairfield, Ohio. | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| 16 | Logan town, Logan town. | Hocking, Ohio. | 748 | ... | ... | 683 | 45 | ... | ... | ... | ... |
| 17 | Columbus Hocking C. & I. Columbus Hocking C. & I. | Athens, Ohio. | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| 18 | Waverly O. & Gas. Co. Waverly O. & Gas Co. | Pike, Ohio. | 575 | ... | ... | ... | ... | ... | ... | ... | ... |
| 19 | Buckeye Coal Co. No. 2. Buckeye Coal Co. | Jackson, Ohio. | ... | ... | ... | 640 | 20 | ... | ... | ... | ... |
| 20 | At Vinton village. | Gallia, Ohio. | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| 21 | At Mt. Vernon village. | Lawrence, Ohio. | ... | ... | ... | 1,095 | 25 | ... | ... | ... | ... |
| 22 | At Ironton town. | Lawrence, Ohio. | 500 | ... | ... | 1,010 | 47 | ... | ... | ... | ... |
| 23 | Burns well. | Morgan, Kentucky, | ... | 368 | 106 | 1,095 | 10 | ... | ... | ... | ... |

*Wells Shown on Plate 1*

| Helderberg, Salina and Niagara, "Big Lime of Ohio" | | Medina ("Clinton") Sand | | Trenton Limestone | | Total Depth, Feet | Reference to Detailed Record | Remarks |
|---|---|---|---|---|---|---|---|---|
| Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | | | |
| 1,810 | 288 | 2,127 | 13 | ... | ... | 2,140 | Ohio G. S. 4th series, *Bull*. 1, p. 303. | Oil well in "Clinton" sand. |
| 1,210 | 1,455 | 2,680 | ... | ... | ... | 2,770 | Ohio G. S. 4th series, *Bull*. 12, p. 65. | Heavy flow of water and oil show in Niagara lime. Gas in "Clinton" sand. |
| 440 | 850 | ... | ... | 2,645 | ... | .... | Ohio G. S. 4th series, *Bull*. 12, p. 64. | No "Clinton" sand. |
| 1,512 | 1,127 | 2,749 | 14 | ... | ... | 2,777 | Ohio G. S. 4th series, *Bull*. 12, p. 62. | |
| 2,180 | 350 | 2,500? | 30 | ... | ... | 3,006 | Ohio G. S. 4th series, *Bull*. 1, p. 296. | Gas show(?) in "Clinton" sand. Rock salt and lime in lower 300 ft. of hole. |
| 1,830 | 1,085 | 3,104 | 31 | ... | ... | 3,140 | Ohio G. S. 4th series, *Bull*. 12, p. 60. | 1,440,000 cu. ft. of gas daily and oil show in "Clinton" sand. Heavy flow of brine at 2,055 ft. |
| 1,725 | 960 | 2,851 | 28 | ... | ... | 2,957 | Ohio G. S. 4th series, *Bull*. 12, p. 59. | Gas show in "Clinton" sand. |
| 2,035 | 934 | 3,141 | 21 | ... | ... | 3,194 | Ohio G. S. 4th series, *Bull*. 12, p. 37. | "Clinton" sand oil well. |
| 2,120 | 1,275 | 3,543 | 21 | ... | ... | 3,567 | Ohio G. S. 4th series, *Bull*. 12, p. 35. | Salt water 2,560 ft. and 3,205 ft. Shows of oil and gas in "Clinton" sand. |
| 2,640 | 1,005 | 3,709 | 38 | ... | ... | 3,805 | Ohio G. S. 4th series, *Bull*. 12, p. 36. | Dry hole. |
| 2,195 | 970 | 3,296 | ... | ... | ... | 3,314 | .................... | Gas in "Clinton" sand. |
| ... | ... | ... | ... | ... | ... | 3,186 | Ohio G. S. 4th series, *Bull*. 1, p. 145. | Gas in Berea sand. |
| 2,700 | 900 | 3,935 | 9 | ... | ... | 3,947 | Ohio G. S. 4th series, *Bull*. 12, p. 38. | No oil or gas; 1 bailer of water per hour in Niagara lime. |
| 2,170 | 1,170 | 3,175 | 17 | ... | ... | 3,213 | Ohio G. S. 4th series, *Bull*. 12, p. 23. | Oil well in "Clinton" sand. |
| 1,378 | ... | 2,180 | 5 | 3,470 | ... | 3,569 | Ohio G. S. 4th series, *Bull*. 12, p. 28. | No oil or gas noted in record, but was probably a "Clinton" gasser. |
| 1,715 | 715 | 2,627 | 18 | ... | ... | 2,694 | Ohio G. S. 4th series, *Bull*. 12, p. 39. | Dry hole. |
| ... | ... | ... | ... | ... | ... | .... | Ohio G. S. 4th series, *Bull*. 12, p. 41. | Drilled to "Clinton" sand. No oil or gas. |
| 485 | 490 | ... | ... | 1,960 | 808 | 3,335 | Ohio G. S. 4th series, *Bull*. 12, p. 48 | Some salt water and gas in Niagara lime; oil and gas show in Trenton. |
| 660 | 1,454 | 2,255 | 8 | ... | ... | 2,315 | Ohio G. S. 4th series, *Bull*. 12, p. 44. | 4,000,000 cu. ft. of gas daily in "Clinton" sand. |
| ... | ... | ... | ... | ... | ... | 3,200 | Ohio G. S. 4th series, *Bull*. 12, p. 44. | Dry hole. No "Clinton" sand. |
| 1,810 | 620 | ... | ... | ... | ... | 2,730 | Ohio G. S. 4th series, *Bull*. 12, p. 50. | Dry hole. No "Clinton" sand. |
| 1,827 | 584 | ... | ... | 3,440? | ... | 3,600 | Ohio G. S. 4th series, *Bull*. 12, p. 49. | Dry hole. No "Clinton" sand. |
| ... | ... | 1,408 | 94 | ... | ... | 1,508 | Ky. G. S. *Bull*. 1, p. 159. | "Clinton" sand oil well. |

| No. on Plate 1 | Farm Name and Number and Oil Company | County and State | Elevation above Tide, Feet | Big Lime of W. Va. | | Berea Sand | | Corniferous Limestone | | Oriskany Sandstone | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet |
| 24 | Caney Creek well, G. M. Sullivan. | Morgan, Kentucky | ... | 453 | 69 | 1,035 | 24 | 1,370 | ... | ... | ... |
| 25 | Blaine Creek well. | Lawrence, Kentucky | ... | ... | ... | 1,660 | 2 | 2,592 | −5 | ... | ... |
| 26 | Central City well, T. H. Harvey. | Cabell, W. Va. | 530 | 970 | 150 | 1,730 | 25 | 2,760 | 10 | ... | ... |
| 26 A | David Bartram No. 483, United Fuel Gas Co. | Wayne, W. Va. | 595 | 1,220 | 80 | 1,875 | 20 | 2,760 | 410 | ... | ... |
| 27 | Slaughter Creek C. & L. Co. W. S. Edwards Oil Co. | Kanawha, W. Va. | 640 | 1,453 | 207 | 2,093 | 12 | 4,945 | 90 | 5,035 | 15 |
| 28 | Griffith No. 1, Rich Creek O. & G Co. | Mercer, W. Va. | 2,090 | ... | ... | ... | ... | ... | ... | ... | ... |
| 29 | B. T. Baker No. 1, Summers O. & G. Co. | Summers, W. Va. | 1,586 | 1,020 | 1,057 | 2,600 | 25 | ... | ... | ... | ... |
| 30 | P. C. Daniels No. 1. Tygarts Valley Oil Co. | Randolph W. Va. | 1,990 | ... | ... | ... | ... | ... | ... | ... | ... |
| 31 | Alonso W. Murphy No. 1. Mead *et al.* | Randolph, W. Va. | 1,995 | ... | ... | ... | ... | ... | ... | ... | ... |
| 32 | Parsons Pulp & Lumber Co. Parsons Pulp & Lumber Co. | Tucker, W. Va. | 1,655 | ... | ... | ... | ... | 3,825 | 20 | 3,980 | 80 |
| 33 | J. C. Benson No. 3,612 Hope Natural Gas Co. | Barbour, W. Va. | 1,026 | 1,235 | 90 | 1,615 | 138 | ... | ... | ... | ... |
| 34 | Porter Maxwell No. 3,869, Hope Natural Gas Co. | Harrison, W. Va. | 1,050 | 1,531 | ... | ... | ... | ... | ... | ... | ... |
| 35 | M. J. Coplin No. 1, Tri-State Gas Co. | Harrison, W. Va. | 1,030 | 1,288 | 77 | 1,711 | 13 | ...... | ... | ... | ... |
| 36 | Absalom Knotts No. 1, Greensboro Gas Co. | Taylor, W. Va. | 1,460 | 960 | 50 | 1,460 | 120 | ... | ... | ... | ... |
| 37 | R. S. Hauser No. 1, Greensboro Gas Co. | Taylor, W. Va. | 1,480 | 950 | 40 | ... | ... | ... | ... | ... | ... |
| 38 | Russell McCartney No. 1, Greensboro Gas Co. | Taylor, W. Va. | 1,580 | 1,055 | 35 | 1,500 | 90 | ... | ... | ... | ... |
| 38 A | T. E. Dye No. 1, Am. Hydroscope Co. | Ritchie, W. Va. | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| 39 | J. F. Dobbs No. 1, South Penn Oil Co. | Marshall, W. Va. | 980 | 1,760 | 30 | ... | ... | ... | ... | ... | ... |
| 40 | Boggs Run well, Wheeling Dev. Co. | Marshall, W. Va. | 900 | ... | ... | ... | ... | ... | ... | ... | ... |
| 41 | Wm. Bedell, Forest Oil Co. | Allegheny Penna. | ... | 1,050 | 50 | 1,515 | 50 | ... | ... | ... | ... |
| 42 | R. A. Geary No. 1,770, Peoples Nat. Gas Co. | Washington, Penna. | 1,050 | 953 | 29 | 1,610 | 12 | 6,008 | 37 | 6,045 | 155 |
| 43 | Witherop, Conway Bros. | Venango, Penna. | 955 | ... | ... | ... | ... | 3,870 | 10 | ... | ... |

| Helderberg, Salina and Niagara, "Big Lime of Ohio" | | Medina ("Clinton") Sand | | Trenton Limestone | | Total Depth, Feet | Reference to Detailed Record | Remarks |
|---|---|---|---|---|---|---|---|---|
| Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | Depth Top, Feet | Thickness, Feet | | | |
| ... | ... | 1,408 | ... | ... | ... | 2,021 | Ky. G. S. *Bull.* 1, p. 72. | Oil and gas in "Clinton" and Caney Sands. Salt water in Clinton. Probably stopped just short of Trenton. |
| ... | ... | ... | ... | ... | ... | 2,597 | Ky. G. S. *Bull.* 1, p. 78. | Gas in Pottsville Conglomerate and Berea sand. |
| ... | ... | ... | ... | ... | ... | 2,770 | W. Va. G. S. Vol. I (A), p. 495. | Oil and gas in Berea sand; gas in bastard limestone at 2,485 ft. |
| 2,760 | 410 | ... | ... | ... | ... | 3,130 | Unpublished. | Dry hole; oil show in Weir sand at 1,790 ft.; Corniferous and Niagara recorded together. |
| 5,050 | 545 | ... | ... | ... | ... | 5,595 | W. Va. G. S. Kanawha Report, p. xix. | Squaw sand gas; water in Niagara Lime at 5,592 ft. |
| ... | ... | ... | ... | ... | ... | 4,105 | W. Va. G. S. Raleigh Report. | Still drilling; base of Pottsville, 105 ft. |
| ... | ... | ... | ... | ... | ... | 2,664 | W. Va. G. S. Raleigh Report. | Dry hole. |
| ... | ... | ... | ... | ... | ... | 2,000? | W. Va. G. S. Barbour-Upshur-Randolph Report. | Starts near base of Chemung Series. Shows of oil and gas. |
| ... | ... | ... | ... | ... | ... | 2,385 | W. Va. G. S. Barbour-Upshur-Randolph Report. | Starts near base of Chemung Series. Dry hole. |
| ... | ... | ... | ... | ... | ... | 4,250 | Not published. | Starts near base of Chemung Series. Pockets of gas at 825 to 835 ft., 1,053 and 3,830 ft. Strong salt water at 3,990 ft. to 4,000 ft. |
| ... | ... | ... | ... | ... | ... | 4,570 | W. Va. G. S. Barbour-Upshur-Randolph Report. | Gas in Big Lime, Big Injun, Berea, and Kane(?) sands. |
| ... | ... | ... | ... | ... | ... | 4,386 | W. Va. G. S. Barbour-Upshur-Randolph Report. | |
| ... | ... | ... | ... | ... | ... | 4,028 | W. Va. G. S. Doddridge-Harrison Report, p. 120. | Little gas in lime and shells at 2,950 ft. |
| ... | ... | ... | ... | ... | ... | 3,955 | W. Va. G. S. Barbour-Upshur-Randolph Report. | Gas show at 1,210 ft. |
| ... | ... | ... | ... | ... | ... | 4,437 | W. Va. G. S. Barbour-Upshur-Randolph Report. | Light gas in Big Injun sand; Kane(?) sand, 4,108 to 4,162 ft. |
| ... | ... | ... | ... | ... | ... | 3,506 | W. Va. G. S. Barbour-Upshur-Randolph Report. | Gas show, 2,425 ft. |
| ... | ... | ... | ... | ... | ... | 4,450 | W. Va. G. S. Doddridge-Harrison Report, p. 284. | Flow of oil in Speechley sand at 3,125 ft. |
| ... | ... | ... | ... | ... | ... | 3,411 | W. Va. G. S. Marshall-Wetzel-Tyler Report, p. 390. | Gordon sand oil; Speechley sand, 3,344 to 3,376 ft. with some dark oil. |
| ... | ... | ... | ... | ... | ... | 4,500 | W. Va. G. S. Vol. I, p. 364. | Base of Big Injun at 1,570 ft.; show of oil at 2,955 ft. |
| ... | ... | ... | ... | ... | ... | 5,575 | W. Va. G. S. Vol. I (A), p. 104. | Speechley sand, 3,150 to 3,165 ft.; Bradford sand, 3,500 to 3,520 ft. with trace of oil; hole stopped in Marcellus shale. |
| ... | ... | ... | ... | ... | ... | 6,299+ | G. S. A., *Bull.* Vol. 24, p. 275, 1913. | Gas, 4,850 ft., 4,870, 5,900, 5,905, 5,910, 5,915, 6,060 ft. Not yet completed. |
| ... | ... | ... | ... | ... | ... | 3,880 | W. Va. G. S. Vol. I (A), p. 86. | Slight shows of oil and gas. |

depth of 1,000 ft. In 1890, when development was at its height, 3,000 ft. was considered an unusual depth, but by 1910 several holes had been drilled more than 5,000 ft. and at the present time at least two wells in the world have been drilled more than 7,000 ft. The advance in drilling has been just as rapid as in any other line of human endeavor where mechanical ingenuity and skill must be used and in these qualifications the Scotch-American well driller stands second to none. To say that drilling cannot be continued to greater depths than the Geary and Slaughter Creek wells would call for a total cessation of the enterprising experimentation that has characterized oil-well drilling from its very beginning.

### *Suggestions for Drilling*

A comparison of the cross-sections on Plate 3, in conjunction with the data exhibited on Plates 1 and 2, shows that ideal conditions may be found along several of the anticlines east and west of the main Appalachian Basin, among which may be mentioned the New Martinsville, Washington, Brownsville, Arches Fork, Big Moses, Chestnut Ridge (known as the Warfield in southern West Virginia), and last and best of all, the Burning Springs, which stretches in a north and south direction for more than 30 miles through the very heart of the oil fields and in a region where the Devonian shales are probably much thinner than at points farther east. A well drilled at the point where this anticline crosses Walker Creek, near Sandhill, 10 miles southward from St. Marys, and 5 miles north of Petroleum on the Baltimore and Ohio Railroad, and 5½ miles south of the point where cross-section *CC'* crosses the anticline, would have a tidal elevation of 920 ft. Being at the summit of the great Volcano dome, it would start at an approximate interval of 1,000 ft. below the Pittsburgh Coal and should find the Big Lime of West Virginia at about 100 ft., the Berea sand at about 600 ft., the Corniferous limestone at about 3,500 ft., the Oriskany sandstone at about 3,550 ft., the Medina white sandstone ("Clinton") at about 4,700 ft., and should reach the Trenton limestone at about 6,000 ft. The dome-shaped structure at this point would almost insure the absence of water in the sands, at the same time making the financial risk much less than at any other point in the well-developed oil fields of the State. Oil or gas secured at this point would immediately open a prospective field for 30 miles along the crest of this great anticline where conditions are similar all the way from St. Marys on the Ohio River to Burning Springs near the corner of Wirt, Roane and Calhoun Counties.

Should this region prove to be commercially productive from the lower sands, attention would naturally turn to the other anticlines mentioned, where drilling would be more expensive but the possibility of finding oil and gas reasonably good.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

# Petrography of the Mount Morgan Mine, Queensland.

BY W. E. GABY, B. S., M. A.,* BUTTE, MONT.

(Arizona Meeting, September, 1916)

## INTRODUCTION

SINCE the time of their discovery, the genesis of the ores at Mount Morgan, and the nature of the changes which have affected the surrounding rocks, have been the subject of investigation and speculation by geologists and mining engineers. Developments at this mine, for a long period the greatest gold producer of the world, show that with depth the orebody grades into copper, of which metal it now furnishes a large output, exceeding in value the gold.

In earlier times, the purity of the gold and the first extremely heterogeneous character of the ore made it an interesting problem. Now, the probable depth of the copper enrichment, genesis, and true relations of the neighboring rocks furnish equally profitable material for investigation, and the present somewhat detailed description of these ores and rocks, from microscopic study, may prove of interest.

The mine occupies a highly silicified portion of a belt of quartz-porphyry, which is believed to be the oldest rock of the region. This belt of quartz-porphyry and allied rocks runs north and south between two areas of a later intrusive granite, as shown on the geologic map,[1] Fig. 1. Near the orebody the quartz-porphyry has been intruded by a dense basaltic mass called the "Old Basic." Cutting all three of these igneous masses are two sets of basaltic dikes, one running northwest-southeast, and the other, cross dikes, northeast-southwest. These dikes may be contemporaneous with, instead of later than, the ore. Their number has caused peculiar difficulties in extracting it, as the clay selvage of decomposed material affords no adhesion between dike and ore, causing great blocks to loosen and come down.

The investigation was based on specimens furnished by B. Magnus, former general manager of the mine, and notes on the areal geology by

* Assistant Geologist, Anaconda Copper Mining Co.

[1] Reproduced by permission of the Australasian Institute of Mining Engineers from the plate accompanying the paper of J. M. Newman and G. F. Campbell Brown, *Transactions, Australasian Institute of Mining Engineers*, vol. 15, part 2 (1910).

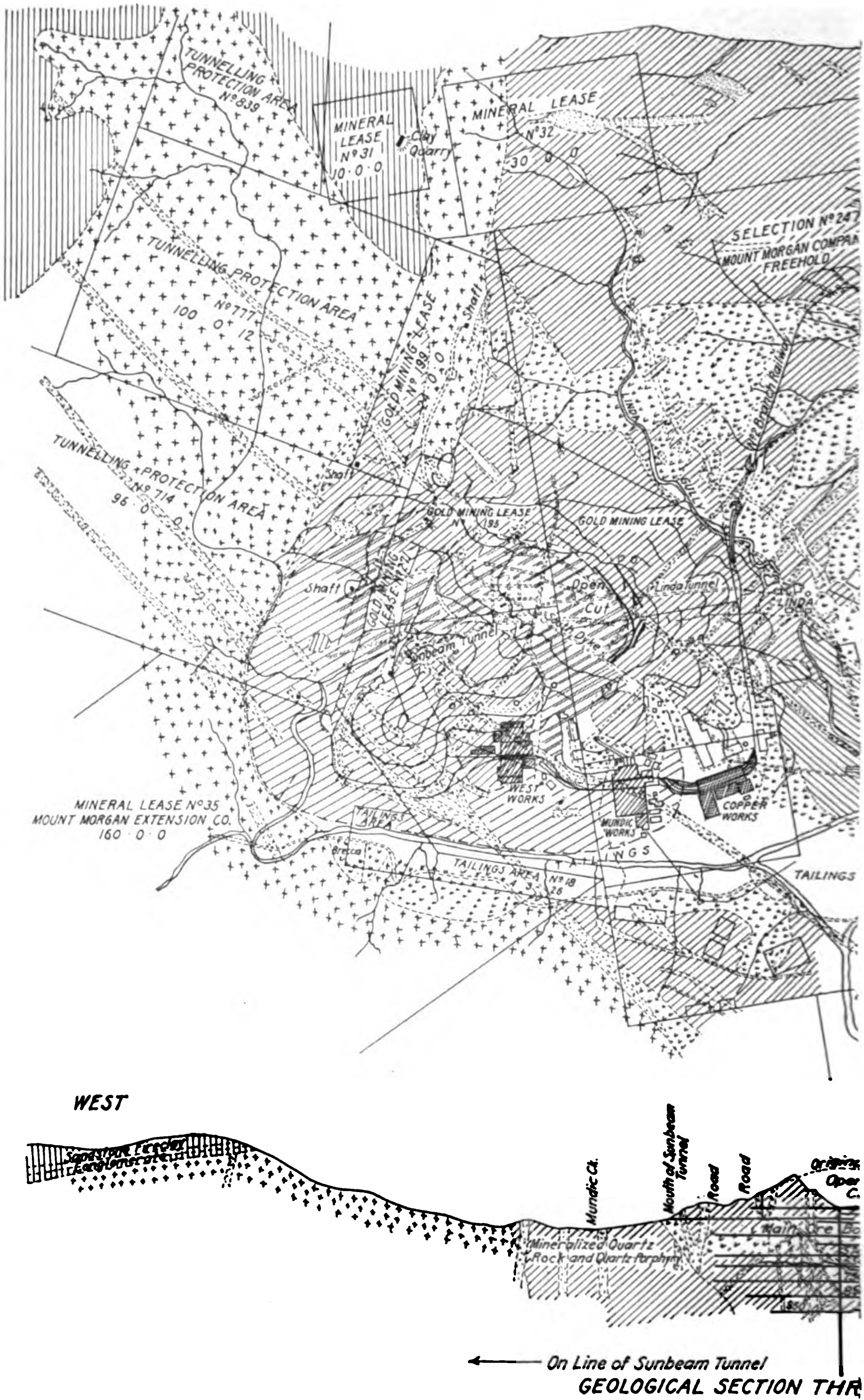



FIG. 1.—GEOLOGICAL MAP OF MOUNT MORGAN AND VICINITY.

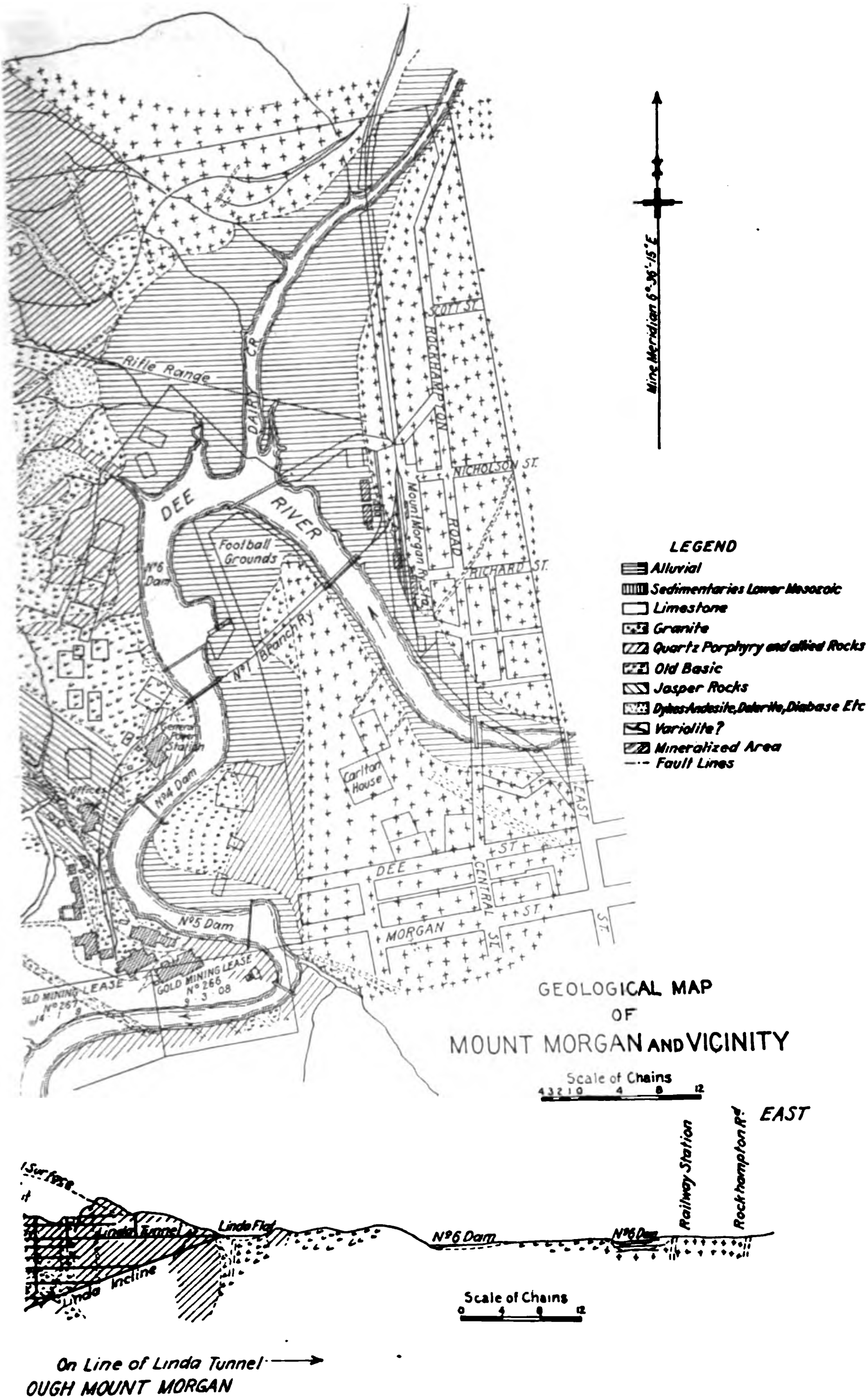



**Fig. 1a.—Geological Map of Mount Morgan and Vicinity.**

J. M. Newman and G. F. Campbell Brown.[2] Reference to the literature shows a wide divergence of opinion concerning its geology to have existed at different times.

## Summary of Opinions on Genesis of Mount Morgan Orebody

The first examination was made by R. L. Jack, as geologist for the Queensland Government, in 1884, and he came to a conclusion, from the

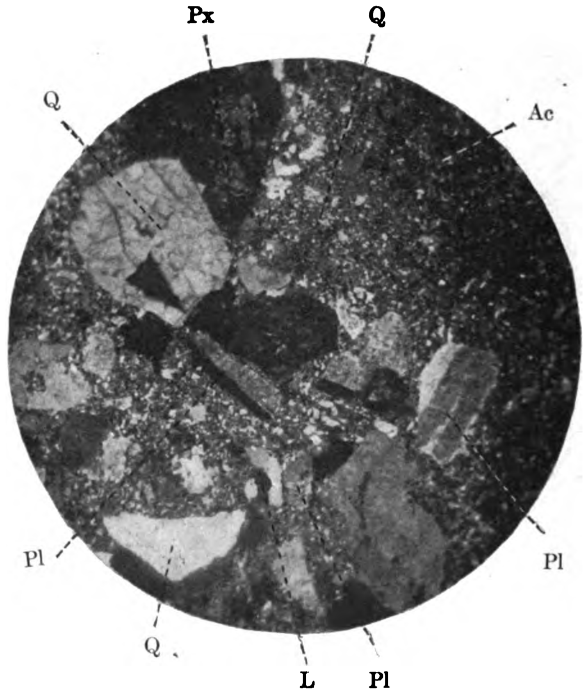



Fig. 2.—Quartz-Porphyry of the Composition of Quartz-Diorite or Quartz-Monzonite, with a Ground-Mass of Fine Granular Secondary Quartz, Chlorite, Leucoxene, and Hematite. Plagioclase and Quartz Make Up the Phenocrysts, and Patches of Pyroxene are Common. The Latter has been Partly Altered to Chlorite and Epidote. Crossed Nicols. × 24.

evidence then available, that nothing but a thermal spring in the open air, or geyser, could account for the formation of the orebody. In 1891, Walter H. Weed of the United States Geological Survey was asked by Mr. Jack to examine and compare a suite of specimens from the mine with the siliceous sinters of the hot spring region of Yellowstone Park. Weed came to a similar conclusion, citing the Steamboat Springs of Nevada, which have long been known to be surrounded by deposits of sinter in fissures of which ore deposition is taking place.

T. A. Rickard, after an examination of the mine the same year, sum-

[2] *Loc. cit.*, pp. 439 to 470.

marized the various views held up to this time as follows: 1. That the deposit is one of a geyser (R. L. Jack, 1884). 2. That it is an auriferous zone traversed by a series of quartz veins of auriferous mundic (gold-bearing pyrite) (J. MacDonald Cameron, 1887). 3. That it is the decomposed cap of a large pyrite lode. (The view held by several local and other mining engineers at the time of his examination.)

Mr. Rickard dismissed the geyser theory as an altogether local appearance, based as it was on a small open cut which showed a fan-shaped

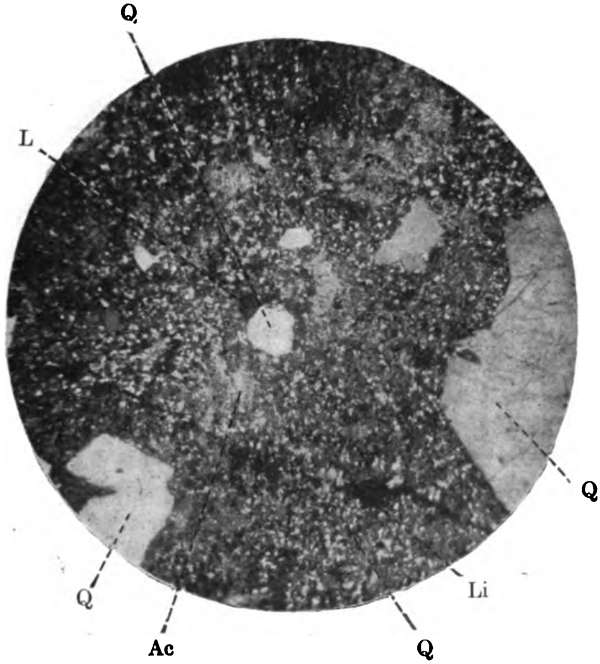



FIG. 3.—QUARTZ-PORPHYRY. THERE IS A STRONG DEVELOPMENT OF ACTINOLITE AMONG THE SMALL GRAINS OF SECONDARY QUARTZ WHICH MAKES UP THE GROUND-MASS, PROBABLY A RESULT OF PROXIMITY TO THE BASALTIC ROCKS. LEUCOXENE (WHITE AND OPAQUE, HENCE DARK) IS ABUNDANT. PHENOCRYSTS OF QUARTZ, AND GROUND-MASS LARGELY SECONDARY QUARTZ. LIMONITE, AS EVIDENCE OF WEATHERING, OCCURS DISSEMINATED AND ALONG FRACTURES. CROSSED NICOLS. × 24.

arrangement of ore, and his own explanation of Mount Morgan was, that it "represents an altered portion of shattered country rock, which, by reason of its crushed condition, was readily acted upon by mineral solutions, and that these solutions replaced the basic and feldspathic with acidic and quartzose material which was also gold-bearing."

The geyser and replacement theories were later reviewed by Dr. Frank D. Adams of McGill University with an interesting account of the discovery and subsequent history of the mine. But by 1899, the reaching of depths below the zone of oxidation revealed data which did not agree with some of these earlier conceptions.

Many different names and origins have also been assigned to the rocks of the district. Metamorphism was supposed by some to have transformed original sedimentary rocks into the present ones, which are, as is generally stated, difficult to distinguish from igneous rocks. The supposed graywacke, quartzites, and shales of sedimentary origin are now known to be igneous or derived from igneous rocks. There are no true sediments near the mine except those to the northwest of it, which overlie

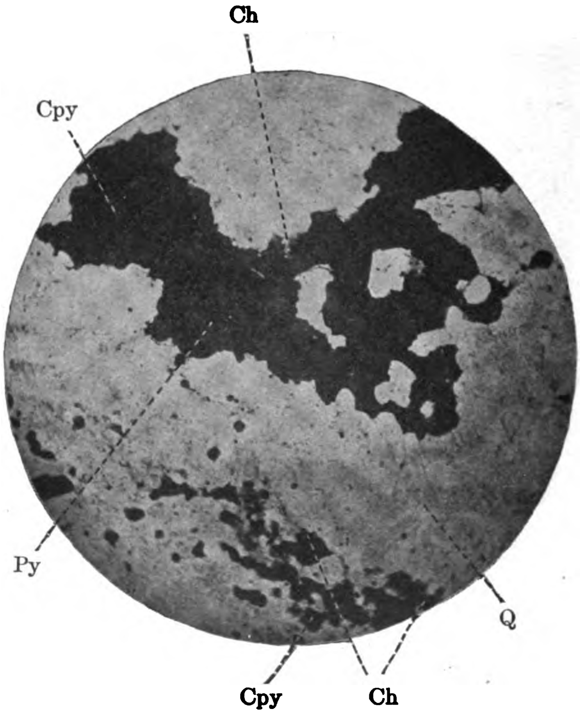



FIG. 4.—ORE, SHOWING PYRITE AND CHALCOPYRITE WITH CHLORITE IN GANGUE OF QUARTZ, THE CHLORITE WITHIN AND AROUND THE SULPHIDES, ALL THREE APPEARING TO OCCUPY THE SPACES AND CHANNELS BETWEEN THE QUARTZ GRAINS. SOME SMALL GREEN NEEDLE-LIKE CRYSTALS, NOT PLAINLY SEEN, ARE PROBABLY ACTINOLITE. PLANE LIGHT. × 24.

the very latest of the intrusive masses. These sediments are Lower Mesozoic in age and consist of conglomerates, fireclays, and sandstones, with a possible bed of volcanic ash or tuff. No specimen of the latter material was available, so this point could not be determined.

The notes by Newman and Brown, and by G. S. Hart, in the *Transactions* of the Australasian Institute of Mining Engineers, from whose maps and descriptions the field relations are taken, represent the most recent published work on the mine.

It is hoped the additional facts here set forth, from microscopic study, may be of assistance in solving the problem of the origin of its ores, as well as give additional information concerning the associated rocks.

In some cases the names of rocks have been changed to more nearly fit their mineralogy and textures as revealed in thin section, the names here given according with the American usage. Acknowledgments are due the authors of the above papers, to Mr. Magnus for specimens and notes, and to Dr. Charles P. Berkey of Columbia University for his kindly assistance in the laboratory.

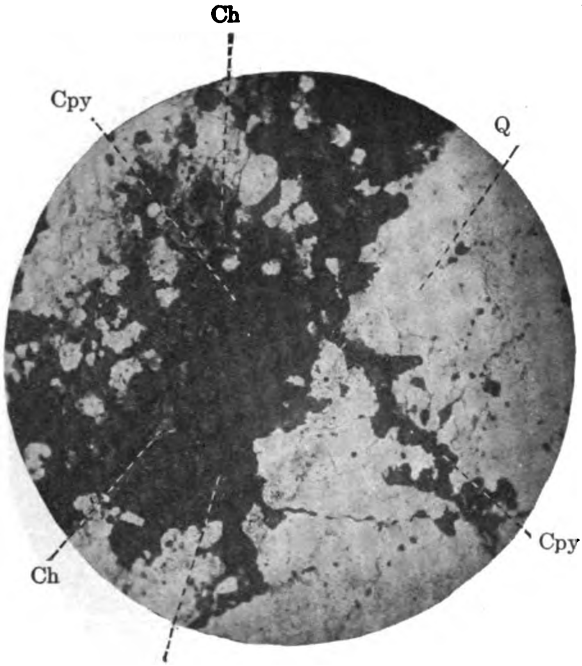



FIG. 5.—ORE, SHOWING RELATION OF CHLORITE TO THE SULPHIDES, AND DISTRIBUTION OF THE PYRITE AND CHALCOPYRITE WITH THE CHLORITE ALONG THE VEINLETS AND SPACES IN THE QUARTZ. PLANE LIGHT. × 24.

## DESCRIPTION OF MICROGRAPHS

The accompanying geologic map, Fig. 1, will make clear the general relationship of the Mount Morgan rocks, and with this in mind, the descriptions follow in the order of their relative geologic age.

Key to Minerals in Photomicrographs

| | | | |
|---|---|---|---|
| Ac | = Actinolite | L | = Leucoxene |
| Ap | = Apatite | Li | = Limonite |
| Aug | = Augite | M | = Magnetite |
| C | = Carbonate | Pl | = Plagioclase |
| Ch | = Chlorite | Px | = Pyroxene |
| Cpy | = Chalcopyrite | Py | = Pyrite |
| Hb | = Hornblende | Q | = Quartz |
| Il | = Ilmenite | | |

*Quartz-Porphyries and Ore*

These rocks are grayish green and porphyritic with a fine-grained ground-mass and scattered irregular-shaped crystals of quartz. Two photomicrographs, Figs. 2 and 3, show them in thin section. Plagioclase is the predominant feldspar and ilmenite is the chief accessory mineral. Secondary development of quartz, actinolite, chlorite, epidote, kaolin, leucoxene, and hematite is a very pronounced characteristic. Later

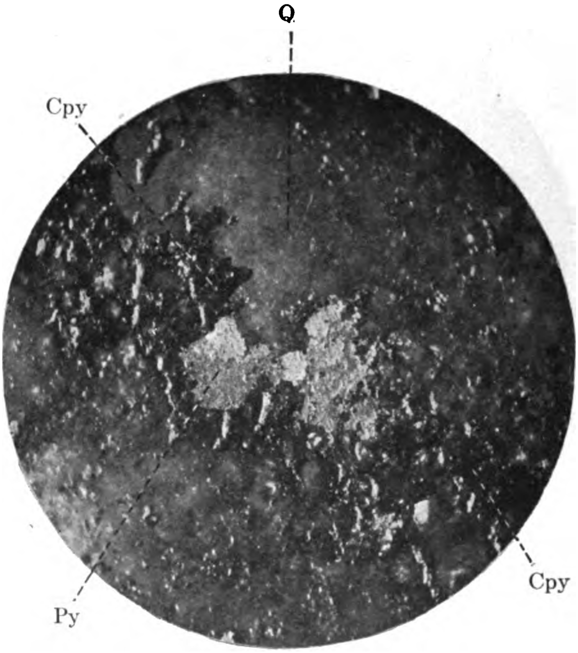


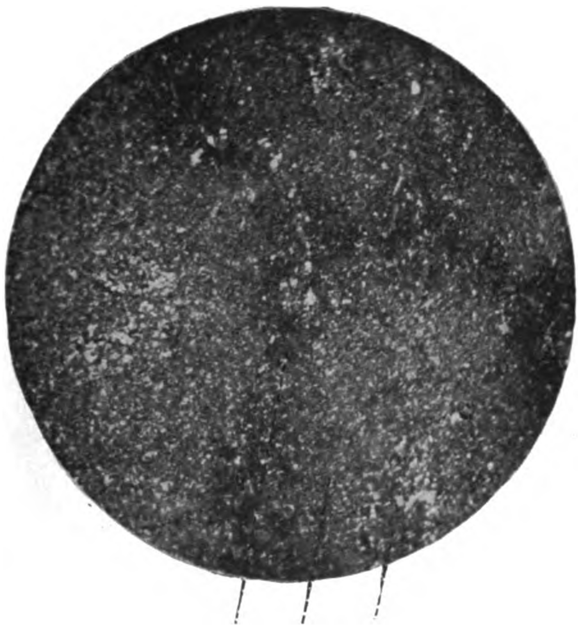

FIG. 6.—ORE. POLISHED SECTION SHOWING DISTRIBUTION OF PYRITE AND CHALCOPYRITE IN THE QUARTZ GANGUE. SILVER NITRATE HAS BEEN USED TO STAIN THE CHALCOPYRITE. THE MINERALS ARE DISSEMINATED AS REPLACEMENTS IN THE QUARTZ, AS WELL AS DISTRIBUTED ALONG THE VEINLETS AND CHANNEL BETWEEN THE QUARTZ GRAINS. BOTH SULPHIDES APPEAR TO HAVE BEEN DEPOSITED AT THE SAME TIME. REFLECTED LIGHT. × 24.

changes have developed some limonite, which is probably derived from pyrite, though occurring chiefly along fractures. Other fractures appear in the slides which are still more recent than these.

The belt of quartz-porphyry, which is the oldest of the rocks, is peculiar on account of the fact that the phenocrysts in it seem to have been subjected to a certain amount of primary fracturing. This would indicate that their original matrix had become somewhat viscous before movement within the mass entirely ceased, although all evidence of flow structure in the matrix or ground-mass has since been obliterated, if ever developed. Specimens of the unaltered rock would undoubtedly

tell this. Nevertheless, its general heterogeneous nature suggests that it was originally a surface flow or large extrusive mass.

The quartz-porphyry within the boundaries of the ore deposit has been thoroughly silicified, and this gray quartzose material is streaked with veinlets and bands of pyrite and chalcopyrite. The original structure of the rock is only slightly preserved and Figs. 4 and 5 will show it to be replaced by aggregates of quartz grains of irregular size and shape.



A B C

FIG. 7.—SO-CALLED VARIOLITE, A MARGINAL PHASE OF THE QUARTZ-PORPHYRY, MADE UP OF SECONDARY QUARTZ AND ACTINOLITE. THE APPARENT PECULIAR SPHERULITIC STRUCTURE SHOWN IN THE PHOTOGRAPH IS A SECONDARY DEVELOPMENT. A, ACTINOLITE IN FIBROUS PATCHES BORDERING THE FINE QUARTZ AGGREGATES. B, FINE SECONDARY QUARTZ. C, COARSER-GRAINED NUCLEUS OF SECONDARY QUARTZ. CROSSED NICOLS. × 24.

Veinlets occur which include chlorite and the sulphides. The banded nature of the secondary quartz is also apparent. The interior structure of the quartz grains, their outline and irregular shape, features which are not brought out clearly except under crossed nicols, suggest the replacement of porphyritic material.

The photograph of a polished section of the ore, Fig. 6, shows the distribution of the pyrite and chalcopyrite. The minerals occur partly disseminated as small crystalline aggregates in the quartz mass and partly concentrated along veinlets and channels between the quartz grains. They appear to have been deposited simultaneously, though somewhat later than the period of silicification of the quartz-porphyry.

The chlorite occurs within and around the sulphides as patches, and possibly actinolite along with it as needles, occupying with the sulphides the spaces and channels between the quartz grains. The chlorite is believed to have formed from the ferromagnesians of the original rock, incident with its silicification, and to have had some influence on the precipitation of the sulphides. As a mineral, it is too universal an alteration product to have a more direct significance. The actinolite will be discussed later in connection with the associated rocks.

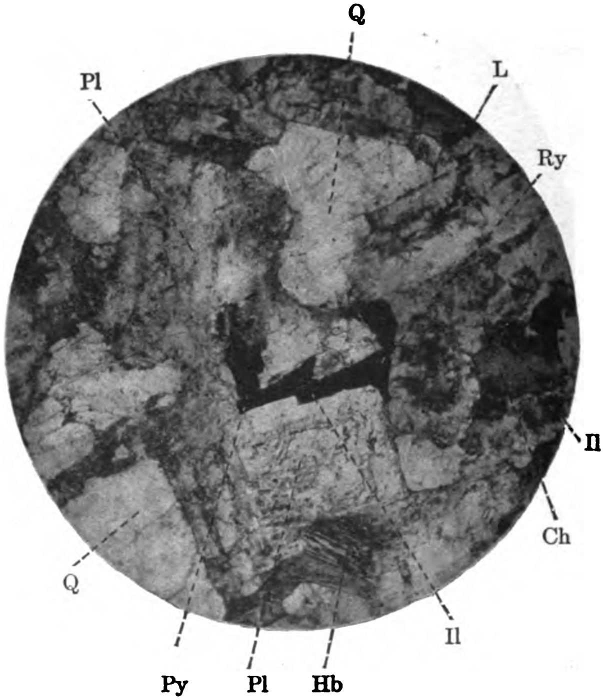



FIG. 8.—QUARTZ-DIORITE, TYPICALLY GRANITOID IN TEXTURE AND MADE UP OF INTERLOCKING GRAINS OF PLAGIOCLASE, QUARTZ, AND HORNBLENDE. ILMENITE AND APATITE ARE ACCESSORY MINERALS. PYRITE OCCURS IN THE ROCK AND MAY POSSIBLY BE PRIMARY, AS IT DOES NOT APPEAR TO CUT THE OTHER CRYSTALS. PLANE LIGHT. × 24.

A peculiar marginal phase of the quartz-porphyry has been developed at the northeast edge of the orebody near the dikes for which the name variolite, see Fig. 7, has been suggested on account of its peculiar spherulitic structure. But variolite is a variety of basaltic glass. The apparent spherulitic structure here is merely a peculiar secondary development of quartz and actinolite.

*The Quartz-Diorite*

The granite, on account of the predominance of plagioclase feldspar, is best termed a grano-diorite or quartz-diorite. It is a speckled, greenish-

gray and white rock showing abundant hornblende and plagioclase, with quartz, in large interlocking grains. It shows some slight fracturing, and zonal growth of plagioclose. The feldspar is basic in composition, probably labradorite, and comparatively free from alteration. Some carbonization is observable in the feldspar, and the hornblende is largely changed to chlorite. Ilmenite and apatite are accessory minerals, some of the apatites being 2 to 3 mm. in length. Pyrite occurs in the rock and may possibly be primary, since it does not appear to cut structures in the other minerals to any considerable extent (see Figs. 8, 9, and 10).

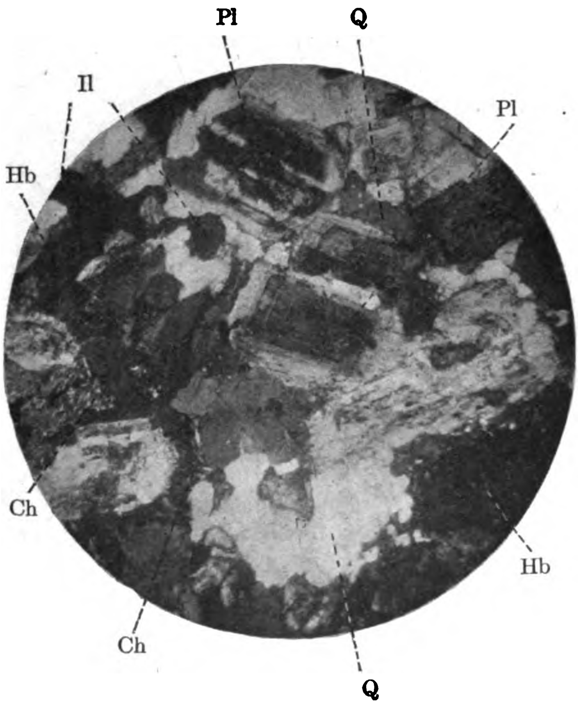



Fig. 9.—Same Rock as Fig. 8, Showing Zonal Growth of Plagioclase, Also the Development of Chlorite from Hornblende. Crossed Nicols. × 24.

## *The "Old Basic"*

This body is intrusive into the quartz-porphyry as laccoliths and numerous steep and inclined dikes. It is a dark gray felsitic rock, weathering on its surface to a whitish limonite coating. Though felsitic, it does have a few very small feldspar phenocrysts (see Figs. 11 and 12). The rock is composed of small criss-crossed columnar crystals of plagioclase with some larger irregular grains of the same mineral. All are surrounded by pale green actinolite and chlorite, derived from altered pyroxene, and small grains of magnetite are scattered through the mass.

Quartz is present in small grains but is very largely of secondary origin. Small and sparse crystals of pyrite occur associated with patches of chloritic material and actinolite. The heavy iron content in magnetite attests to the basic character of the rock, the magnetite having also resulted from the alteration of the pyroxene. Evidently the ferromagnesian minerals were unstable under conditions which did not affect the feldspars. The rock is an unusual type, though composed of common minerals, and may be called an andesine basalt.

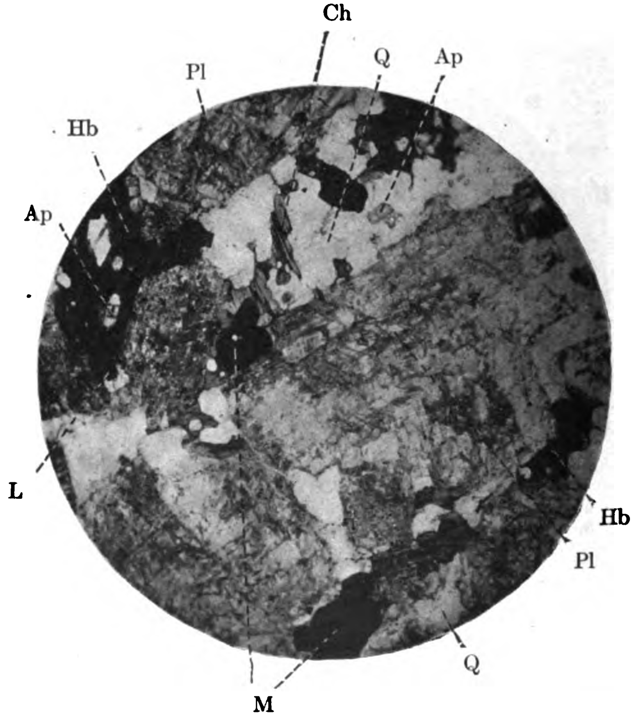



FIG. 10.—THIS PHOTOMICROGRAPH IS SIMILAR TO FIG. 8. IT SHOWS ABUNDANT PLAGIOCLASE, QUARTZ, AND HORNBLENDE. MAGNETITE, POSSIBLY ILMENITE, IS A PROMINENT ACCESSORY MINERAL, AND APATITE CRYSTALS OF FAIRLY GOOD SIZE ARE SCATTERED THROUGH THE ROCK. ALTERATION HAS AFFECTED MOST OF THE FELDSPARS TO SOME EXTENT, ATTACKING THEM ALONG STRUCTURE AND FRACTURE LINES. PLANE LIGHT. × 24.

Fig. 13 illustrates one of the thin sections under high power, and shows very clearly the small fibrous crystals of actinolite associated with the magnetite. As above stated, the actinolite, and probably most of the magnetite, is regarded as a metamorphic derivation from the original pyroxene in the rock, pseudomorphs of which are common through it. Pyrite also occurs in places penetrated by crystals of the actinolite, tending to the belief that this mineral, as well as the pyrite, is made up partly

of introduced material. However, the pyrite may have formed around the actinolite at a later stage without replacing it.

### *Stope Dike*

This rock, Fig. 14, belongs to the older dike series. It is dense, black and felsitic, very much resembling the "Old Basic," if indeed it is not the same rock. It includes an occasional fine patch of minute pyrite crystals, and weathers with a reddish iron stain. The structure of the

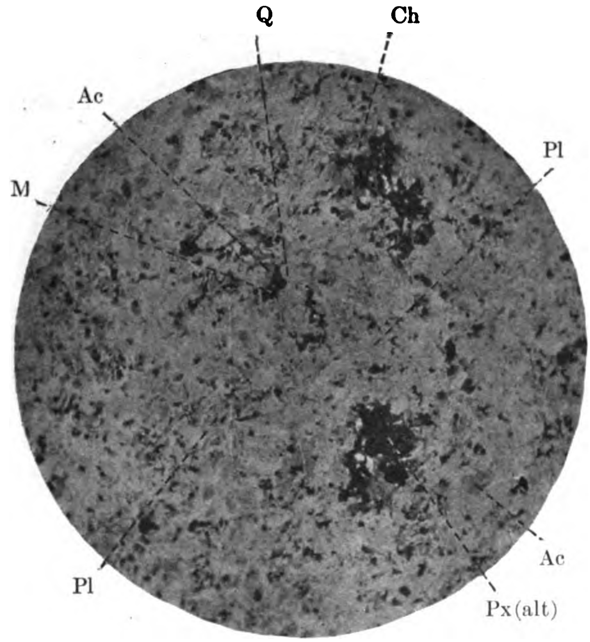



FIG. 11.—ANDESINE BASALT, SHOWING PYROXENE ALTERED TO SCATTERED AGGREGATES OF MAGNETITE WITH ACTINOLITE AND CHLORITE. FELDSPAR IS PROMINENT AND APPEARS TO HAVE BEEN THE FIRST MINERAL TO CRYSTALLIZE. ITS VARIETY IS ANDESINE. QUARTZ, LARGELY SECONDARY, IS PRESENT AS SMALL GRAINS THROUGHOUT THE MASS. THOUGH COMPOSED OF COMMON MINERALS, THIS IS AN UNUSUAL TYPE OF ROCK. PLANE LIGHT. × 24.

rock may be described as a network of small columnar crystals of plagioclase and fibers of actinolite, filled in with irregular grains of quartz and magnetite. A few fine fractures appear which are healed with chlorite. Pyroxene was an original mineral, which, together with the feldspar, has been largely altered to quartz and actinolite.

### *Dolerite Dikes*

The material of these dikes varies slightly in composition and greatly in texture, some being diabasic and others basaltic. All the material

is dark, massive and more or less felsitic. Fig. 15 shows the diabasic type. It is made up of rounded irregular grains of augite in a plexus of lath-shaped crystals of feldspar. The feldspar is plagioclase. The augite grains are small and irregular in outline, and appear to fill the spaces between the crossed crystals of plagioclase. Alteration has changed a considerable portion of it to chlorite and actinolite, part of which has replaced occasional lines and twinning bands in the plagioclase. Alteration has also changed some of the plagioclase to quartz. This

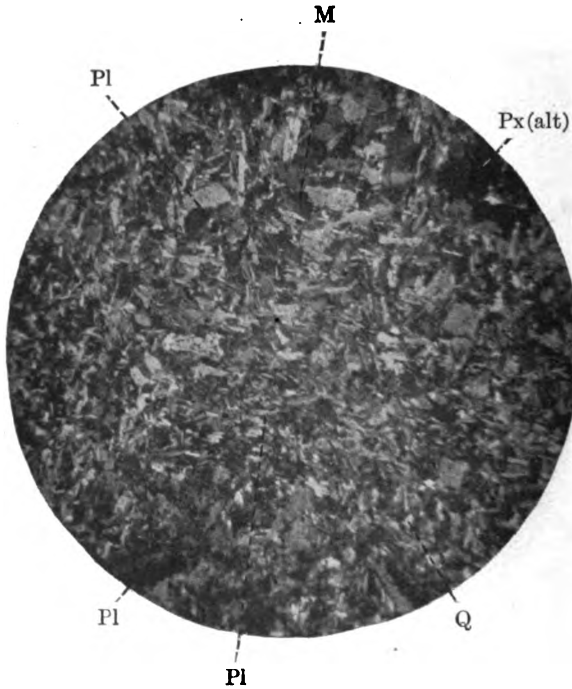



FIG. 12.—SAME ROCK AS IN FIG. 11. THE TEXTURE IS COARSELY FELSITIC TO PORPHYRITIC, AND THE CRISS-CROSSED ARRANGEMENT OF THE COLUMNAR PLAGIOCLASE CYSTALS IS APPARENT. THERE ARE A FEW PLAGIOCLASE PHENOCRYSTS VISIBLE, AND MAGNETITE IS DISTRIBUTED THROUGHOUT. CROSSED NICHOLS. × 24.

quartz seems to have been derived from both pyroxene and feldspar, being penetrated by fibrous actinolite crystals of simultaneous growth. However, it can not be said that all the quartz is secondary, for in the case of many interstitial grains, the traces of a replaced mineral are too slight to make it possible to say it existed.

The term Dolerite, as used on the map, while indicating rock of this general character, does not sufficiently emphasize the important structural feature shown in the arrangement of the feldspars in this particular specimen, and it would, therefore, be more properly called Diabase. The idiomorphism of the feldspars with respect to the pyroxene, as shown by

Fig. 15, is also a distinguishing characteristic of diabase, along with their columnar structure and crisscross arrangement.

Similarly, for textural reasons, the dolerite shown in Fig. 16 is more properly called basalt. This material is a more felsitic variety of the rock composing this group of dikes. It seems to possess the same general alteration effects except that the quartz, if present, is not visible, which might argue that the quartz of the other basic rocks is entirely secondary. In the ground-mass are many fine rods of plagioclase, but as a whole it

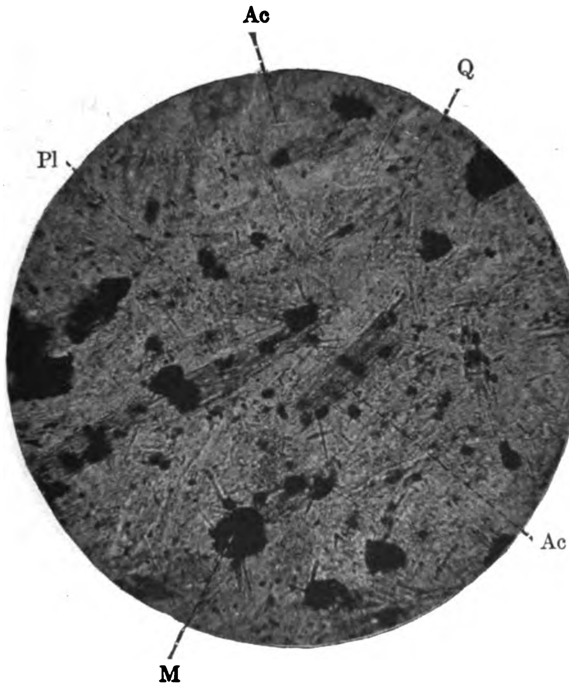



FIG. 13.—SAME ROCK AS FIGS. 11 AND 12. THE HIGHER POWER REVEALS SMALL FIBROUS CRYSTALS OF ACTINOLITE ASSOCIATED WITH THE MAGNETITE. THE QUARTZ AND PLAGIOCLASE ARE RATHER INDISTINCT. THE ACTINOLITE, AND PROBABLY MOST OF THE MAGNETITE, IS BELIEVED TO BE DERIVED FROM THE BREAKING DOWN OF ORIGINAL PYROXENE UNDER THE ACTION OF SILICEOUS MAGMATIC WATERS. PLANE LIGHT. × 257.

is finely felsitic and the rock a typical basalt. It contains some pyrite, which is generally taken to be an introduced substance.

The rock of the East Dike is a dark massive diabase, shown in Fig. 17. Quartz occurs in much the same relation to the crisscrossed columnar plagioclase as the pyroxene, and because this quartz may be partly primary the rock might well be called a quartz-diabase, which is a rather rare rock. In this original, the augite-pyroxene and plagioclase have been largely changed to chlorite, actinolite, and secondary quartz. The quartz and the unaltered plagioclase are penetrated to some extent by

fibers of actinolite. The scattered magnetite is probably also secondary, and pyrite is present.

The variation in texture of these dike rocks is mainly due to different rates of cooling, occasioned either by the influence of wall rock, which may have been hot or cold, or by the thickness of the dikes themselves. Their chemical composition must also have exerted a strong influence on the order of crystallization of the minerals, producing the tendency to idiomorphic development of the feldspar.

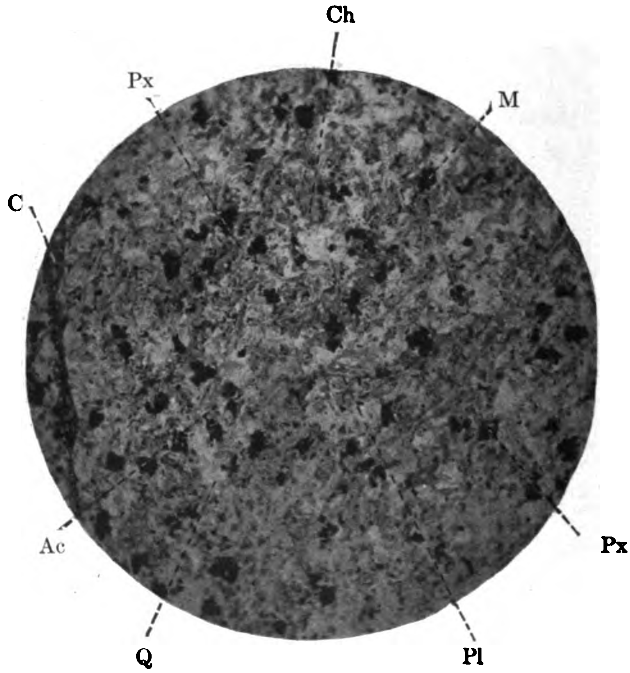



FIG. 14.—QUARTZ-BASALT, A FELSITIC COMPLEX OF PLAGIOCLASE AND PYROXENE WHICH HAS BEEN PARTLY ALTERED TO QUARTZ, ACTINOLITE, AND MAGNETITE, LEAVING ONLY A SMALL PART OF THE PLAGIOCLASE INTACT. PART OF THE QUARTZ, HOWEVER, APPEARS TO BE ORIGINAL. ABUNDANCE OF MAGNETITE, PART OF WHICH IS ALSO PRIMARY. SMALL CARBONATE VEIN. PLANE LIGHT. × 24.

## SUMMARY

It will be seen from the descriptions above that, aside from the quartz-porphyry, the principal rocks associated directly with the ore are basaltic in character. They contain quartz, part of which I judge to be primary and part secondary. Magnetite is widely distributed and appears to have resulted in large part from the breaking down of the original ferromagnesian minerals. The rocks have undergone great change, but do not appear to have been much affected by superficial weathering.

The strong development of actinolite in all the specimens indicates

either the alteration of pyroxene and plagioclase by solutions, or their change to a more stable mineral under conditions of regional metamorphism. The first is more probable, since there is not the slightest evidence of metamorphism in the surrounding grano-diorite. The pyrite in the basic rocks appears to be later than the actinolite, but as a whole is probably associated with the same period of alteration by hot solutions, the phenomena of expiring vulcanism. It is not known whether this pyrite contains gold or copper. In respect to the actinolite, it is not

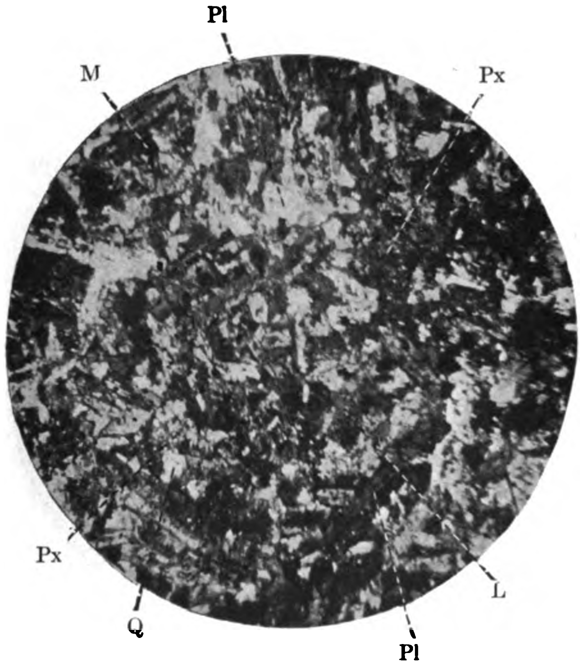



FIG. 15.—A DIABASE WITH THE CHARACTERISTIC TEXTURE. THE AUGITE GRAINS ARE SMALL AND IRREGULAR IN OUTLINE. MUCH OF THE PYROXENE HAS BEEN CHANGED TO CHLORITE AND SOME ACTINOLITE. TWINNING OF THE COLUMNAR PLAGIOCLASE CRYSTALS CAN BE SEEN. THE QUARTZ IS PROBABLY DERIVED FROM BOTH THE PYROXENE AND FELDSPAR, BEING PENETRATED BY FIBROUS CRYSTALS OF ACTINOLITE OF SIMULTANEOUS GROWTH. THE NAME DOLERITE IS AS GOOD A ONE AS ANY, BUT THE USAGE VARIES AMONG PETROGRAPHERS. CROSSED NICOLS. × 24.

possible to say that the alteration is due to direct igneous influence; but because actinolite is generally regarded as a high-temperature mineral there is the possibility of regarding it in much the same light as tourmaline, on the presence of which in the volcanic copper deposit at Braden, Chile, the primary character of the deposit was regarded as certain.

The quartz in general seems also to be a result of metamorphic action on the plagioclase and pyroxene, penetrated as it is by the contemporaneous crystals of actinolite. Part of the quartz in the dikes, however, must be primary, as the sharp outline of some of the interstitial grains

involve difficulty in conceiving the replaced original if these particular quartz grains were secondary. Hence the basic rocks might well be called quartz-basalts and quartz-diabases, which rocks are rather rare occurrences.

The grano-diorite or quartz-diorite intrusion shows some evidence of superficial weathering, but otherwise no alteration, and is considered a fairly fresh igneous rock. Pyrite is present in it, and the more altered character of the rock near the quartz-porphyries has led to the belief that its intrusion had something to do with the formation of the ores.

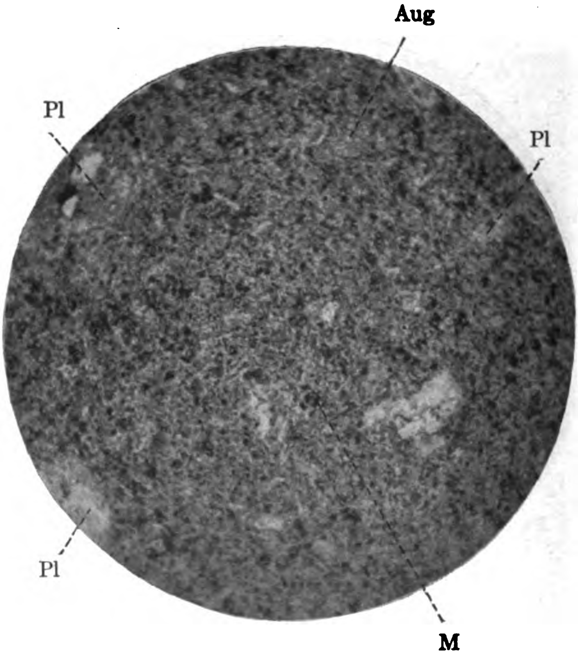



Fig. 16.—Basalt, a More Finely Felsitic Variety of the Foregoing Rocks, but Containing Small Scattered Phenocrysts of Plagioclase. In the Ground-mass there are Many Little Fine Rods of Plagioclase. Quartz, if Present, is not Visible. Plane Light. ×24.

The quartz-porphyries are thought to be such, rather than volcanic tuffs, because they contain no material foreign to the composition of a crystalline igneous rock, except the secondary quartz. By this quartz, however, these rocks are almost entirely replaced, the ground-mass completely so, and on this account their appearance under the microscope does somewhat resemble that of siliceous sinter. From the specimens examined and the mapping of this mass as quartz-porphyry, jasper, etc., its heterogeneous character, as of acid extrusive rock in general, is apparent. One of the specimens is a rhyolitic porphyry and another dacitic, and the structure of both, as already described, also suggests that the mass is an

ancient surface flow of large extent. The later intrusions of quartz-diorite and the basic rocks have further altered its local character.

In regard to the sequence of the different igneous masses, it seems most reasonable from this study to put the "Old Basic" later than the grano-diorite and of the same age as the Stope Dike. No very strong structural evidence to the contrary is shown on the map, as the two apparent intrusions of grano-diorite into the "Old Basic" on the western and southern borders of the mineralized area may just as well be interpreted as

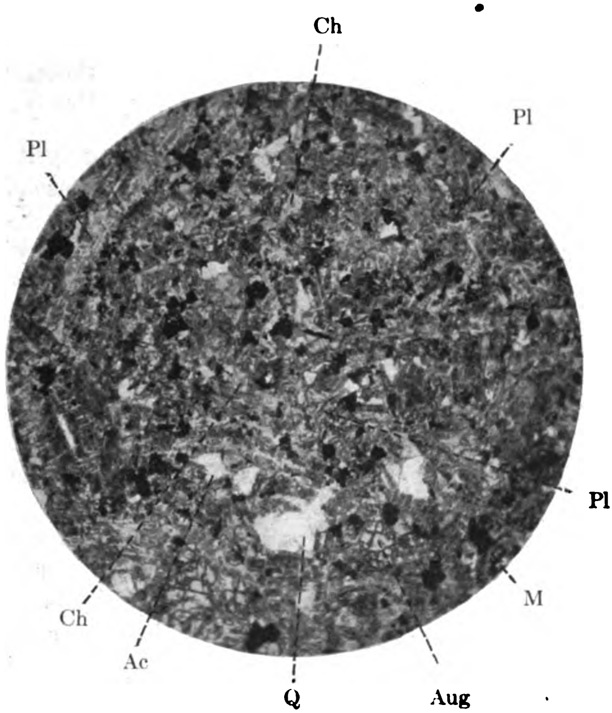



FIG. 17.—ORIGINAL QUARTZ-DIABASE IN WHICH THE AUGITE AND PLAGIOCLASE HAVE BEEN LARGELY CHANGED TO CHLORITE, ACTINOLITE, AND QUARTZ. THE DIABASIC TEXTURE IS EVIDENT, PYROXENE AND CHLORITE FORMING A FILLING BETWEEN THE CROSSED, COLUMNAR CRYSTALS OF PLAGIOCLASE. ACTINOLITE NEEDLES PENETRATE THE QUARTZ WHICH, HOWEVER, IN SUCH A BASIC ROCK, MUST BE LARGELY SECONDARY. PLANE LIGHT. × 24.

intrusions of "Old Basic" around spurs of the grano-diorite, and the petrographic evidence we have in addition is very strongly in favor of placing the "Old Basic" in the same age relation to the grano-diorite as the two systems of basic dikes which cut it. All the basic-rock types show too marked a mineralogical and textural similarity, even in the character of their alteration, to be divided into two periods of intrusion by another rock of opposite character, the grano-diorite. However, such occurrences are not unknown in the history of rock magmas.

According to this interpretation, the relative ages of the different masses are as follows:

| Formation | Rock Name |
|---|---|
| 9. Sedimentaries—Mesozoic. | |
| 8. Faults. | |
| 7. "East" Dike (Dolerite). | Quartz-diabase. |
| 6. "North" and "South" Dikes (Diabase). | Quartz-basalt. |
| 5. Andesite Dikes. | |
| 4. "Old Basic," "Stope," "East and West" and "Cross" Dikes. | Andesine- and quartz-basalt. |
| 3. "Flat" Dike.(?) | |
| 2. Granite. | Quartz-diorite. |
| 1. Porphyry. | Quartz-porphyry. |

The ore is of granular secondary quartz containing pyrite and chalcopyrite with chlorite and actinolite, a highly silicified phase of the quartz-porphyry. No evidence of a brecciated or crushed zone appears in any of the slides studied. The enrichment has been derived from magmatic solutions, and there is no reason to believe that the pyrite and chalcopyrite were deposited at different times. The more abundant occurrence of the sulphides along spaces or channels between the grains of quartz, and in bands or streaks through the gangue, seems to indicate that they were deposited shortly after the quartz replacement, rather than at the same time. The associated chlorite and actinolite seem to have been derived mainly from the ferromagnesians of the original quartz-porphyry during the period of replacement by quartz.

The chlorite seen in the ore is, as stated before, believed to have no important significance, but referring to Figs. 2 and 3, the strong development of actinolite in the surrounding quartz-porphyry is believed to be closely associated with the sulphide mineralization. If such is the case, the presence of this mineral in the basic dikes might imply that they had to do with the enriching solutions rather than the earlier grano-diorite intrusion, as actinolite is a metamorphic mineral formed either as a result of hot water action or of direct igneous metamorphism. It is not uncommon to find it at the contact of basic with quartzose rocks, as, for instance, near the periodotite dikes in the Franciscan sandstone of California, which formation itself is highly metamorphosed.

It might be noted here also that many gold deposits are associated with basic rocks, and the further structural fact can not escape attention that the ore deposit here occurs near the intersection of two systems of basic dikes. Hence, to say the mineralization was contemporaneous with these dikes seems a closer correlation of the data than to associate it with the earlier grano-diorite, for there are numberless other points in the quartz-porphyry near the grano-diorite contact where enrichment might have occurred and did not do so.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Manganese Ores of the Lafayette District, Minas Geraes, Brazil

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(New York Meeting, February, 1917)

## Introduction

For a number of years Russia, India and Brazil have outranked all other countries as producers of manganese ores. During the 5 years immediately preceding the European war, the average annual production of Russia was 844,000, of India 694,000 and of Brazil 200,000 long tons. Since the outbreak of the war there has been a considerable falling off in the Russian and Indian production, particularly in the former, whereas the production of Brazilian ores has been greatly increased, amounting in 1914 to over 250,000 tons and in 1915 to nearly 350,000, with conditions favorable for a still larger production in 1916. Manganese is one of the few industrially important metals that are not produced in the United States in quantity commensurate with our needs, so that we have been compelled to import annually about 300,000 tons, having a value of over $2,000,000, and these ores have been obtained from the three countries mentioned above. Their rank as contributors to our imports of manganese ores was India, Russia, Brazil; and in 1913, Brazil contributed only one-fifth. During the past 2 years Brazil has furnished rapidly increasing quantities to this country, and, with the falling off of imports from India and Russia, has become our principal foreign source. In 1914, Brazil furnished two-fifths of the imports and more than India or Russia, and in 1915 over nine-tenths. The manganese ores of Brazil are consequently of more than usual interest to us at this time.

## The Manganese-Ore Districts of Brazil

The manganese mining industry of Brazil dates from the year 1894, and since that year the total production has been over 3,000,000 tons. With the exception of a small quantity produced in the State of Bahia from deposits west of the City of Bahia, this output has come from the

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State of Minas Geraes and has been exported through the port of Rio de Janeiro. In Minas Geraes there are two distinct, though not widely separated, manganese districts, known generally as the Miguel-Burnier and the Queluz or Lafayette districts. The Miguel-Burnier district was the first to be developed and for a number of years was the more important producer; but, since the phenomenal development of the Morro da Mina mine, the Lafayette district has assumed far greater importance.

In order to contrast the modes of occurrence of the deposits in the two districts, a brief description of the Miguel-Burnier district will be given, in the nature of a summary of a previously published fuller account.[1] The deposits of this district extend as a narrow belt about 10 miles long, parallel to the Ouro Preto branch of the Estrada da Ferra Central do Bresil, and lie at the southern edge of the great iron-ore region of central Minas Geraes. The orebodies occur as lenses or beds intercalated in a series of sedimentary strata showing a rapid succession of itabirite, schist, calcareous schist, and limestone. Stratigraphically they are found in the upper part of the Itabira iron-formation and in the lower part of the overlying Piraçicaba schist,[2] formations of probable Algonkian age. The ores are very high-grade manganese oxides, chiefly a mixture of psilomelane and pyrolusite, averaging 50 per cent. manganese, 1 per cent. silica, and 0.03 to 0.05 per cent. phosphorus. The great drawback of the district is the fact that the beds are steeply dipping and narrow, rarely over 6 ft. in width, so that expensive underground mining must be resorted to, and the individual deposits are relatively small. Two views have been advanced to explain the origin of the ores. H. K. Scott, who has written the most complete account of them, says:[3] "Whatever may have been the original state of the manganese ore bed, there can be no doubt that in its present condition, and down to the level to which it has been worked, it is a residual deposit from which the other elements have been leached out." O. A. Derby, in a discussion of Scott's paper, endorsed this view and referred to the original state of the manganese ore beds as limestone with varying proportions of metallic carbonates and siliceous impurities.[4] On the other hand, Harder and Chamberlin say:[5] "From their occurrence it must be assumed that they are similar in origin to the associated rocks, that is, that they are original sedimentary

[1] Joseph T. Singewald, Jr., and Benjamin LeRoy Miller: High-Grade Manganese Ores of Brazil. *The Iron Age*, vol. 97, pp. 417–420 (1916).

[2] E. C. Harder and R. T. Chamberlin: Geology of Central Minas Geraes, Brazil. *Journal of Geology*, vol. 23, pp. 358–363 (1915).

E. C. Harder: Manganese Ores of Russia, India, Brazil and Chile. *Bulletin* No. 113, p. 788 (May, 1916).

[3] H. K. Scott: Manganese Ores of Brazil. *Journal of the Iron and Steel Institute*. vol. 57, pp. 188–189 (1900).

[4] *Idem*, p. 212.

[5] *Op. cit.*, p. 406.

deposits of manganese oxide." The same statement is made by Harder alone.[6] Our own feeling is that while one cannot positively state that the ores were not laid down in the form of manganese oxides as integral parts of a sedimentary series, their relations to the associated limestones are such as to make their interpretation as residual products of decomposition and replacement of manganiferous limestone the more probable explanation of their origin.

## The Lafayette District

Manganese deposits were discovered in the Lafayette district immediately after the inauguration of mining operations in the Miguel-Burnier district stimulated a search for manganese ores in Minas Geraes, but it was not until the year 1900 that the district became a regular producer, an output of 31,000 tons coming from the Piquery and São Gonçalo Mines in that year. This was increased to nearly 75,000 tons the following year, a production considerably in excess of that of the entire Miguel-Burnier district. In 1902, the Morro da Mina came in as a producer and firmly established the district as the chief manganese-producing district of Brazil.

Lafayette is a station, located at the edge of the town of Queluz, on the Estrada da Ferra Central do Bresil, about 32 km. south of Burnier and 462 km. from Rio de Janeiro. From the fact that the most important producing mines have been located in its vicinity the district has been generally referred to as the Lafayette district. The most important mine at present is the Morro da Mina, owned and operated by a Brazilian company known as the Companhia Morro da Mina, which has increased its production to 700 tons per day. It is located 7 km. northeast of Lafayette. On the same hill, a German company under the name Mineração de Agua Preta has been working the rubble ores to the east of the Morro da Mina ground. This company is producing at the rate of 2,000 tons per month and in the 6 or 7 years it has been operating has produced a total of 200,000 tons. On a small hill to the southeast of the Morro da Mina, the extension of that zone is being developed by a company known as the Companhia Queluz da Mina. The only other producing mine is the Cocuruto which lies about 40 km. southwest of Lafayette and is connected with the railroad at Christiano, a station 23 km. further south, by a 60-cm. gage line 40 km. in length. This mine has been operated about 7 years and is producing 3,000 tons monthly. It is owned by a Belgian company, the Société Anonyme de Manganese de Ouro Preto, which formerly worked the São Gonçalo Mine. After each had produced about 250,000 tons of ore, the São Gonçalo and Piquery mines were abandoned 10 years ago as worked out. They

[6] *Op. cit.*, p. 791.

were located about 15 km. northwest of Lafayette. The Piquer Mine is of particular interest in that it is the only one of which there is a geological description and it was there that Derby obtained the fir evidence of the original character of the manganese rock from which th ores were derived.

In addition to those mentioned, a number of other deposits hav been discovered and prospected to some extent, but apparently the result were not favorable enough to warrant further development as none o them became important producers. It is certain, however, that the dis trict has not been thoroughly prospected and there is every reason t expect that systematic exploration would discover deposits equal t those that have been found.

## *Geology of the Lafayette District*

The geology of the Lafayette district differs markedly from that o the Miguel-Burnier. It lies to the south of the area underlain by th

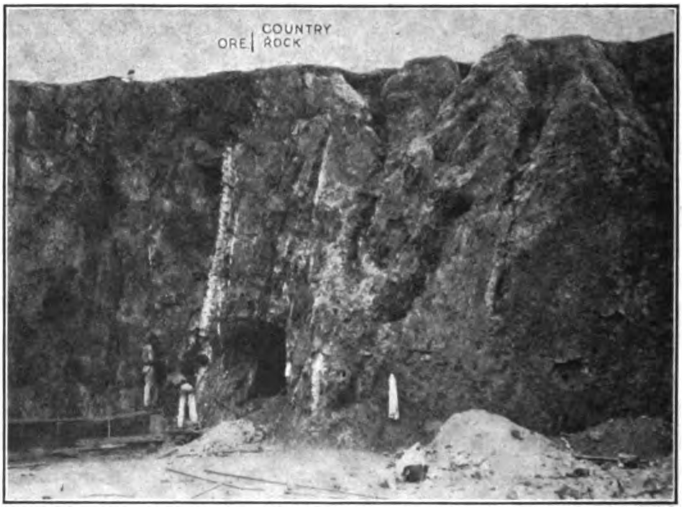


FIG. 1.—SHARP CONTACT OF MANGANESE ORE AND COUNTRY ROCK IN MORRO D MINA MINE.

great iron-bearing series, and its ores are found in the basement comple of supposed Archean age which underlies a large part of the state of Mina Geraes. The rocks making up this basement complex are chiefly granit and gneiss with which are associated amphibolite, and micaceous an quartzose schists. There are also small intrusions of diorite and gabbro mostly in the form of dikes. The granite seems to be intrusive into th gneiss and schist, but the relations between the schist and gneiss are no clear.

The manganese deposits occur as elongated masses of more or less lenticular shape within the rocks of the basement complex. As will be explained below, they represent residual products of decomposition of an original manganiferous rock made up of manganese carbonate and silicates. The immediate wall-rock of the deposits has likewise undergone decomposition, in many instances being nothing more than a clay in which the original rock texture is poorly preserved, so that it is usually difficult to determine its original character. In most cases, however, it seems to have been either gneiss or schist. The contact of wall-rock and ore generally appears quite sharp, as is shown in Fig. 1, but closer examination often reveals small nests and stringers of manganese oxide in the decomposed rock. There are also horses of equally decomposed rock and of the same character within the orebodies themselves.

### *The Piquery Mine*

The geological relations of the Piquery orebody have been described by O. A. Derby in two papers published in 1901 and 1908,[7] and the following account is abstracted from them.

"The Piquery orebody presents the appearance of a mass of secondary material, or gossan, resulting from the alteration of a vertical dike or vein, some 10 or 12 m. wide. . . . The ore is a hard spongy black oxide, apparently consisting for the most part of psilomelane but with an admixture of other oxides that frequently occur in beautiful crystallizations in the spongy cavities. . . . In the midst of the merchantable ore occur inconstant bands and patches of hard siliceous material with the appearance of a quartzite, but which on examination proves to be composed almost exclusively of a finely granular mass of ashy white manganese garnet. A complete series of alteration phases between perfectly typical garnet rock and merchantable ore can be readily selected, and there can be no doubt that the latter results from the decay and leaching of the former."

Derby describes three phases of the garnet rock that he observed in 1901:

"1. A very fine-grained, compact and finely jointed rock of bluish-gray color with partings lined with asbestus. Under the microscope the rock is seen to be composed almost exclusively of closely appressed idiomorphic grains of white garnet showing a clear border but with the center highly charged with a fine black opaque powder that appears to be graphite. . . .

"2. A dark brown rock heavily charged with manganese oxide and too friable to permit the preparation of microscopic sections is evidently of the same type but more completely decomposed. . . .

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[7] O. A. Derby: On the Manganese-Ore Deposits of the Queluz (Lafayette) District Minas Geraes, Brazil. *American Journal of Science*, Ser. 4, vol. 12, pp. 18–32 (July, 1901).

O. A. Derby: On the Original Type of the Manganese-Ore Deposits of the Queluz District, Minas Geraes, Brazil. *American Journal of Science*, Ser. 4, vol. 25, pp. 213–216 (March, 1908).

"3. A milky white rock which under the microscope is seen to be composed of about equal parts of garnet and quartz. . . . The quartz in a fine mosaic about the garnet grains and in minute refilled joints is almost certainly secondary."

Residues of manganese garnet were likewise found by Derby at the São Gonçalo, Morro da Mina, Agua Limpa and Barroso Mines. The last two mines were located about 10 km. southeast and 9 km. south of Lafayette respectively. As the result of his observations at these mines, Derby drew the following conclusions concerning the origin of the deposits:

"The orebodies of the Queluz district are residual deposits derived through decomposition and leaching from an original type or types of rock in which manganese garnet was the most constant and characteristic silicate element. . . . This type, which may appropriately be denominated *queluzite*, is more or less intimately associated at São Gonçalo, Morro da Mina and Barroso with decomposed schistose rocks that evidently contained an original manganese-bearing silicate and which from the absence of recognizable clastic elements and from other characteristics, so far as they can be made out, is presumed to have been an amphibolic schist representing a sheared basic eruptive. . . . In the Agua Limpa schist moreover, the manganese-bearing element is spessartine, as in the orebodies, thus giving greater plausibility to the hypothesis that the relation between these last and the above-mentioned rocks may be a genetic one. If thus related, the orebodies present strong analogies with those of magnetic titaniferous and chromic iron ores that are now generally considered as magmatic segregations in various types of eruptives, and, all things considered, this hypothesis seems the most plausible one for the manganese ores here discussed."

After the Piquery orebody was worked out and the original manganiferous rock was exposed in the bottom, Derby made a further study of it, the results of which are given in the 1908 paper. He found that the rock consists mainly of "a black, fine-grained, highly jointed and somewhat flaggy rock with the aspect of a limestone, with broad bands and patches of a more massive, yellowish-gray rock with the aspect of a quartzite." The latter is the garnet rock described in the previous paper, but which now turns out to be of secondary importance. The limestone-like rock on treatment with cold weak acid effervesces freely with an abundant separation of gelatinous silica and an insoluble residue containing spessartite and graphite. In places there is also a considerable admixture of rose-colored rhodonite in streaks and patches as the predominant component. Microscopic examination by Dr. Hussak showed the rocks to consist of manganese carbonate, tephroite, and spessartite with a small amount of rhodonite. The paper gives three analyses of these rocks, of which No. I was selected with reference to a supposed high carbonate content, No. II as having a lower carbonate content, and No. III was taken from the earlier paper and represents the garnet rock first found.

For the purpose of comparing the rocks represented by these analyses with the Morro da Mina rocks described below, their mineralogic composition has been calculated approximately on the basis of their described

*Analyses of Original Manganese Rock at Piquery*

| | I | II | III |
|---|---|---|---|
| $CO_2$ | 22.62 | 4.59 | |
| $SiO_2$ | 11.80 | 27.69 | 38.47 |
| MnO | 47.52 | 57.48 | 27.90 |
| $Al_2O_3$ | 7.50 | 1.41 | 21.07 |
| $Fe_2O_3$ | ..... | 2.48 | 7.38 |
| CaO | 3.76 | 1.82 | 4.70 |
| MgO | 6.27 | 4.60 | |
| | 99.47 | 100.05 | 99.52 |

mineralogic characteristics. The commercial ores average twice as high in alumina as in ferric iron and on this basis there is just enough ferric iron and alumina in analysis I to take care of the silica in forming spessartite. If all the $CO_2$ is calculated as rhodochrosite, there is a deficiency of 4 per cent. MnO. But there is more than enough CaO and MgO to take care of this deficiency, so that roughly this rock consisted of 32.6 parts of spessartite and 59 parts of manganese carbonate by weight, or 32 and 68 parts respectively by volume. If in analysis II, the ferric oxide and alumina are calculated as manganese garnet, the $CO_2$ as rhodochrosite and the remaining silica as tephroite, there is a deficiency of nearly 9 per cent. MnO which is slightly more than covered by the CaO and MgO. On this basis the rock consisted of 12 parts of rhodochrosite, 14.5 parts of spessartite and 75.2 parts of tephroite by weight, or 17, 17.5 and 65.5 parts respectively by volume. Analysis III represents a garnet rock with a little more alumina and about 4 per cent. more silica than is needed to take care of the MnO, CaO, MgO and iron considered as ferrous iron. On this basis it consisted of 65 parts of spessartite, 12.6 of grossularite, and 15.3 of almandite by weight. Actually, of course, the rock consisted of a garnet intermediate in composition between the three but more nearly approaching the composition of spessartite. These calculations show the same great variation in the mineralogic composition of the Piquery rock that is found at the Morro da Mina Mine, except that rhodonite does not appear to have the prominence it attains at the latter.

These new observations necessitated different conclusions and Derby in this second paper says of the deposits that they "seem to be due to the alteration of an original rock with predominant carbonate of manganese and tephroite rather than of spessartite and rhodonite as hitherto supposed." In regard to the genesis of the rock he is silent in this paper, nor does he apply the term *queluzite* to it.

### *The Morro da Mina Mine*

The Morro da Mina Mine is located on a hill 2 or 3 miles nor
town of Queluz, which has an elevation of 1,110 m. above sea l
rises to a height of 200 m. above the surrounding country. A

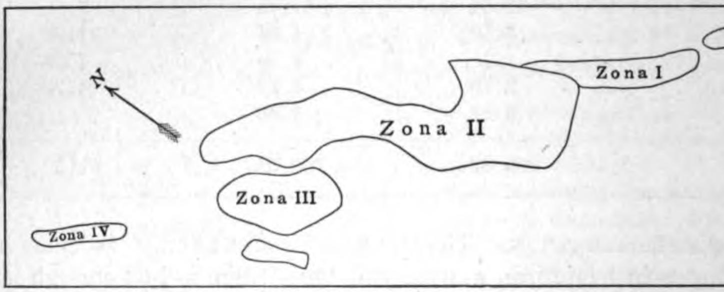


FIG. 2.—HORIZONTAL SECTION OF MORRO DA MINA OREBODIES AT LEV
BELOW SUMMIT OF HILL.

line of the railroad connects the mine with the Estrada da Ferra
do Bresil at Lafayette, so that the ore can be loaded into the ca

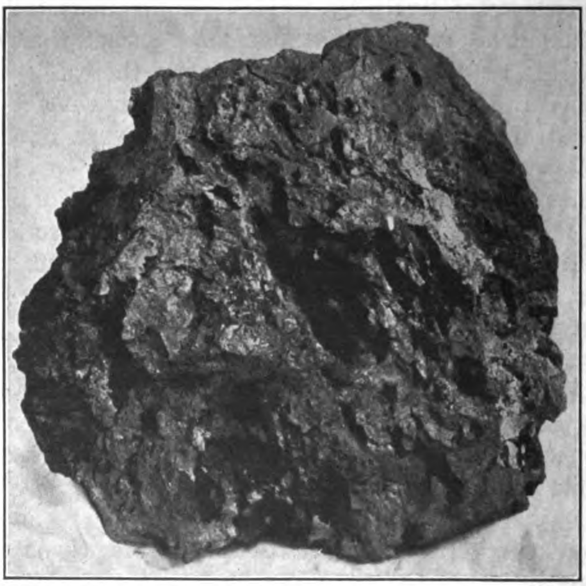

FIG. 3.—SOMEWHAT DRUSY MASSIVE PSILOMELANE, THE MOST COMMON
ORE AT THE MORRO DA MINA MINE.

mine and requires no further handling until transferred to ships f
tation at Rio de Janeiro. The mine presents one of the most
able manganese deposits in the world both in respect to size and

of the ore; in fact, the manager of the mine, J. de A. Lustosa, says it is the largest known deposit of high-grade manganese ore. Since it became an important producer in 1902, it has yielded a total of over 1,000,000 tons, and in 1915 its production was about 200,000 tons. Development work has proved an ore reserve of 10,000,000 tons.

The orebodies occur at the top and on the flanks of the hill as a series of more or less overlapping lenses extending in a direction N 35° W., with a vertical dip and a pitch of 45° to the southeast. The relative positions, shapes and sizes of the orebodies that have been developed as they occur on the level 55 m. below the top of the hill are shown on the mine map in Fig. 2. The four largest are known as Zona I, II, III and IV respectively and have maximum dimensions of 200 by 30 m., 420

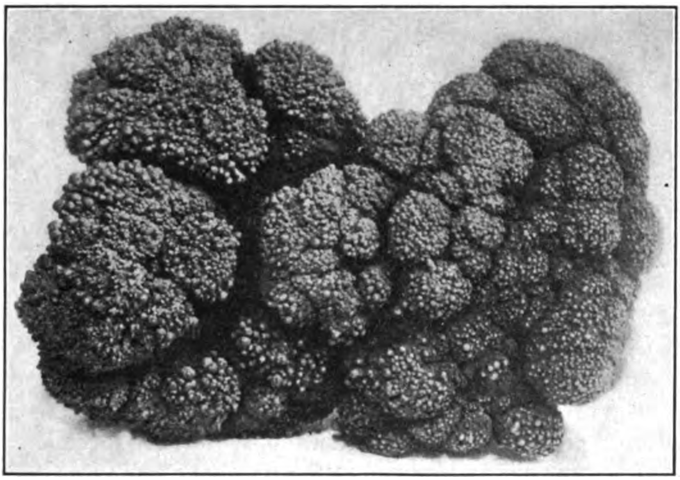

Fig. 4.—Cauliflower-like Cluster of Manganese Oxides Found Quite Frequently in the Morro da Mina Ores.

by 120 m., 140 by 70 m., and 100 by 20 m. In depth Zona I and Zona II have been cut by a development tunnel 130 m. below the summit of the hill, indicating that the ores extend at least that far down. In addition to the ores *in situ*, a large part of the hillside below the ore outcrops is covered with rubble ore derived from them.

The ore consists for the most part of psilomelane, which occurs in a variety of forms. Most commonly it is simply more or less drusy massive psilomelane as in Fig. 3, but mammillary, botryoidal and concretionary forms are abundant and frequently quite elaborate. The surface of some of these, except for the black color, reminds one of cauliflower. A typical specimen is illustrated by Fig. 4. Associated with the psilomelane is considerable manganite and pyrolusite which occur for the most part as cavity linings and fillings in the former. The manganite occurs lining the cavities both in radiating groups of acicular crys-

tals and in distinct prismatic crystals. The pyrolusite is often pseud morphic after manganite showing that it has in part been derived from The average composition of the ore as shipped is:

*Average Analysis of Morro da Mina Ore*

| | |
|---|---|
| Water at 100° C. | 2.50 per cent. |
| Volatile | 12.40 chiefly oxygen. |
| Insol. Residue | 3.46 |
| $Fe_2O_3$–$Al_2O_3$ | 8.75 alumina about twice ferric content. |
| Silica | 1.76 |
| P | 0.069 |
| S | Absent |
| Manganese | 50.47 |

It is of interest to compare this average with the following mean analyses of cargoes of manganese ores landed at Middlesborough, En land, during the years 1897 to 1906 as compiled by L. L. Fermor[8] f samples dried at 100° C.:

| | India | Russia |
|---|---|---|
| Mn | 50.86 | 49.58 |
| Iron | 6.31 | 0.83 |
| Silica | 5.71 | 10.17 |
| P | 0.127 | 0.161 |
| Alumina, etc. | 6.80 | 12.77 |

These figures indicate that the three countries produce very high-gra ore with the advantage in favor of the Brazilian ores, particularly ov the Russian, as regards silica and phosphorus.

The ore is mined by hand for the most part in open cuts. A numb of tunnels and adits have been driven, but these were intended either f development purposes or to connect the various open cuts to give acce to the loading platforms and bins at the railroad. The method in vog is to strip the orebody of such overburden as it may carry, and then the ore is mined screen it over iron screens with 0.8-in. square opening The oversize is the merchantable ore. The screenings constituting per cent. of the crude ore, carry 34 to 35 per cent. Mn and are bei stored apart from the waste to be beneficiated at some future time. T ore *in situ* furnishes two-thirds of the present output, and one-third derived from workings in the rubble ores.

## The Original Manganese Rock

Though decomposition of the original manganese rock has extend to considerable depth in places in the Morro da Mina Mine, to at lea 130 m. as demonstrated in the exploratory tunnel previously mentione

[8] L. L. Fermor: Manganese-Ore Deposits of India. *Memoirs of the Geologi Survey of India*, vol. 37, Pt. III, p. 518 (1909).

portions of the rock have escaped alteration and are well exposed in the mine workings. A large mass of the rock is actually exposed at the surface at the southwest corner of the outcrop of Zona II. It is characterized by a predominance of the silicates, and especially garnet, and for that reason has not succumbed to the processes of weathering. The abrupt transition from this rock to ore is shown in Fig. 5, and it is only the presence of numerous stringers, tongues and patches of manganese oxide penetrating the rock close to the ore that makes clear their genetic relations. More interesting and instructive exposures are found in the 130-m. tunnel. This tunnel crosscuts in a westerly direction the decomposed country rock for 120 m. to Zona I, then cuts diagonally across it in good ore for 150 m., and on emerging penetrates 50 m. of the manganese


FIG. 5.—SHARP CONTACT BETWEEN THE MANGANESE ORE AND THE ORIGINAL MANGANESE ROCK CHARACTERIZED BY AN ABUNDANCE OF MANGANESE GARNET.

rock to the ore of Zona II. After again cutting 50 m. of ore, a small horse of the manganese rock is encountered with ore on the opposite side. The four contacts of ore and rock exposed in this tunnel showed the same rapid change from one to the other with only a narrow transition zone marked by stringers and tongues of the black manganese oxide penetrating the manganese rock.

Most of this rock has the appearance of a fine-grained dark gray crystalline limestone and is easily scratched with the point of the pick. Its specific gravity, however, is considerably above that of limestone. Here and there are brown patches and streaks with a violet tinge that consist of massive garnet, and there are frequently spots and stringers of pink rhodonite that at once attract attention. A closer examination

reveals the presence of sufficient light pink silicate in much of the mater to give a pinkish tone to its dominant gray color.

Examination of thin sections shows that the essential constituer are manganese carbonate, spessartite, rhodonite and tephroite. Taki the average of all the rock, the manganese carbonate is the most abunda mineral and the tephroite the least abundant. The spessartite is mo widespread in its distribution but probably not much in excess of t rhodonite in actual quantity. The relative quantity of the differe minerals varies most widely, so that some of the rock consists alm entirely of one of the minerals while some has them present in alm equal quantity, with the exception of the tephroite, which, in the s tions examined, was never present in more than subordinate amou In this respect the rock differs from the Piquery rock described by Der in which tephroite was prominent and rhodonite a subordinate consti ent. Which of the two minerals was formed in a given case depend undoubtedly on the relative amount of silica available, a low silica co tent giving rise to the formation of the orthosilicate tephroite and higher silica content to the metasilicate rhodonite.

A thin section of a specimen most closely resembling crystalline lin stone consists almost entirely of anhedral grains of manganese carbona averaging between 0.16 and 0.25 mm. in diameter. That the carbona is essentially manganese carbonate rather than calcium or magnesiu carbonate is shown by the specific gravity of the rock, which is 3.5 The manganese carbonate is swarming with flakes of graphite rangi in size from 0.03 to 0.08 mm. and somewhat less abundantly with sm garnets ranging from 0.01 to 0.03 mm. in diameter. In the entire sli there are only a few small areas containing rhodonite or tephroi The minute garnets in this slide differ from the larger garnets in the mo siliceous phases of the rock in being perfectly clear, whereas the latter a characterized by the presence of numerous minute black inclusions su as described by Derby as graphite in the rocks studied by him. Anoth specimen of more siliceous looking rock with a pinkish tone and strea of rhodonite shows in thin section manganese carbonate, garnet, rhod nite and tephroite in abundance. The constituents average in size fro 0.08 to 0.16 mm. with a few larger crystals of rhodonite. As is general the case, the garnets are pronouncedly euhedral while the other minera are anhedral. A specimen of particularly siliceous looking rock havi a specific gravity of 3.8 consists for the most part of rhodonite with whi is associated considerable euhedral garnet. The rock itself is light brov in color with a distinct pinkish tone. In addition to the silicates the is also an appreciable amount of the carbonate present. The garne range in size from 0.10 to 0.15 mm. and the rhodonite crystals avera about two or three times as large. Other thin sections show essential similar features.

The manner in which this rock undergoes alteration to manganese oxides is very interesting. A specimen undergoing oxidation is illustrated

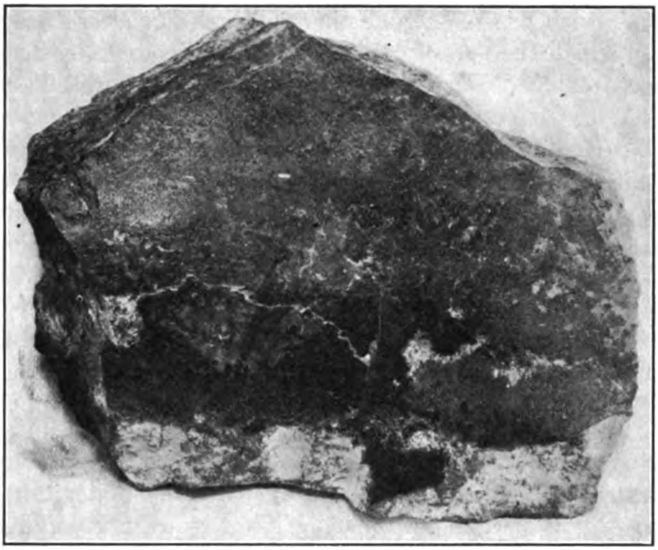

FIG. 6.—POLISHED SURFACE OF THE ORIGINAL MANGANESE ROCK IN PART ALTERED TO OXIDES OF MANGANESE. (Black areas and patches represent the manganese oxides.)

in Fig. 6. As the oxidizing solutions penetrate the rock they first break down the manganese carbonate, tephroite and rhodonite and leave

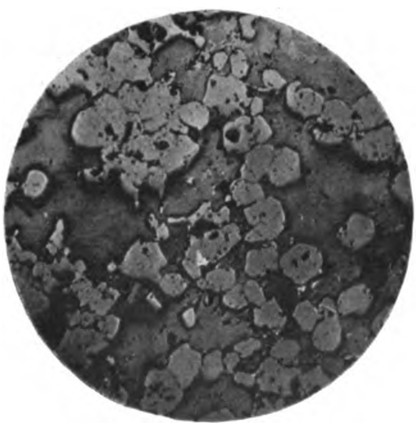

FIG. 7.—RESIDUAL GARNET GRAINS IN A GROUNDMASS OF MANGANESE ORE RESULTING FROM THE OXIDATION OF THE MANGANESE ROCK.

behind embedded in the manganese oxides the grains of garnet. That is, oxidation does not run ahead decomposing the manganese carbonate first

and then as a more advanced stage tephroite and rhodonite successively, as one might expect; but the three minerals are replaced simultaneously at the very front of the advance of oxidation. A magnified polished surface of rock which has undergone this first stage of alteration is shown in Fig. 7, in which the garnet crystals are seen disseminated in a matrix of manganese oxides. Such material from this mine was described by Derby in his first paper and both minerals considered primary elements of the rock. In his second paper he doubted this first interpretation and was more inclined to regard the manganese oxide a residue of the alteration of original carbonate and silicate minerals with retention of the spessartite. The evidence of the thin sections makes clear that the latter is the true explanation. As this material is further worked on by the oxidizing solutions, the garnet begins to succumb around the periphery of the grains and along fractures until finally it too has been replaced by the manganese oxides.

## Genesis of the Lafayette Type of Deposit

The manganese deposits of the Lafayette district inevitably call t one's mind the Indian deposits which likewise consist of manganes oxides associated with manganese silicate rocks, and as we shall have occasion to refer to them in the discussion of their genesis, it will be helpfu to summarize at the outset certain of the salient features of the Indian deposits as described by L. L. Fermor.[9] There are two main types o manganese deposits in India, associated respectively with the *kodurit* and *gondite* rocks. The former are found in the Vizagapatam distric on the east coast, the latter in the Central Provinces and other parts o India. Typical kodurite is composed of potash feldspar, manganes garnet and apatite, with or without pyroxene. From the mineralogica and chemical composition of the rock and its geologic relations, Fermo concludes that kodurite is an igneous rock. It was subjected to alteration under oxidizing conditions and gave rise to the manganese ores. H finds that the manganese garnet is the most stable mineral and is ofter left behind in a matrix of psilomelane. To the gondite rocks he give an entirely different interpretation. He thinks they were originally deposited as sediments and the manganese which they contain a chemical sediments, most probably in the form of oxide, though perhap to a limited extent as carbonate. These sediments were later subjected to intense dynamo-metamorphism, and where they consisted of relatively pure chemical sediments of manganese were converted into compac psilomelane and braunite. Where alumina and silica had also been deposited, spessartite and rhodonite were formed, and if there was an excess of silica it was crystallized as quartz. The resultant rock is calle

[9] *Op. cit.*

gondite. Typical gondite is a very fine-grained rock consisting of tiny round grains of spessartite set in a mosaic of quartz. Where alumina is deficient rhodonite is found. The ores associated with the gondite rocks, Fermor believes are the result of combined decomposition and replacement of them by waters containing $CO_2$ and O, but that this action took place at considerable depth and that the carbon dioxide was a portion of that liberated in the metamorphism of the rocks of the region and the oxygen a portion of that liberated in the conversion of the original manganiferous sediments into the manganese silicate rocks. A small proportion of softish and more or less porous ore he attributes to the later surficial alteration of the gondite.

An explanation of the genesis of the Lafayette manganese deposits involves two problems: First, the genesis of the original manganese rock; second, the alteration of it to manganese oxides, that is, to the manganese ores.

As the evidence is beyond question in regard to the second stage in the formation of these ores, that problem will be discussed first. The manner of occurrence of the orebodies, as for instance the change in depth of the Piquery ores into the manganese rock, the abundant evidence of the alteration near the surface of the manganese rock into ore and the microscopic evidence of the same phenomenon, point unmistakably to the derivation of the ores from the manganese rock under conditions of weathering in the zone of oxidation. That is, the ores were formed under conditions similar to those that gave rise to the ores of the kodurite rocks of India and not as Fermor thinks most of the ores in the gondite series were formed. The chemistry of the alteration of the Brazilian manganese rock is simple compared with that of the kodurite. The principal constituents to be removed are the $CO_2$ of the manganese carbonate and the $SiO_2$ of the silicates. Examples of the efficacy of meteoric waters for that purpose are so abundant and generally recognized that this part of the process hardly requires elaboration. The universal alteration of iron carbonates to oxides and hydroxides in the zone of oxidation and the enormous amount of silica removed from the iron silicates in the formation of the Lake Superior iron ores are illustrations of the same chemical actions on similar compounds of a chemically closely related element. In the kodurite rocks the removal of considerable alumina was also necessary and the chemistry of this was the only step that Fermor found at all difficult to explain. But even the very slight solubility of alumina in ordinary dilute meteoric waters he thought would suffice for its ultimate removal. This difficulty, however, hardly enters in our problem. Except in the case of the local phases of garnet rock, which has anyhow to a large extent resisted alteration, the alumina content of the original rock is no higher, and the few available analyses would indicate actually a little lower, than that of the ores; and, further-

more, the total amount involved in either is not over 5 per cent. quently the problem of the removal of silica does not confront

The ores have been described as somewhat porous and drus no means to such an extent as would be called for if they mere sented the residual product of leaching of manganese carbo silicates. Nor is there evidence to indicate that their pres compact form is due to shrinkage in volume or compression of th manganese oxides; but the evidence both megascopic and mi shows that the decomposing solutions deposited manganese o ume for volume for the silica and carbon dioxide removed. waters encountering such large quantities of manganese carbo

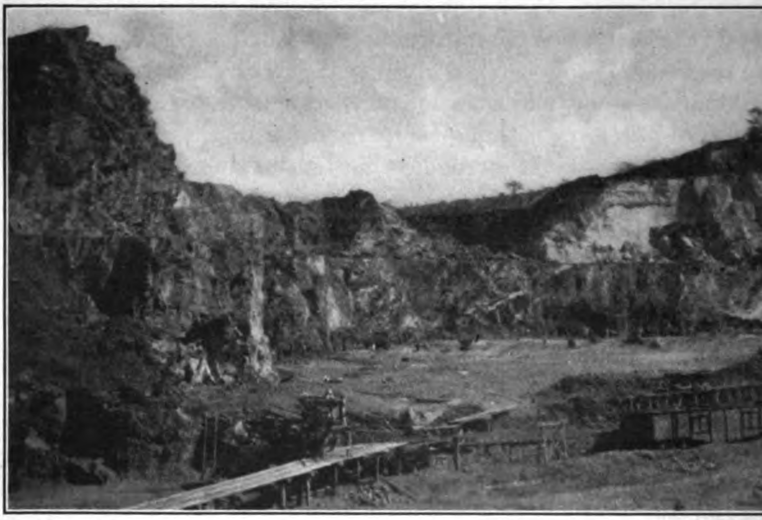

FIG. 8.—VIEW OF SOUTH END OF OPEN CUT OF THE MORRO DA MINA M TO BE THE LARGEST KNOWN DEPOSIT OF HIGH-GRADE MANGANESE

tainly took a great deal of it into solution as manganese bic which would be deposited at other points as the oxide; and in the deposition of manganese oxide accompanied the leaching of constituents of the rock. The alteration of the manganese roc was, therefore, one of simultaneous leaching and addition, res a relatively compact mass of the oxides. The present drusy of some of the ore is clearly the result of subsequent action of solu redeposition by meteoric waters upon the manganese oxides f this way.

The problem of the origin of the manganese rock is not so easy One is at once confronted with the fact that our knowledge meager concerning the exact nature of the rocks with which it is a

and its geologic relations to them, a full knowledge of which is essential to a final solution. The chemical, mineralogic and petrographic characters of the rock, however, do rule out some explanations, and point strongly to another. Further, the geologic relations that are reasonably well established are not at variance with this explanation but in a measure support it. That is, the sum of all evidence points to an analogy of this rock with the gondite of India. We believe that the rock is the product of dynamo-metamorphism of manganese sediments deposited in the form of manganese carbonate with varying but considerable quantities of silica and varying but smaller quantities of alumina. These sediments differed, therefore, initially from the gondite sediments in averaging much lower in silica, and by the deposition of the manganese in the form of carbonate instead of oxide; and this initial difference in composition accounts for the present difference in mineralogy of the two rocks.

The greater resistance to decomposition of the garnet in the manganese rock at the Piquery mine led Derby in 1901 to the erroneous conclusion that it consisted essentially of manganese garnet and to establish the rock type *queluzite* to which he attributed an igneous origin. Later, in 1908, he found that the garnet rock was but a subordinate phase of an entirely different type of rock which is essentially the same as that which we found abundantly at the Morro da Mina Mine. Attention has been called to the fact that in this later paper he does not apply the term queluzite to this rock and ventures no opinion as to its genesis. Though he did not specifically retract his earlier statement, the presumption is that, in the light of the discovery that the original rock was entirely different from what he had supposed, he abandoned his interpretation of its origin. In the meantime, L. L. Fermor,[10] on the basis of Derby's first paper, drew an analogy between this rock and his kodurite as regards their genesis, though calling attention to their dissimilarity in composition. A persistence of this same view is manifested in the statement by Harder and Chamberlin[11] in their discussion of these ores that "manganese ores associated with igneous rocks, such as those described above, occur abundantly in India—" and the statement by Harder[12] that "The relation of the manganese rock to the enclosing crystalline rocks has not been definitely determined; it may be interlayered with the gneiss or crystalline schist, or perhaps intrusive into them." The chemical and mineralogic composition of this rock as now established at both Piquery and Morro da Mina effectually precludes an igneous origin for it and demonstrates the presence of an original carbonate rock.

Even before Derby's second paper appeared, a view at variance with it

[10] *Op. cit.*, pp. 273–274.
[11] *Loc. cit.*, p. 405.
[12] *Loc. cit.*, p. 790.

was expressed by his associate E. Hussak.[13] He regarded the spessar rock described by Derby together with a banded spessartite-rhodo rock, which he mentions as occurring on the periphery of the Piqu orebody, the products of contact metamorphism by the eruptive gr of impure manganese carbonate sediments. The postulation of the d vation of the silicates from manganese carbonate in advance of the covery of the presence of large amounts of carbonate is interest Beyschlag, Krusch and Vogt[14] are inclined to carry Hussak's ideas a further and suggest that the manganese may have been contribute part of the process of contact metamorphism. That this was not the is proved by the fact that the carbonate still present, constituting a l percentage of the rock, is the manganese carbonate; and that conseque the original carbonate was a manganese carbonate. There remain decide between the view of Hussak that the rock is the result of con metamorphism of sedimentary beds of impure manganese carbonate our own that it is a product of dynamo-metamorphism of such b

In the first place, the country rock of the ore deposits is not ev where gneiss, but is in some cases schist, and these schists are admi to be metamorphosed sediments. Nor are all of the gneisses necessa igneous, though it has been generally held that they are such for the n part. Consequently the presumption is against assuming an ign contact for all of the deposits. In the second place, the texture appearance of the silicates is not that of typical skarn minerals, or p ucts of contact metamorphism; but it is that of silicates crystallize a carbonate rock that has been subjected to regional metamorph The reply might be made that the whole region has been subjecte dynamo-metamorphism subsequent to the contact metamorphism, b is equally true that the texture of the rock is not what one would expe result from a skarn rock subjected to regional metamorphism. On other hand, we cannot escape the conclusion, demanded by both hypo ses, that there first existed impure manganese carbonate, and it is obv that the rocks have undergone regional metamorphism. The inevit result would be a rock such as we have. Our explanation is, theref in harmony with the available evidence and makes the manganese of the Lafayette district genetically identical with the gondite of In the mineralogic difference of the two rocks being due to their initial ference in chemical composition.

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[13] Eugen Hussak: Ueber Atopit aus den Manganerzgruben von Miguel Bur Minas Geraes, Brasilien. *Centralblatt für Mineralogie, Geologie und Paläontologie,* pp. 240–245.

Eugen Hussak: "Über die Manganerzlager Brasiliens. *Zeitschrift für prakt Geologie,* vol. 14, pp. 237–239 (1906).

[14] Beyschlag, Krusch, Vogt: *Die Lagerstätten der nutzbaren Mineralien Gesteine,* vol. 2, p. 596 (1913).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 39 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Recent Geologic Developments on the Mesabi Iron Range, Minnesota

BY J. F. WOLFF, E. M.*, DULUTH, MINN.

(New York Meeting, February, 1917)

DURING the past 4 or 5 years, much has been added to the detailed geologic knowledge of the Mesabi Range. This has not been in the direction of discovery of any new fundamental facts, but of detailed study, subdivision and correlation of different parts of the whole formation and of individual orebodies. Prior to this time, mining engineers in the district were so engaged with the commercial interests of exploring and developing orebodies that close geologic study and subdivision was done in only a few instances. The demand for refinements of work in this direction has caused extensive structural subdivision and correlation to be done in all exploration work during the past 4 years by the engineers of the Oliver Iron Mining Co. Such work has become a commercial necessity rather than a scientific refinement, and at the present time is being extended to all parts of the Range.

In the summer of 1914, the author of this paper wrote a series of articles on the orebodies and special features of the Mesabi Range, which was published in the *Engineering and Mining Journal*, issues of July 17 to Aug. 7, 1915. Since that time studies of subdivision of the iron-formation have been extended considerably. The principle feature of this paper is the presentation of the subdivisions of the iron-formation. To this is added a discussion of the relations of the orebodies to the gentle folding and fracturing of the formation, and special features of the Range.

The outline will be as follows:

I. General geology.

II. Subdivision of Biwabik iron-formation.

III. Relation of orebodies to folding and fracturing of the iron-formation.

IV. Special features.

Acknowledgment is due to W. J. Olcott, President, and John Uno Sebenius, General Mining Engineer, of the Oliver Iron Mining Co., for permission to use the information presented in this paper.

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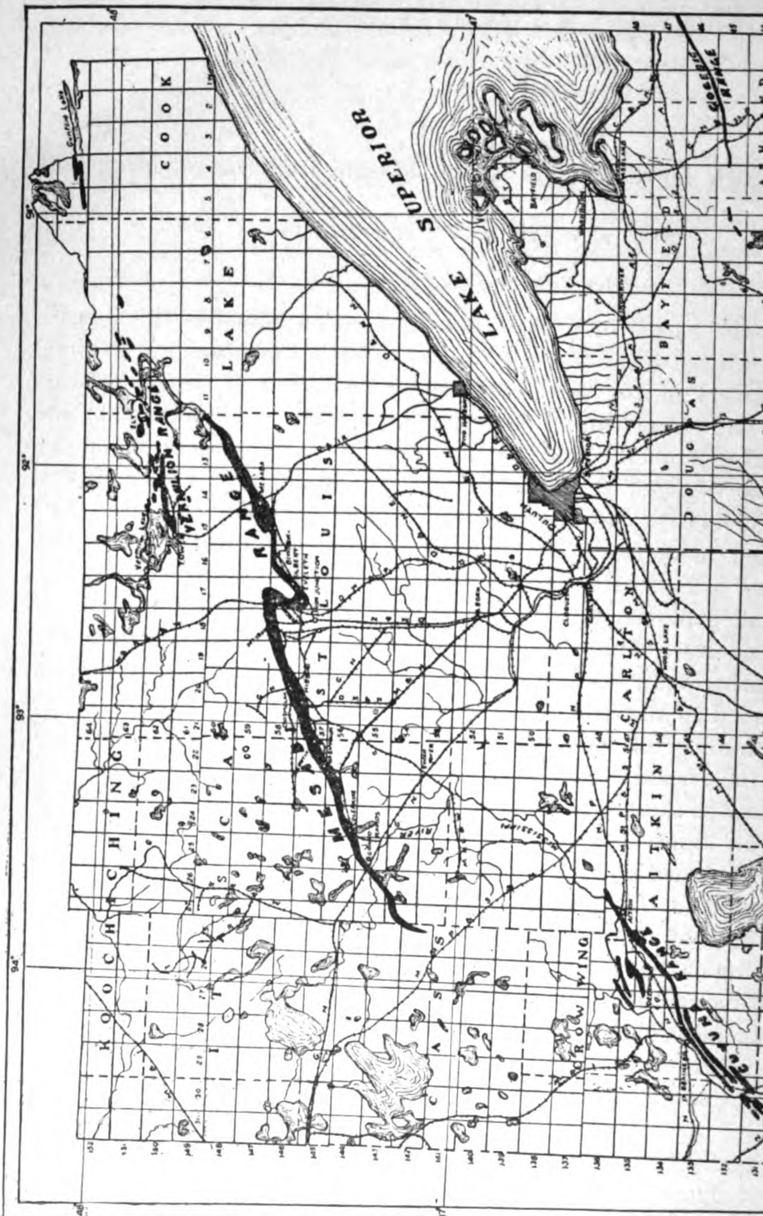
LAKE SUPERIOR
COOK
LAKE
VERMILION RANGE
MESABI RANGE
ST. LOUIS
ITASCA
KOOCHICHING
CASS
CROW WING
AITKIN
CARLTON
DOUGLAS
BAYFIELD
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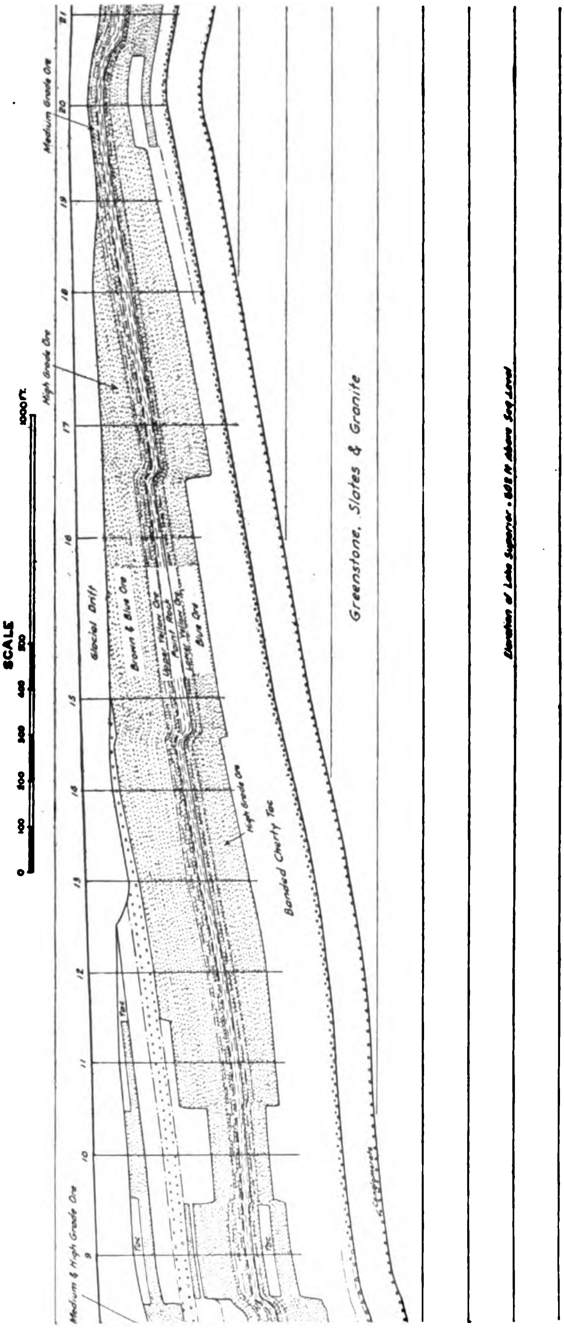
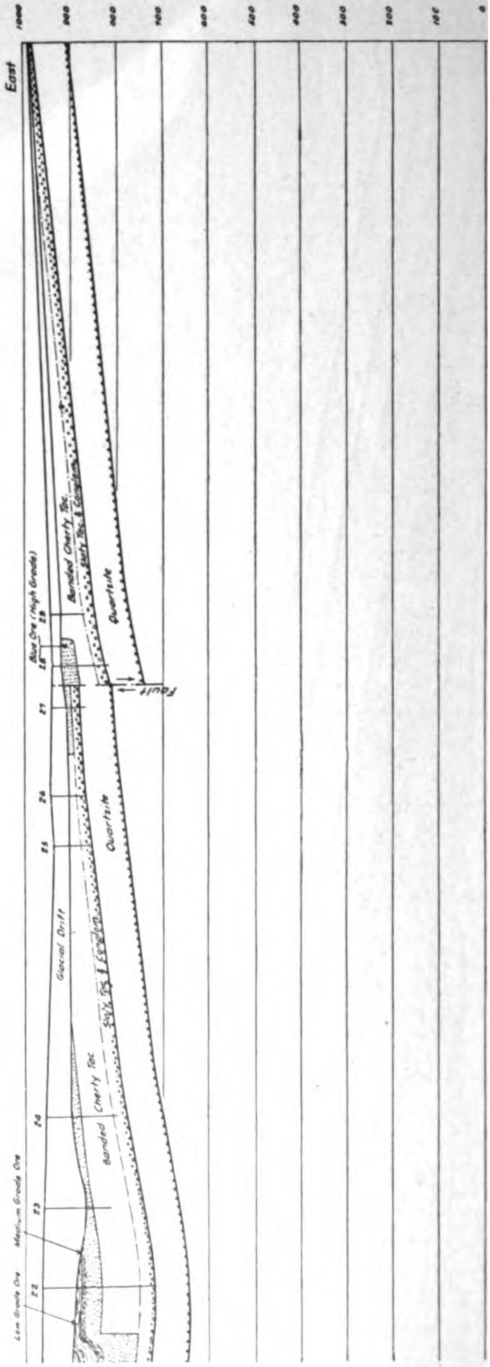



FIG. 6.—(*Continued.*)


East
Blue Ore (High Grade)
Banded Cherty Tac.
Slaty Tac. & Conglom.
Quartzite
Fault
Glacial Drift
Banded Cherty Tac
Low Grade Ore
Medium Grade Ore

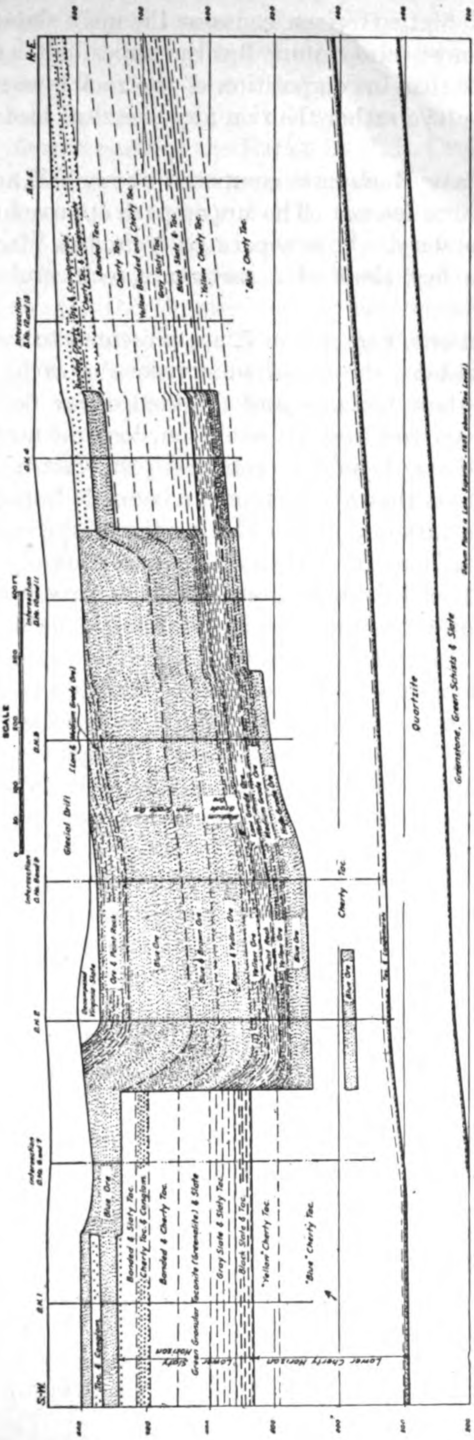



FIG. 7.—TRANSVERSE CROSS-SECTION OF A TROUGH OREBODY SHOWING ORE DERIVED FROM THE DIFFERENT HORIZONS AND TROUGH STRUCTURE OF OREBODY.

half of the Lower Slaty Horizon contains the most slate, th
ft. or so being almost pure slate. But the upper half is slat
in structure rather than in composition of the rock layers.
sions are made by eye rather than on any chemical basis or
examination.

The Upper Slaty Horizon is composed about half and
layers and greenalite lenses. The upper part approaching
slate is the more slaty. It is separated from the Virgini
layer 10 or more feet thick of a carbonate rock, probabl
limestone.

Two cross-sections, Figs. 6 and 7, are presented to show
four main subdivisions, the minor subdivisions of each, the
the orebodies to these horizons and the kind of ore derive

Fig. 6 shows an east-west cross-section, looking north,
Adams, Hull, Nelson, Leonidas orebodies just west of Ev
the best cross-section that has been or can be made from pre
tions, so far as the author knows. The location of the sectio
the accompanying map, Fig. 11, and is approximately at
to the strike. It is developed from drill-hole classificatio
seen, but practically the entire section of ore is being devel
open-pit and underground workings. Because this cro
taken along the longitudinal axis of the ore trough, the trou
does not show. In the area of drill holes 6 to 8 inclusive, a
trough is tributary to the east-west trough through which t
tion is taken, and the trough structure is apparent. The ge
of the entire series can be noted. The principal warping is at
to the strike, however.

The relations between the different horizons and the
ores are so evident from the cross-section that a detailed des
not be given. In the area of drill holes 2 to 15 inclusive, th
the Lower Slaty and Upper Cherty Horizons is so badly
classification is very difficult.

A few features may call for comment. The black slate
the base of the Lower Slaty Horizon is the so-called "Ir
Slate, which makes the characteristic paint rock layer of
Mesabi orebody. The orebodies first developed on the
located in that part of the formation shown between drill hol
therefore, it is easily seen that the typical orebody had five la
and lower blue ore layers, upper and lower yellow ore layers,
mediate paint rock layer.

Interbedded in the Upper Cherty Horizon and the top
Slaty Horizon are distinct conglomerates. At the base o
Slaty Horizon there is also a fine interbedded conglomerate.
Cherty conglomerate is continuous from the far eastern part

to the western part. It was recognized in drill cores only a few years ago by W. H. Crago, head of the exploration division of the Mining Engineering Department of the Oliver Iron Mining Co. All of the earlier drill cores have not been reëxamined for it as yet, but it has been correlated extensively in different parts of the Range by F. B. Cronk and the writer. The cross-section in Fig. 6 shows a maximum development of interbedded conglomerate. It is not as thick either on the western or eastern end of the range. On the eastern part it is distinctly developed in both Upper Cherty and upper part of Lower Slaty Horizons, but in the west central part of the Range only the Upper Cherty conglomerate has been recognized, and it is quite thin. This section (Fig. 6) shows practically a maximum thickness of iron-formation. The Lower Slaty Horizon is abnormally thick (260 ft.) whereas the average thickness is only about 150 ft. In the central and western part of the Range, the Upper Cherty Horizon is exceptionally thick, the slaty layers of the upper part of the Lower Slaty Horizon being replaced by cherty and banded taconite. Evidently, more muds were deposited with the iron-formation in the east central part of the district than in the central and western parts of the district. Perhaps the underlying rocks outcropping to the north are accountable for this. From Mt. Iron east to Aurora, large areas of greenstones and slates lie to the north of the iron-formation. If the original shore line of the sea in which the iron-formation was laid down occupied approximately its present position, the weathering and erosion of these rocks contributed muds and argillaceous sediments to the sea water contemporaneous with the deposition of iron.

Fig. 7 shows a cross-section across a typical trough orebody in the Virginia district. The Upper Cherty and Slaty Horizons have been eroded from the sides of the rockwalls of the orebody, but all horizons from the Virginia slate down are represented in the orebody. The typical trough structure is well shown.

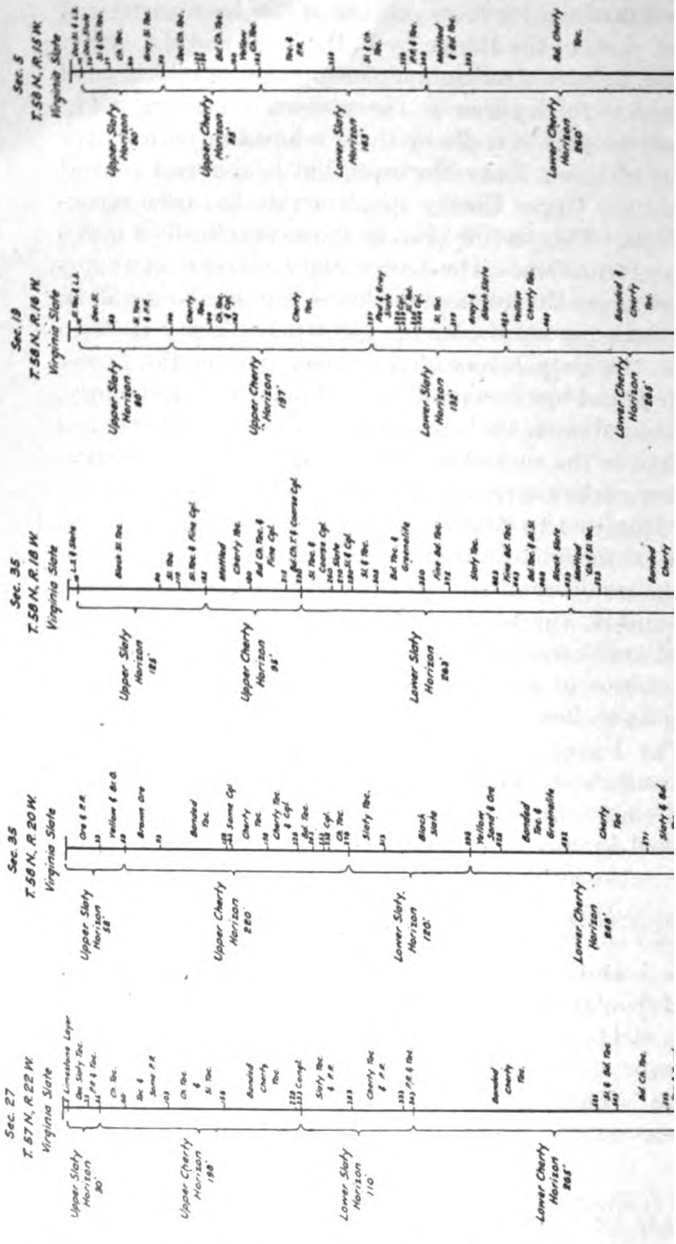
### *Records and Subdivisions of Drill Holes in Different Districts*

Fig. 8 shows records and subdivisions of drill holes in the Nashwauk, Hibbing, Virginia, McKinley and Aurora districts, comprising the territory from the west central to the eastern part of the range. These records all show the same major divisions, though varying in the minor subdivisions and dimensions. The interbedded conglomerate in the Upper Cherty Horizon is persistent in all the districts.

### *Ores Derived from the Different Horizons*

The characteristic ore derived from the Lower Cherty Horizon is a coarse "blue" high-grade ore. It contains practically no paint rock

seams. About 30 or 40 ft. of the top of this horizon i
mottled cherty taconite (originally a greenalite rock




slate seams in its upper measures, and this layer makes
ore of medium grade. At the base of the Lower Ch

above the basal conglomerate is a layer of fine slaty taconite (see Fig. 5) which makes a yellow ocherous ore. With the exception of these top and bottom layers, the Lower Cherty Horizon makes a "blue" high-grade ore. As used here, high-grade ore means ore averaging above 59 per cent. dry iron, medium-grade averaging 55 to 56 per cent. dry iron, and low-grade averaging about 50 per cent. dry iron.

The characteristic physical feature of ores from the Lower Slaty Horizon is their finely banded and slaty texture. As previously indicated, the black slate at the base makes the paint rock layer so prominent in every typical orebody. This material is not a commercial ore. It is highly aluminous and contains 20 per cent. or more of moisture. The gray slate and greenalite and slate above this black slate (Fig. 7) make a medium-grade yellow and brown ore, the yellow ore being quite aluminous. The banded-cherty and banded-slaty taconite of the top of the Lower Slaty Horizon make a very fine-grained blue and brown ore of high grade. It was this kind of ore which was so objectionable to furnace men because of excessive fines, in the early days of the Mesabi Range exploitation.

The Upper Cherty Horizon makes a high-grade coarse blue ore, in all of the large well-concentrated bodies. It is indistinguishable in texture from the blue ore of the Lower Cherty Horizon.

In some orebodies toward the south side of the formation, such as the Morton Mine in the Hibbing district, the Duncan in the Chisholm district, the Leonidas in the Eveleth district and the Schley and Hobart in the Gilbert district, the ore in this horizon is somewhat sandy and cherty, due in part to incomplete concentration and in part to secondary silica deposited by ground waters.

The Upper Slaty Horizon generally makes a low-grade non-merchantable ore. The one known exception to this is shown in Fig. 7, in which orebody most of this ore probably will be merchantable. This orebody, however, is one of the most highly concentrated on the Range. In most orebodies which have this Upper Slaty member, the material from it is a soft plastic paint rock with decomposed chert and greenalite layers. It resembles very much the so-called "Intermediate" paint rock layer.

From the cross-section, Fig. 6, it is evident that all orebodies will not contain all of these horizons or layers. The unit of land subdivision is a 40-acre tract and many mines occupy one or a part of one such tract. If a mine is located near the quartzite outcrop, most of the upper horizons will be eroded away, and as the mine location approaches the Virginia slate outcrop more of the upper layers will be found in the orebody.

## Relation of Orebodies to Folding and Fracturing of the Iron-Formation

In the *Engineering and Mining Journal* articles above referred to, the author stated that the data then at hand indicated that the orebodies

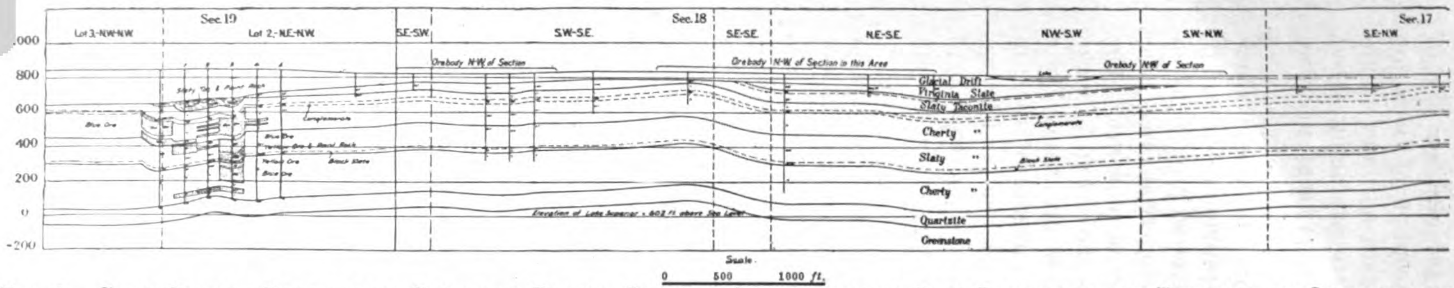
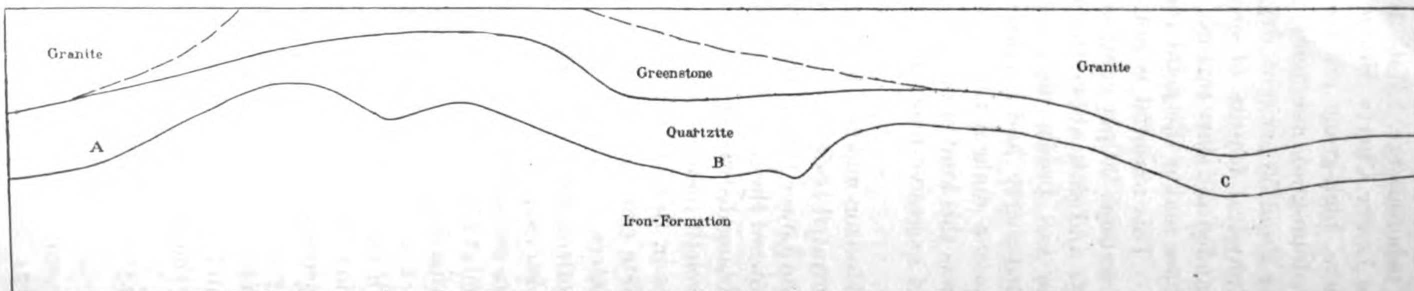



Fig. 9.—Cross-Section Parallel to Strike of Range, Showing Subdivisions of Iron Formation and Relation of Orebodies to General Structure.




Part of Geological Map showing Quartzite Outcrop

Longitudinal Cross-Section

have formed in places where gentle folding and warping of the iron-formation had fractured it considerably, allowing easy access and circulation of ground-waters. Evidence along this line has been assembled since the publication of those articles and in every case where the exploration data is complete enough it has been found that the orebodies occur where the whole formation has been warped. In the eastern part of the district (Virginia and eastward) the orebodies are on the crests of gentle anticlines. Fig. 9 shows a cross-section parallel to the strike of the formation in T. 58 N., R. 16 W., location of which is shown on the map in Fig. 11. This section is taken well to the south side of the formation and only one orebody reaches it. But the locations of other orebodies northwest of it are shown on the cross-section, and in each case where the exploration data is complete enough to show it, the orebody is located on the axis of an anticlinal flexure or combined anticline and syncline.

In the central part of the Range, great broad flexures rather than merely localized ones seem to have determined the locations of the orebodies. The formation was very generally cracked up and the broad structural basins directed the flow of underground waters. Fig. 10 shows a structural cross-section taken midway between Virginia slate and quartzite outcrops approximately parallel to the strike, through the Hibbing-Chisholm districts. A part of the quartzite outcrop to the northwest is also shown. Three prominent anticlines, *A*, *B* and *C*, with two intervening synclines, are shown. The cross-section is taken about a mile south of the quartzite outcrop. It has been published previously in the *Engineering and Mining Journal*, Aug. 7, 1915. It shows that the orebodies occur quite continuously over both crests of anticlines and troughs of synclines, indicating a very general fracturing of the formation, vigorous circulation of ground-water and consequent complete alteration and concentration of iron-formation. These major flexures can be determined only by such correlation and drawings as are shown in the two cross-sections, Figs. 9 and 10, but the minor flexures within the larger ones often can be observed in the field.

## Special Features

Special features of the Mesabi Range may be of interest and deserving of inclusion in this paper.

### Faults

The two principal faults known to date on the Mesabi Range are those known as the Biwabik and Alpena faults, both of which were described in the *Engineering and Mining Journal*, July 24, 1915. Mention is made of them here only because they have been followed further since that time. Fig. 11 shows the location of both. The Biwabik

fault has been traced to the NW of SE, Sec. 5, 58–15, where it practical disappears. It is a hinge-type gravity fault, the south side of whi has been depressed. The greatest throw is at its west end at the Biwab Mine, Lot 4, Sec. 2 and Lot 1, Sec. 3, Tp. 58 N., R. 16 W., where t vertical displacement exceeds 200 ft. The underlying greenstone faulted up against the ore, though the faulting probably occurred pri to the formation of the ore. Fig. 12 is a cross-section of the Alpena o body north of Virginia, showing the largest fault known on the Rang The location of the cross-section is shown on Fig. 11. As indicated, t strike of the fault is approximately north and south. It is a fault thrust-fold, the probable development of which is shown by Fig. 1

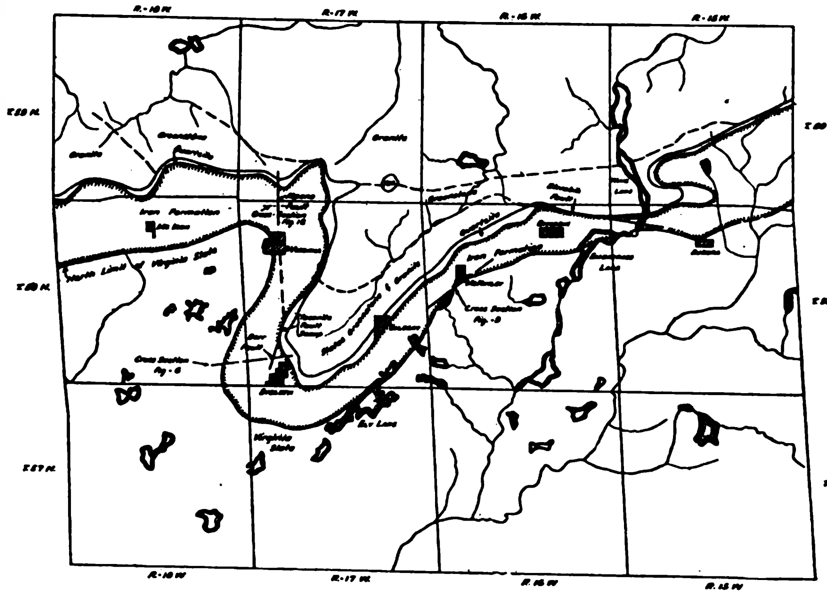



FIG. 11.—PLAT SHOWING AREA OF EAST CENTRAL MESABI RANGE.

The genesis of the fault is directly connected with the gentle uplift crustal warping which produced the large Z-shaped bend in the Ran known locally as the Virginia "horn." This was discussed also in t *Engineering and Mining Journal*, July 24, 1915. It is repeated he because since that series, a fault (shown on the east end of cross-secti in Fig. 6) has been discovered by correlation of drill holes. The locati of this fault, which may be called the Dorr fault from the property which it is located, is shown on the map, Fig. 11, and is of the same ty (reverse or thrust) as the Alpena fault. It is almost directly south the Alpena. Between the two and about ¾ mile north of the Dorr fau is a taconite bluff, the east side of which is a very steep wall, undoubted a fault scarp. It is of the same type as the Dorr and Alpena fault

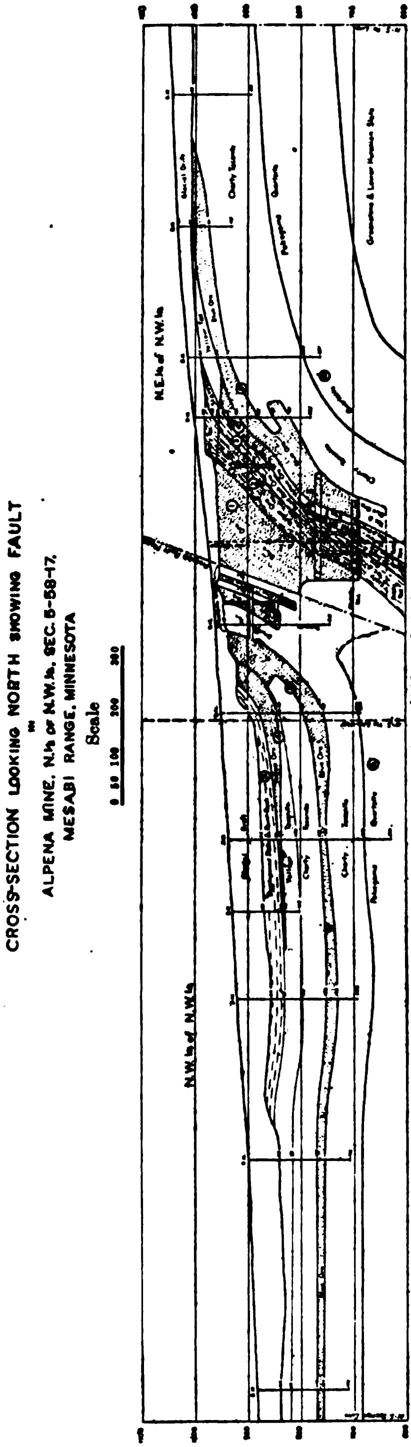



FIG. 12.—SHOWING ALPENA THRUST-FAULT.

Although we have as yet no complete exploration data, it appears tha the Alpena and Dorr faults and the intervening escarpment are on continuous fault produced by the crustal movements which caused th Virginia "horn." This probable connection is indicated on Fig. 11 However, it is possible that the Alpena fault is entirely separate from th Dorr fault.

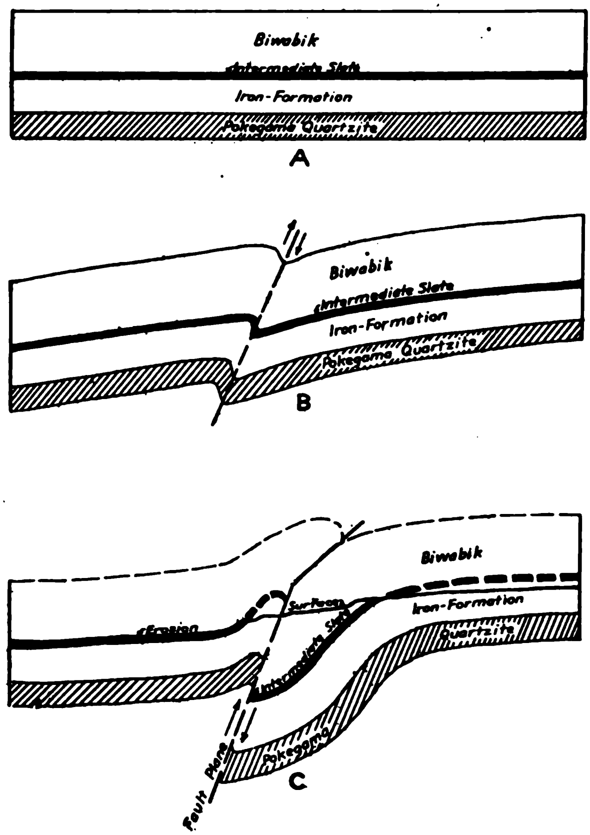



FIG. 13.—SHOWING PROBABLE DEVELOPMENT OF THE ALPENA THRUST-FAULT.

### *Thickness and Dip of the Iron Formation*

The average thickness and dip of the whole iron-formation in differen parts of the Range may be of interest. So far as they have been deter mined from the present very extensive explorations, they are shown i the accompanying table, tabulated by Ranges.

It will be noted that west of the Alpena fault the dips are quite lo and uniform, while east of it they are higher and rather irregular. Fro this evidence and that of the Alpena-Dorr fault, it appears that the dis turbance caused probably by the intrusion of the great mass of Dulut

| | Average Dip, Degrees | Thickness, Feet |
|---|---|---|
| T. 56 N.—R. 24 W | 6 | 520 |
| T. 56 N.—R. 23 W | 3 to 8 | 475 (approx.) |
| T. 57 N.—R. 22 W | 5 | 615 |
| T. 58 N.—R. 21 W.<br>T. 57 N.—R. 21 W. | 5 | 590 |
| T. 58 N.—R. 20 W.<br>T. 57 N.—R. 20 W. | 4½ | 650 |
| T. 58 N.—R. 19 W | 3 to 7 | 660 |
| T. 58 N.—R. 18 W | 3 | 630 west of Alpena fault |
| T. 58 N.—R. 18 W.<br>T. 58 N.—R. 17 W. | 6 to 9 | 755 east of Alpena fault |
| T. 57 N.—R. 17 W | 10 to 15 | |
| T. 58 N.—R. 16 W | 12 | 650 |
| T. 58 N.—R. 15 W | 25 | 530 (Secs. 5 and 6) |

gabbro between Lake Superior and the eastern Mesabi range tilted the eastern part of the Range considerably; that the Alpena fault took up most of the crustal shortening in the Upper Huronian series due to the compression from the east; and that, because of the faulting, the sedimentary series west of it was relatively undisturbed.

The thickness is seen to be greatest in the vicinity of Eveleth and thinnest on the extreme eastern and western ends. The average of the figures given is about 620 ft.

### *Volumetric Shrinkage in Solid Taconite*

Recent structural study has shown that the alteration which the original iron formation has suffered has produced a volumetric shrinkage even where the iron formation still retains its solid condition. However, in such solid taconite, hard layers of high-grade iron ore are interbanded with chert layers, the whole mass being firmly cemented together. Drill holes in fresh unaltered iron-formation alongside of holes in altered but still solid taconite show this fact to exist. Such a case is shown in comparing the depths of formation below the Lower Slaty Horizon in holes 1 and 2 with those in holes 4 and 5 in the orebody in Sec. 19, Fig. 9; also holes 1 and 2, Fig. 6.

This fact is a very important one to the mining engineer in the district in working out the structure of orebodies. The knowledge of it will aid him to establish more accurately the correct structure of certain ore layers. It explains the apparent lack of parallelism between different members of the iron-formation, which cross-sections often indicate unless this fact is known and applied.

### *Post-Algonkian Conglomerates on the Upper Huronian Series*

In the *Engineering and Mining Journal*, July 17, 1915, the writer called attention to at least one conglomerate, and possibly two, on the top of the

Biwabik iron-formation, older than the so-called Cretaceous conglomerate, beds and remnants of beds of which are found on most of the orebodies west of Eveleth. Since that time he has observed in two mines a conglomerate capping the orebody, containing large boulders which themselves were composed of the typical Cretaceous conglomerate. These conglomerates were overlain by layers of plastic shale and mud. Undoubtedly these are local lakebed or stream bed conglomerates and muds, intermediate in age between Cretaceous and Pleistocene.

### *Newly Discovered Fossil Remains in the Cretaceous (?) Shale*

The conglomerate and shale found on top of many orebodies were correlated by the U. S. Geological Survey as Cretaceous from the fossil remains of bivalves and some teeth and vertebræ of the Mosasaur found in the shale. During the past year, in the Canisteo pit at Coleraine, one almost perfect fossil and part of others of Ammonites were found. A reproduction is shown in Fig. 14. As nearly as the writer and associates can correlate these specimens, they belong in the Jurassic and not in the Cretaceous. They show no such complicated sutures nor such ornamentation as is characteristic of the Cretaceous Ammonites. The Specimen *A*, Fig. 14, is about 15 in. in diameter and 3 in. thick. On Specimen *B*, which fits into the cast *C*, and *C* also, can be seen minute veins, but no evidence of complex sutures. It is possible, of course, that this Jurassic form lived over into the Cretaceous, but the numerous Saur remains the writer has seen could hardly have belonged to a reptile as large as the Mosasaur. They seem more fitted to a smaller creature, perhaps not over 10 ft. long, rather than to one 30 or 40 ft. long. May they not belong to a Jurassic saur, one less fully developed than the Cretaceous Mosasaur? Is it not possible, or even probable, that the conglomerates and shales of the Mesabi Range which have been called Cretaceous really are of Jurassic age and formerly were connected with the extensively developed Jurassic shales of Northwestern Minnesota? Northern Minnesota is such a well developed peneplain that it is difficult to imagine a Jurassic ocean covering any part of it without reaching at least the foothills of the Giants of Mesabi Range.

### *Virginia Slate, Iron-Formation Contact*

Because of possible value in connection with a revision of the correlation of the Huronian series in the Lake Superior district, the writer wishes to append to this paper the following record of observations of the relation of the Virginia slate to the underlying iron-formation, particularly because it is not in accord with statements as to this relation made in *Monograph* 52 of the *U. S Geological Survey.* The latter publication states that at the top of the iron-formation and the base of the Virginia

*A*

*C*

*B*

Fig. 14.—Fossils Found at Coleraine.

slate, there is a horizon perhaps a few hundred feet thick which is of gradation, in which layers of iron-formation and slate alternate, that "the layers of slate are found well down in the iron-bearing for tion, and layers of the iron-bearing formation are found well up in slate" (page 174, *Monograph* 52, *U. S. Geological Survey*).

An examination of cores from a great number of holes which p trated through the Virginia slate into the underlying iron-formation failed to substantiate this statement. Cores from two drill holes w penetrated between 500 and 600 ft. of Virginia slate and the entire un lying iron-formation to quartzite, and from scores of other holes, fa to show the presence of a single layer of greenalite in the true Virg slate. Several thin layers of a carbonate rock (probably calcium bonate) and a few crystals of iron carbonate were observed. In the i formation proper a very few bands of a carbonate rock were discove As shown on Fig. 5 of this paper, the upper horizon of the iron-forma is a slaty horizon in which layers of dark slate and greenalite altern At the top of this horizon and separating it from the true Virginia s (which is a dense dark gray or black slate), is a layer several feet t of calcium carbonate, amorphous or very finely crystallized. This l is mentioned on page 171, *Monograph* 52, *U. S. Geological Survey*. W ever drill holes have penetrated through the Virginia slate all along Range, this carbonate has been found immediately beneath it. I few holes, cores of which were examined very carefully, a small amo of conglomeratic material was found in the upper part of this carbo layer.

Although the average thickness of the iron-formation in adjacent a is quite uniform, as shown in the table of average dips and thicknes there are marked irregularities within short distances. Difference total thickness of iron-formation of 20 to 50 ft. in drill holes 1/4 mile a are common. Two holes, 1 1/4 miles apart show a difference in total th ness of 121 ft.; two holes 2 1/2 miles apart show a difference of 184 ft. enough close subdivision and correlation work has yet been done to termine whether such differences are due to initial deposition or ero from the top of the iron-formation. There is so much interbedded conglomerate in the upper horizons of the iron-formation that we k definitely that this part of the formation at least was deposited in v shallow water. It is not at all improbable, therefore, that prior to deposition of the Virginia slate, the iron-formation may have been rai above water, and its upper surface somewhat eroded. The variatio thickness of the Upper Slaty Horizon gives support to this idea. marked unconformity between the two formations can be establis however. They are conformable stratigraphically, as far as now kno but the significant fact of the absolute lack of any known greena (so far as revealed by many years of observation of thousands of f

of drill cores by the director of explorations of the Oliver Iron Mining Co.) in the Virginia slate and its marked prevalence immediately beneath the Virginia slate seems to call for such a pronounced change in conditions of deposition as to demand some time interval between the two. The universal prevalence of the calcium carbonate layer with some conglomerate between the two gives further support to this idea of a time interval.

These facts, minor though they may be, are presented here in the hope that they may be of some value to the geologists who are engaged in the revision of the correlation of the Huronian Series of the Lake Superior District.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba

BY MAX ROESLER, PH. D.,* SANTIAGO DE CUBA, CUBA

(New York Meeting, February, 1917)

## TABLE OF CONTENTS



* Geologist, Spanish American Iron Co. and Juragua Iron Co.

## I. INTRODUCTION

The following article concerns the geological occurrence of the iron-ore deposits on the south coast of Cuba. The article is based on a detailed field study, made in the hope that some information would be gained which might be of value in the search for further orebodies, or in the economic development of the ore already found.

### Location

The iron-ore deposits of the Firmeza district lie near the coast, in the southeastern part of Cuba, in the Province of Oriente. The orebodies of this district form part of a belt of deposits that extends from Sigua, 25 miles east of Santiago, to Sevilla, 5 miles east of Santiago, and lies on the seaward slope of the Sierra Maestra range of mountains

This range roughly parallels the coast in the southern part of Oriente Province. Its crest is about 6 miles north of the mines near Firmeza. The town of Firmeza lies 9 miles east of Santiago, and about 2½ miles from the Caribbean Sea. The mines included in the Firmeza district are the Ocania Mine and a group of mines that extends from West Five Mine to the Concordia Mine. The elevation of the mines is from 400 to 1,000 ft. above sea level. Their exact location is shown on the map (Fig. 1) which is a copy of a map furnished by the Juragua Iron Co.

Immediately east of the Firmeza district lies the Daiquiri district of the Spanish-American Iron Co.

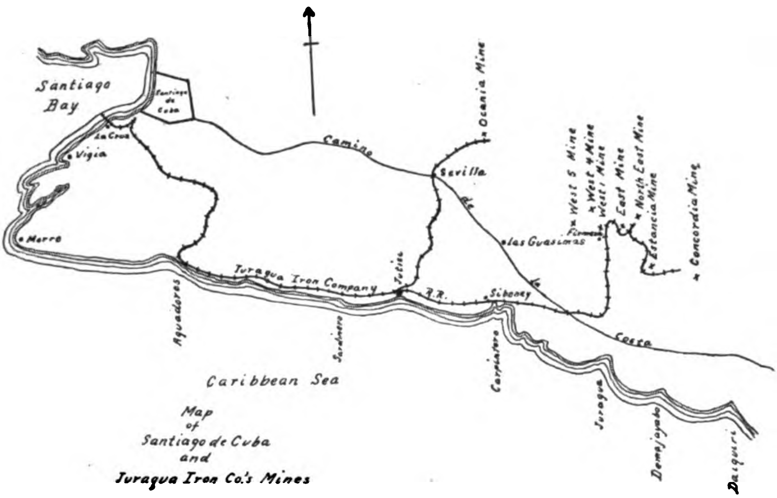



FIG. 1.

## Scope of Work and Acknowledgments

Detailed examinations have been made by J. P. Kimball,[1] A. C. Spencer,[2] and J. F. Kemp,[3] and many others have contributed to our knowledge of the deposits. The writer has made use of all the available information and will endeavor to give full credit for it.

The present article has been written under the first award of the S. F. Emmons Memorial Fellowship, and the writer takes this occasion to thank the committee in charge of the fellowship for their assistance and criticism. Thanks are also due to the Juragua Iron Co. for permission to make the examination, for the use of their maps, and for their kindness in giving access to all the available data and facilities. The Spanish-American Iron Co. also made possible the visiting of neighboring

[1,2,3] See Bibliography at end of paper.

properties and most grateful acknowledgment is hereby made to t officials and staff of both companies.

The Geological Faculty of Yale University, and Professor C. Berkey of Columbia University, have been freely consulted on differe phases of the problem, and their advice has been of much service.

## History and Mining

The Firmeza orebodies have been worked continuously since 188 except for a short period during the Spanish-American war, and t total production of the mines, from 1884 to 1913 inclusive, has be 6,776,171 tons.[4] The mines of the Firmeza district belong to the Jurag Iron Co., which is now controlled by the Bethlehem Steel Co., thou it was at one time also allied with the Pennsylvania Steel Co.

The mining of the ore and the stripping of the overburden is do by steam shovel. Where the orebody is small or intimately associat with waste, it is found advantageous to lease the workings to contracto and pay for the ore on the basis of tonnage and grade, delivered in mine ca

The ore from the mines is put through the crusher at Firmeza, fro which the major part is shipped direct to La Cruz, the company's shippi point on Santiago harbor. A small part of the output, high in sulph is roasted in the valley south of Firmeza, near Siboney, before it is shipp to La Cruz.

## II. TOPOGRAPHY AND ITS INTERPRETATION

The topography of the Firmeza district falls naturally into th main divisions which have a lateral extent parallel to the sea coast a the main range of the Sierra Maestra,—that is, in an east-west directi Beginning at the coast there is a range of terraced cliffs rising to unifo height of about 300 ft. The terraces all face the south, to the sea, a appear to be of wave-cut origin, excavated in flat-lying coral limesto There is no beach, the sea beating directly against the limestone cli except at the mouth of the Rio Carpintero at Siboney, where there i small beach made up of coral fragments and an arkosic sand in wh feldspar predominates.

Back of the terraces there is a fairly gentle landward slope to a s covered, flat, east-west valley. In this valley the streams from mountains are generally lost in lagoons and swamps. Some of the lar ones, such as the Rio Carpintero, find their way to the Caribbean throu comparatively narrow gorges cut in the cliffs that border the sea. Th streams appear to be of an intermittently torrential type. In the season they show very small streams of water flowing through stre beds covered with boulders up to several tons in weight.

[4] D. B. Whitaker: *Engineering and Mining Journal*, vol. 97, p. 677 (1914).

North of this valley rise the foothills of the Sierra Maestra—sharp, steep hills, covered as a rule with dense forest growth. They are connected with the main range of the mountains by ridges that have steep slopes and narrow crests. The connecting ridges are less heavily forested than the foothills but are covered with high grasses and an occasional grove of trees. The mountains themselves are thickly covered with pines and rise to an elevation of about 3,500 ft.

The limestone cliffs, with their sea-cut terraces, have been interpreted as evidence of periodic movements of the island, but it seems to the writer that the widespread occurrence of a terrace about both Jamaica and Cuba at the same elevation above sea level indicates a movement of the sea surface, rather than of the land.

In speaking of the Seboruco (the local name for the coastal limestone) R. T. Hill says:[5]

"Nowhere have I seen the elevated reef rock folded or otherwise disturbed except by the gently sloping coastward inclined elevation it has undergone.——The Seboruco as a whole represents a recent and uniform elevation of the whole periphery of the island——."

The following terrace elevations have been taken from the report of C. W. Hayes, T. W. Vaughan, and A. C. Spencer,[6] and arranged in tabular form by the writer:

| Havana, Feet | Matanzas, Feet | Gibara, Feet | Baracoa, Feet | Manzanillo, Feet | Santiago, Feet |
|---|---|---|---|---|---|
| 4–5 | 5–6 | 5–20 | 5–6 | 5–20 | 20 |
| 10–15 | 30 | 40 | 90 | 100 | 100 |
| 100 | 140 | 100 | 250 | 200 | 280 |
| 200 | 200 | 150–180 | ......... | ......... | .......... |
| ...... | 300 | .......... | ......... | ......... | .......... |

R. T. Hill[7] in the report on Jamaica has given less definite figures for the terrace elevations, and the following conclusions:

"In general the old reef rock of Jamaica consists of three distinct formations, occurring at three levels, 70, 25, and 10 ft. (or less) respectively. From the persistency of these three levels on the north, east and southwest end of the island, it is evident that their present position above the water is due to continuous epeirogenic elevation after the present outlines of the island had been chiefly defined."

For Hayti and Porto Rico no exact data are available to the writer.

[5] R. T. Hill: Notes on the Tertiary and Later History of the Island of Cuba. *American Journal of Science*, Ser. 3, vol. 48, p. 203 (1894).

[6] *Op. cit.* (in Bibliography), pp. 18, 19.

[7] R. T. Hill: Geology of Jamaica. *Bulletin* of the Museum of Comparative Zoology at Harvard, vol. 34, pp. 92–100 (1899).

The data quoted show that on the north, northeast, and sou coast of Cuba, and on the north, east, and southwest coast of Ja there is a marine terrace from 5 to 20 ft. in elevation. It seems writer that this evidence might suggest that, at least in part, the ential movements of land and sea in Cuba have been due to moven sea level rather than of land surface. The slight variations in ele would then be explained by unequal erosion of the gently sloping of marine planation. The slight seaward slope of the coast lim in the neighborhood of Santiago, is no greater than is to be ex of a near shore surface of marine planation.

There is further evidence of emergence of the island, in respect level, in the tuff-limestone found at an elevation of about 1,400 f the steep-sided, sharp, mountain topography points to fairly rejuvenation.

The silt-covered, east-west valley, in which the streams are a ing, suggests that the last movement of land with respect to sea le been a downward one, and this is in accord with the evidence harbors of Santiago and Guantanamo, which have been interpre bays formed by drowning.[8]

The topography seems to be almost entirely independent of th ology. An exception to this is the coarse crystalline marble that a capping to many of the foothills. Apparently the massive marbl greater resistance to tropical weathering than do the fine-graine basic or dioritic rocks which marmorized it, for the arroyos are c the latter. The occurrence of marble as a capping is too frequen entirely fortuitous.

## III. PETROLOGY

### Sedimentary Rocks

#### *Description*

The sedimentary rocks are represented in the Firmeza district e by limestones. At the coast, and rising in three sea-cut terrace altitude of about 350 ft., are recent limestones. According to the of Hayes, Vaughan, and Spencer,[9] this rock is

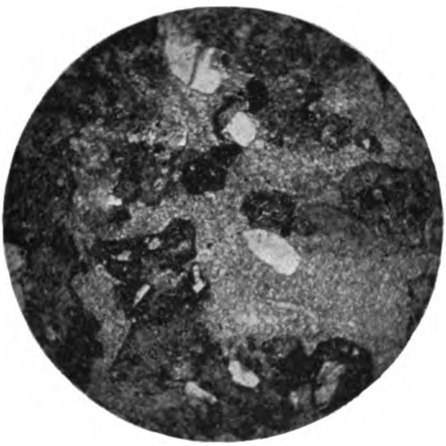
> "replete with the remains of numerous species of corals which are all, s examined, at present living in the surrounding Antillean seas."

There seems to be one rather striking point of difference betwe higher landward terraces and the lower terrace bordering the Cari This is, that while all three terraces contain coral remains, the la

[8] Hayes, Vaughan, and Spencer, *Op. cit.*, p. 17.
[9] *Op. cit.* pp. 23–24.

rraces are almost entirely massive, while the seaward terrace is made up more loosely cemented coral remains. On the land, where the limeone comes into contact with the underlying igneous rock, there are ulders of rock and of iron ore cemented by the lime, showing that osion of the orebodies had begun before these coral rocks were formed.

In the foothills of the Sierra Maestra, which vary in altitude from 0 to 1,300 ft., there occur, usually as a capping on the hilltops, bodies massive limestone now largely marmorized. Bedding is almost tirely lost, but where thinner masses have been involved in the volcanic cks it is possible to discern a pitch toward the southeast at an angle about 30°. The freshest pieces of this limestone, taken from boulders the north side of the hill north of West Five Mine, show it to be a ne, dense, fine-grained limestone. The microscope shows no evidence



. 2.—Tuff-Limestone from Ridge Connecting Foot Hills with Main Range of the Sierra Maestra. Limestone—groundmass. Plagioclase fragments—white. Volcanic rock fragments—dark. × 18.

organic remains in the thin section, but only a granular aggregate of cite.

On one of the ridges connecting the foothills with the main range, an elevation of about 1,400 ft., an unaltered limestone of unusual type s found. This rock has the deeply pitted surface typical of the weathd outcrops of all the limestone in this region. The color of the rock a fresh fracture surface is blue-gray. Closer examination shows nute areas with a vitreous luster embedded in a granular calcite trix, giving to the rock the appearance of a porphyry. A thin section the rock shows that the crystals are fragments of plagioclase, of about desine-labradorite composition. There are also angular fragments of liabasic rock (Fig. 2). The fragments are all about 0.5 mm. in diame-, and their presence in the limestone shows that the sediment was

formed at a time of volcanic activity on the adjacent land. The pe
freshness of the feldspars is probably due to the fact that the sea v
was already saturated with calcium carbonate, and so was not chemi
active in so far as the calcic feldspars were concerned. The ferro
nesian minerals of the diabasic fragments are entirely altered.
tuff-limestone seems to be very similar to that described by C. P. Ber
among the "probably" pre-Tertiary rocks of Porto Rico.

### *Age*

The nature of this rock shows that at least part of the vol
activity was contemporaneous with the sedimentation. As the and
rocks accompanying this tuff-limestone are believed to represent
initial stages of the igneous cycle, the determination of the age o
sediment must also fix the age of the igneous rocks.

The age of these limestones and the associated volcanics has be
rather vexed question. J. F. Kemp speaks of an article by H. Weddi
in which he places them in the Jurassic, in the horizon of Quenst
Beta, of the upper White Jura. This determination was made
material furnished by G. W. Goetz. The only other definite informa
on this subject of age is that given by J. T. Singewald, Jr., and
Miller.[12] They found a fossiliferous limestone in the Daiquiri dis
and submitted some of the fossils to T. W. Vaughan, who correlate
of the species with the Cretaceous of Jamaica and determined th
of the limestone as "Mesozoic, probably Cretaceous."

W. Lindgren[13] says:

"The idea of the geologists who have done the most work in this section
to be that the lavas and tuffs and associated limestones are of Eocene age——

R. T. Hill,[14] speaking of the volcanic clastics of the Jamaican
Mountain Series of Cretaceous age, says:

"In Cuba these clastic rocks constitute the high divides of the Oriente——

Experience during historic time shows that it would not be justif
to infer volcanic activity in one of the islands of the Antilles on the
of such activity in another. In the present case there is, however
evidence of Cretaceous sedimentation in Jamaica determined by R
Hill, the direct correlation of the Daiquiri specimen with the Jama

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[10] C. P. Berkey: Geological Reconnaissance of Porto Rico. *Annals of the
York Academy of Sciences*, vol. 26, p. 20 (1915).

[11] H. Wedding: Die Eisenerze der Insel Cuba. *Stahl und Eisen*, vol. 12, No
pp. 545–550 (June 15, 1892).

[12] J. T. Singewald, Jr., and B. L. Miller: The Genesis and Relations of the
quiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, pp. 67–74 (1916).

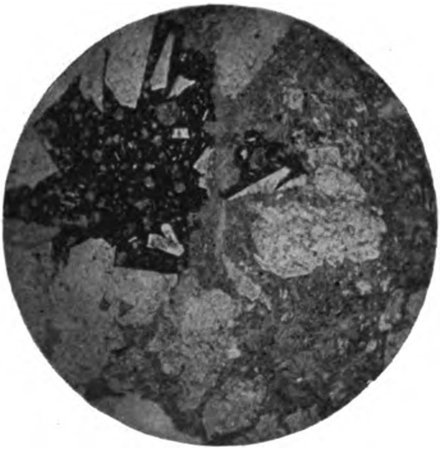
[13] W. Lindgren and Clyde P. Ross: The Iron Deposits of Daiquiri, Cuba. *Tr*
vol. 53, p. 41 (1916).

[14] R. T. Hill: The Geology and Physical Geography of Jamaica. *Bulletin o*
Museum of Comparative Zoology at Harvard, vol. 34, p. 170 (1899).

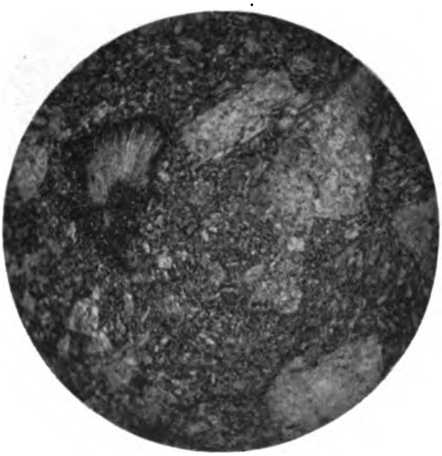
auna by T. W. Vaughan, and R. T. Hill's reference to the clastic rocks f Oriente. E. T. Hodge has informed the writer that he found Comanche ossils among the pre-Tertiary.

## Igneous Rocks

The igneous rocks of the Firmeza district form a natural series of ifferentiation products from a basic magma. They range from a fine-



'ig. 3.—Diabase Porphyry with Included Fragment of Earlier Volcanic Rock. West Four Mine. × 18.



'ig. 4.—Diabase Porphyry with Epidote-Filled Amygdule. From Railroad Cut Opposite West One Mine. × 18.

rained diabase to a highly quartzose aplite. J. F. Kemp[15] has given a iscussion of the nomenclature used in former articles on the district.

[15] The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. *Trans.*, 'ol. 53, pp. 3–38 (1916).

For purposes of mapping, the writer divides the igneous into four groups:

1. The diabasic rocks.
2. The dioritic group.
3. The granitic group.
4. The later dike rocks.

FIG. 5.—DIORITE FROM LEVEL 1 OF WEST FIVE MINE. CROS ED NICOLS

FIG. 6.—QUARTZ DIORITE FROM EAST MINE. THIS IS THE BASIC MEMBER GRANITIC SERIES. CROSSED NICOLS. × 18.

*Diabasic Rocks*

The diabasic rocks are fine-grained, as a rule, and porphyritic. are the representatives of the original magma. Megascopically th dense, dark rocks. Under the microscope they show as felty agg of plagioclase laths with interstitial ferromagnesian minerals. Th

ually fragmental (Fig. 3) and frequently show epidote-filled amygdules
g. 4).

These diabasic rocks show a bedded structure in places, with inter-
ated limestones and are undoubtedly in part extrusives.

. 7.—Granite Showing Micrographic Quartz Feldspar Intergrowth about
lagioclase Nucleus. From the Juragua Valley. Crossed Nicols. ×18.

*Dioritic Rocks*

The wall rock of many of the orebodies is a gray, fine-grained, and

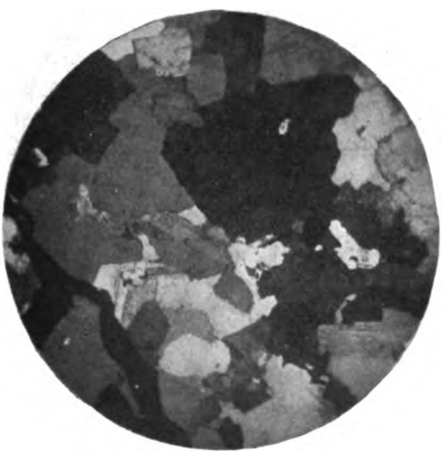


Fig. 8.—Specimen B from Dike in West Side of West Five Mine. Eleva-
n 520 Ft. Note Corrosion of Plagioclase and Dominance of Quartz.
ossed Nicols. × 18.

en-textured rock. Plagioclase, varying from labradorite to bytownite,
the most abundant mineral, with much hornblende between the feld-
rs. On the basis of the feldspars it is a gabbro, but in the older

classification the hornblende makes of the rock a diorite. Since tha terminology has been adopted by earlier writers on the district it accepted here. Fig. 5 shows a typical diorite.

The diorites and diabases are frequently found merging into eac other without any visible contact. Both are the hosts of the orebodie

### *Granitic Rocks*

J. F. Kemp[16] has used the term granite for the quartz-bearing di rites and the true granites "to avoid all confusion of this rock with tl diorites which are associated with the ore." That usage is adopte in this paper, and the scope of the term granitic rocks is enlarged include the aplites and quartz porphyries. This inclusive usage seem justified by the fact that the quartz-bearing rocks form a group lat than the ore-bearing rocks, though pre-mineralization.

All the rocks of this group are hornblende-bearing except the mo acid aplites, and even the most acid show very little potash feldspa The basic members of the series are gray, coarse-grained, feldspath rocks (Fig. 6) while the acid members are made up of striking feldsp quartz intergrowths (Fig. 7), or else are fine-grained quartz aplites (Fi 8). The extremely rapid variation in chemical composition is shown l three microscopic analyses of rocks from the same dike at differe elevations and ascending order.

| | Feldspar | Quartz | Ferromag. | Magt. |
|---|---|---|---|---|
| A.......... | 61.8 | 28.0 | 5.5 | 4.7 |
| B.......... | 54.0 | 41.6 | 3.4 | 1.0 |
| C.......... | 52.7 | 45.2 | 1.1 | 1.1 |

### *Later Dike Rocks*

The entire mineralized area is cut by basic dikes that vary fro basalt to andesite. They are post ore, but usually carry pyrite. detailed description would serve no definite purpose in this connection.

## IV. AREAL GEOLOGY

### Rock Types Found in the District

The study of the petrographic features shows that there are sever types of igneous rocks and two limestone formations in the Firmeza di trict. The igneous rocks form a continuous series from diabasic extru sive and intrusive rocks, through diorite and granite, to highly aci

[16] *Op. cit.*, p. 12.

aplites. This series is cut by dike rocks of various types, which range in composition from basalt to andesite. The sedimentary rocks fall naturally into two divisions;—earlier marmorized limestone, probably Cretaceous in age, and now exposed only in scattered outcrops, and younger recent coral limestones deposited as a fringe on eroded igneous rocks.

## Distribution of the Various Rock Types

### *Surface Distribution*

The two appended maps show the surface distribution of the rocks and orebodies. Owing to the lack of topographic maps, and the difficulties attending work in an area covered by tropical verdure, they cannot lay claim to great accuracy of detail. Future work will undoubtedly shift the contacts in many places and add a number of dikes. But enough work was done, and enough outcrops plotted to justify the making of the maps and the belief that they show the general relationship of the formations. Acknowledgment is made to Dean Corsa, whose unpublished map in the possession of the Juragua Iron Co. was in part used.

The different formations lie, as shown by the map, in belts of irregular width, roughly parallel to the coast line.

Nearest the sea is the belt of coral limestone. This belt is about 3,000 ft. wide in the area mapped. To the west, where the railroad runs to the Ocania Mine, the fringing limestone extends inland nearly 2 miles. Apparently the width of the belt is largely determined by the slope of the pre-deposition erosion surface, the wider belt lying ont he more gently sloping surface.

North of the coral limestone belt, the main mass of granitic rocks is exposed. This forms a belt from 6,000 to 8,000 ft. wide. Most of the granite area is covered by an alluvial deposit, through which rise knobs of the igneous rock. These knobs are all granite, usually with inclusions of dioritic material.

North of the granitic area lies the diorite. This forms a very irregular belt on the lower part of the foothills of the Sierra Maestra. From the Demajayabo River south of the Concordia Mine, to south of Loma Alta, the width of diorite is from 1,200 to 3,000 ft. At West Five Mine the diorite area runs back into the hills north of the region mapped. In the Juragua valley the diorite is exposed about 1 mile north of Firmeza.

The diabasic rocks lie higher in the hills than the diorite and in general their exposures on the surface are north of the diorite. No attempt has been made to show on the map the contact between the extrusive and intrusive facies of the diabases. This contact is exceedingly indefinite and rarely distinguishable.

Bodies of granitic rock, that differ from the main granite mass, cu through the diorite and the diabases in the region mapped. These bodi are apophyses of the main granite massif and form dikes, sills, or irregul bosses. They lie in a roughly east-west belt in the foothills, and are great importance because the orebodies are scattered irregularly as a frin to these apophyses. This can be seen clearly both on the map of th district and on the larger scale map of the mines near Firmeza.

The older limestones lie in the diorite and diabase area, rarely contact with the granite. They have been metamorphosed to a den marble in the area mapped, and are preserved as irregular masses cappin the foothills, or as beds interbedded with the extrusive rocks. The ou crops vary in size from small blocks to areas that are as much as 1,20 ft. in diameter. An even larger area at the Ocania Mine does not sho on the map.

All the formations are cut by the later basic dikes. Except in th mines these dikes have not been mapped. They fall into two system one almost vertical, striking northerly, or slightly east of north; the oth an almost horizontal system, that might equally well be classed as system of sills. Petrographically there is no difference between the tw systems.

*Vertical Distribution*

The vertical arrangement of the different rock types is even mo marked than their linear distribution, as it shows on the surface maps.

The coastal limestones rise in terraces to an elevation of about 30 ft., where they terminate in a flat top.

The granite massif has an upper surface which rises gradually from a elevation of about 350 ft. at the eastern end of the district, to about 70 ft. at the Ocania Mine. The granitic apophyses, as now exposed, a rarely more than 150 ft. above the top of the granite massif, and the downward extension where exposed by erosion can be seen to mer into the granite.

The diorite and the diabasic rocks must be considered together, b cause, while the lower limit of the diorite is determined by the top of th granite intrusion, the upper limit is, as a rule, indeterminate, and th diorite merges into the diabases. The two rocks were mapped separatel in the field and a contact drawn, but it is generally arbitrary. Mor rarely the contact mapped represents an observed intrusive contac As established, the contact between diorite and diabase rises from 50 ft. elevation in the eastern to 800 ft. at the Ocania end of the are Fragmental igneous rocks are found at 700 ft. in the East Mine, and a 1,000 ft. elevation at Ocania. No continuous contact between intrusiv and extrusive diabase was mapped.

The older limestone masses have not been found in this district, a

a lower altitude than 400 ft. The highest body of limestone examined was the tuff-limestone found at 1,400 ft.

The lack of contour maps makes it difficult to draw accurate sections. But the elevations can be tabulated as follows:

| West End of District (Ocania Mine), Feet | | East End of District, Feet |
|---|---|---|
| 1,000 | Fragmental igneous rocks | 700 |
| 800 | Diorite-diabase contact | 500 |
| 700 | Top of granite massif | 350 |
| 300 | Top of coastal limestone | 300 |

This table shows the distinct vertical distribution of the igneous rock types, and a well-defined pitch of their surfaces of contact.

### *Causes of Present Areal and Vertical Distribution of the Rock Types*

*Faulting.*—The linear distribution of the rock types, as shown on surface maps, suggests at once a direction of major faulting. But except for two minor faults in the East Mine, the one northeast-southwest, the other northwest-southeast in strike, and a somewhat larger east-west fault in West Five Mine, no displacements worthy the name of fault were found in the area. The nearest approach to a fault system is shown by the later basic dikes. These, with the one vertical north-south trend and a horizontal system, appear to have filled a definite fissure system, but there is no evidence of any appreciable movement along most of them. In West Five Mine an aplitic dike is offset 6 in. on the opposite sides of a 2-ft. wide basic dike. If the dikes do represent an older fissure system, it is a system due to contraction during cooling of the igneous rock, rather than to faulting. The Firmeza district must be added to the long list of those in which ore deposits are connected with fissures of minor or no displacement.

It is the writer's opinion that the present areal and vertical distribution is due to three different causes—magmatic differentiation, tilting, and erosion. Each of three must be considered in an attempt to solve the problem.

*Magmatic Differentiation.*—To show that the vertical distribution of the rocks is due to magmatic differentiation it will be necessary to establish the comagmatic origin of the igneous rocks and their age relationship.

The detailed petrographic study has shown that there are present in the district igneous rocks showing all the stages of a gradual transition from diabase porphyry to highly acid aplites. The merging of the types can also be found in the field.

Northeast of Estancia hill the transition of a granite-porphyry sill into the main granite and quartz-diorite massif can be followed. There is no contact between the two types. The granite massif itself contains numerous inclusions of more basic material. These inclusions are an-

gular, subangular, or rounded. J. F. Kemp[17] found apparent inclusions in the Daiquiri district and his interpretation is:

"Apparently the inclusions represent some older solidified rock

With this interpretation the writer does not feel in entire acc inclusions certainly represent early differentiates, either ear

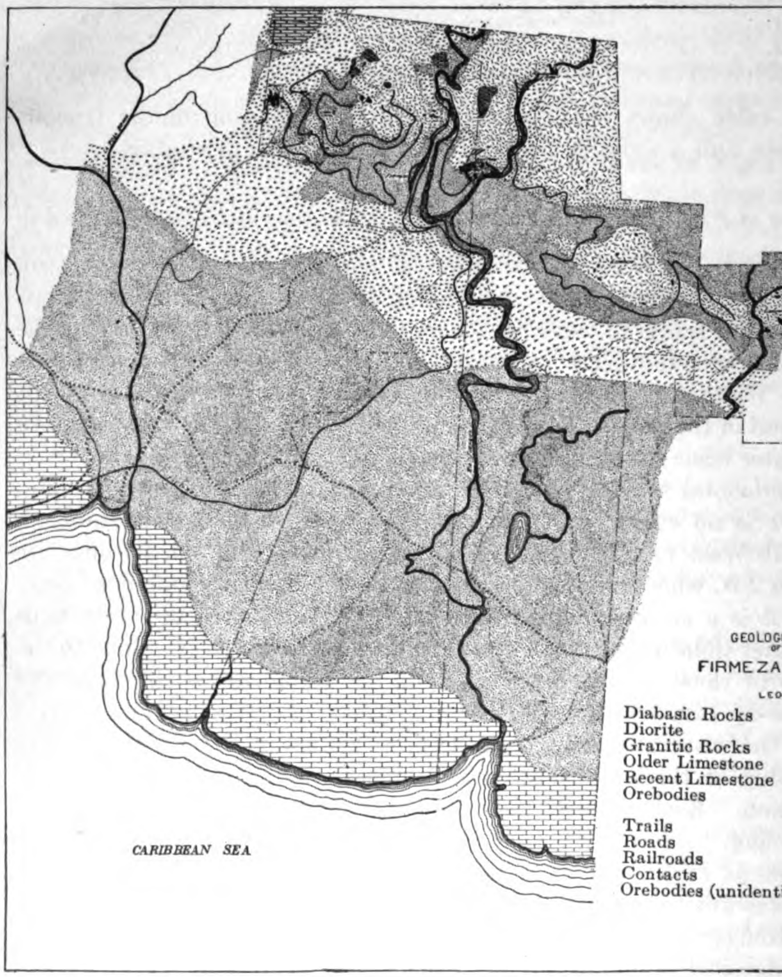



FIG. 9.

accumulations, or possibly segregations in the still liquid st had an opportunity to crystallize; but either way they are m reabsorbed portions of the magma itself, and so endogenous i

[17] *Op. cit.*, p. 16.

After initiation of crystallization the character of the magma changed until it had power to reabsorb part of the already crystallized material. The result of the same action was observed on a small scale in a microphotograph of a specimen from the lower part of the aplite dike in West Five (not reproduced in this paper). It shows a zonal plagioclase, calcic in the central part, sodic at the edges, and much corroded. Obviously the plagioclase must have formed before the liquid was of such a composition that it could corrode it. Similarly it seems possible that, in a slow-cooling magma, crystals form and gather in clusters, and the interstitial liquid, changing in composition, could reabsorb them more or less completely.

Beside the evidence of the endogenous inclusions, the transition from quartz-bearing to quartz-free diorite can be observed in the upper levels of West Five Mine, and that from diorite to diabase in West Four Mine. That the extrusive diabases and intrusive diabases are of the same magmatic origin can be seen from the complete similarity of the two rocks both megascopically and microscopically. The only difference is the greater fineness of the ground mass in the volcanics.

Other features that seem to point toward comagmatic origin of the igneous rocks are the almost total absence of orthoclase in the entire series, and the predominance of hornblende as a ferromagnesian constituent. The latter is considered an indication of the presence of abundant crystallizers which would aid in holding the magma fluid at comparatively low temperature.

The age relationship of the igneous rocks in a country lacking in sediments that are chronologic guides must be determined by intrusive contacts. The intrusive nature of the granitic apophyses in the diabases and diorites can be established in almost any of the mines. That the main granite massif is later than the diorite is well shown in La Posa brook just north of its junction with the Rio Carpintero, where dikes of massive, even-grained granite cut the diorite. Diorite cutting diabase is seen at the contact of the two rocks in the Juragua valley north of the mines. Diabase containing fragments of the volcanic diabases is found all through the district.

The order of formation is, then, the normal one from the basic to the acid end of the series; *i.e.*:

1. Diabasic extrusive.
2. Diabasic intrusive.
3. Diorite.
4. Granite.
5. Aplite.

The origin from a common parent magma and the age relationship established, it can be shown how this would account for the observed

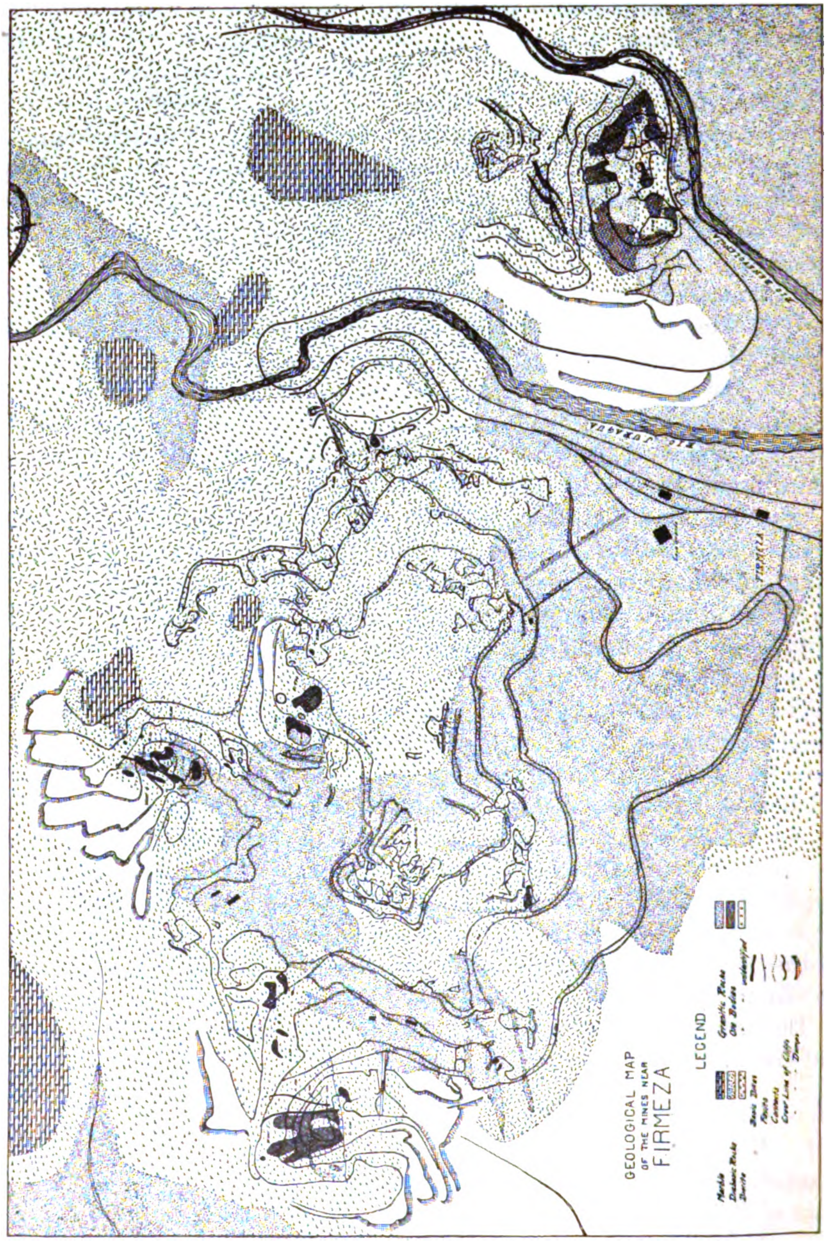

GEOLOGICAL MAP
OF THE MINES NEAR
FIRMEZA
LEGEND
Marble
Granitic Rocks
Ore Bodies
Basic Dikes
Faults
Contacts
Dumps

FIG. 10.

ertical distribution of the igneous rock types. The series from diabase o granite agrees so completely with the series described by N. L. Bowen[18] s the normal result of fractional crystallization in a basaltic magma, hat this mode of differentiation seems to be the probable one in this ase.

The diabase extrusives mark the initiation of igneous activity, and the iabase intrusive and the diorite mark early chilled phases of a magma ecoming more and more acid. The granitic massif is believed to be he batholythic invasion of the more completely differentiated acid liquid, nd the granitic apophyses part of the final differentiate from the granite. They will be discussed more fully with the genesis of the ore deposits.

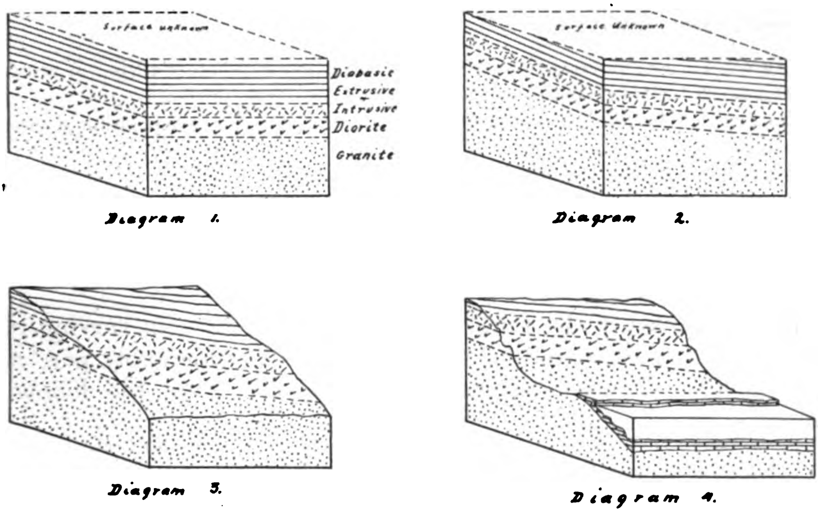



Fig. 11.—Genesis of Present Rock Types.

The position of the granite at a definite horizon is thought to be due to he fact that the cooled shell of the magma had developed cooling cracks o that depth, and as a result the crystallizers were able to escape from the ranite, which could then no longer remain liquid.

Thus differentiation may account for the horizontal arrangement of he rock types observed in the field.

*Tilting*.—However, the arrangement is not entirely horizontal. There is a distinct slant toward the southeast of all the surfaces of conact. The extrusives and the limestones, where bedding can be deermined, all show a pitch to the southeast. Just when the tilting took lace is difficult to establish, but since it does not affect the coastal limetone it may be placed as previous to its deposition.

[18] N. L. Bowen: The Later Stages of the Evolution of the Igneous Rocks. *Supplenent to Journal of Geology*, Vol. 23 (November–December, 1915).

Whether tilting accompanied or followed the igneous cycle and the ore deposition cannot be determined. As the orebodies in the eastern end of the district are lower than those at the western end, it seems more probable that the tilting took place later than the period of ore deposition. The evidence is inconclusive and indefinite, but it is unimportant as the time of the tilting cannot affect any of the major hypotheses.

*Erosion.*—Since the time of ore deposition erosion has produced a surface that slopes down in general from the north to the south. In this way it has cut across the basic rocks and into the granitic massif.

Fig. 11 shows a diagrammatic representation of the way the present arrangement of the rock types is believed to have been brought about. The diagrams are not to scale, and are intended merely as an aid to visualizing the conditions.

Diagram 1. Shows conditions after differentiation and before tilting or erosion.
Diagram 2. Shows conditions after tilting and prior to erosion.
Diagram 3. Shows condition after partial erosion but prior to deposition of coastal limestones.
Diagram 4. Shows conditions as they are at present.

## V. GENERAL DESCRIPTION OF ORE DEPOSITS

### Nature of Ore

The ore mined in the Firmeza district is a mixture, in varying proportions, of magnetite and hematite. The ore is remarkably pure and contains little foreign matter. J. P. Kimball[19] gives the following figures:

| | Per Cent. |
|---|---|
| Moisture (in part hygroscopic) | 0.24 – 0.81 |
| Silica and insoluble | 5.00 –10.50 |
| Phosphorus | 0.009– 0.065 |
| Sulphur | 0.045– 0.248 |
| Iron | 61.00 –68.50 |

Although present mining methods make it possible to ship ores somewhat lower in iron content, and greater depth in mining has exposed ores somewhat higher in sulphur, the figures Kimball gave in 1884 are substantially correct for the ores now being mined. The low phosphorus and the absence of appreciable amounts of titanium make the ore a very valuable one for the manufacture of high-grade steel.

### Shape and Size of Orebodies

In shape the deposits are extremely irregular. Whatever their size they show one common feature throughout the area—a far greater extension in two dimensions than in the third dimension. They resemble a

[19] J. P. Kimball: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Science*, Ser. 3, vol. 28, p. 426 (1884).

series of scattered lenses of irregular outline, that lie in every conceivable position. Fig. 12 shows a series of horizontal projections of the ore-bodies and a few vertical sections. The projections are drawn to scale, and the sections are sketched from exposures in the walls of the mines. The outlines represent the boundary of the ore of commercial grade. They do not represent the edge of the mineralization. There is generally a transition from ore to rock, not a definite contact between them. This subject will be more fully discussed under the mineralogy of the ore deposits.

The size of the ore deposits varies from pockets containing a few tons to lenses whose larger diameters are measured in hundreds of feet and whose thickness is from 10 to 50 ft.

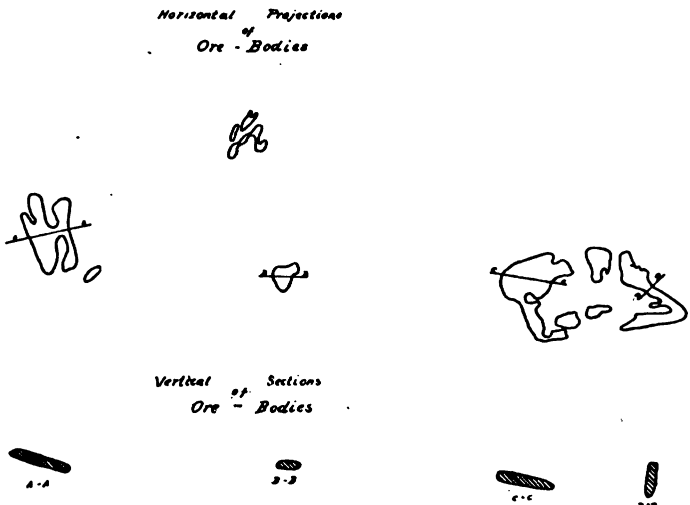



FIG. 12.

## GEOLOGIC POSITION

With the possible exception of the Ocania and Chicharron Mines, the ore deposits lie within 200 ft. of the granitic apophyses. These exceptions are probably more apparent than real and may indicate that the aplite has not yet been exposed, not that it is absent. In most cases the ores are directly in contact with aplitic or pegmatitic rock.

The relation of aplite to ore is exceedingly intimate. In most of the mines the ore lies against a floor or wall formed either of aplite or of pegmatitic granite with much micrographic quartz-plagioclase intergrowth. Frequent aplitic dikes traverse the ore, and form prominent features because of their light colors, in strong contrast to the black ore and the green igneous rocks. In spite of their apparent position as dikes, they are *not* to be considered as later than the mineralization. This will be shown in the discussion of the genesis of the ores.

The host rock of the ore is either one of the basic igneous rocks, or limestone. By far the greater part of the ore is in diorite or intrusive diabase; some is found in the diabasic extrusive rocks, and only a minor amount in the limestones. In at least three of the mines the ore deposits are very close to the overlying limestone masses, but there has been no localization in the limestone, and the larger masses are separated from the limestone by basic igneous rock.

## Mineralogy

The ore minerals are magnetite and specularite, with some amorphous, more or less hydrated, hematite in the upper parts of the ore deposits. The common gangue minerals are quartz, wollastonite, epidote, and lime-iron garnet. The epidote comes in two clearly distinct forms. The one is a well-developed, crystalline form, transparent and pleochroic from green to yellow. The other is a fibrous and scaly form, lacking in well-defined terminations. The well-crystallized form is the one that

Fig. 13.—Pyrite Crystal from West One Mine.

is associated with the ores, and other lime silicates. In some of the mines calcite is intimately intergrown with the ore. Apatite, titanite, and a mineral determined with some doubt as scapolite, were also found in rare cases. Secondary calcite, introduced by recent weathering, is not uncommon. Pyrite and chalcopyrite are found in most of the mines. They are almost always introduced and rarely if ever original constituents of the ore deposits. The pyrite occurs in well-developed crystals. One of these from the West One Mine is shown in Fig. 13. Associated with them is chlorite and some sericite.

### *Distribution*

The distribution of the minerals appears to be haphazard and extremely irregular, but natural phenomena are not haphazard, they are logical and sequential. Detailed study has convinced the writer that the minerals in these deposits are distributed according to a definite order. The most clearly localized material is the massive, fine-grained

intergrowth of magnetite and specularite. This massive ore lies in, and close to, fissures, or else directly next to aplitic or pegmatitic granite. In the minute intergranular spaces of this ore occur wollastonite and some quartz. As the ore becomes less massive toward the margin of the deposit the proportion of specularite to magnetite increases, wollastonite becomes more abundant, as does quartz, and epidote and garnet appear. Still further out from the massive ore, garnet predominates, with some epidote, little or no wollastonite, very little quartz, and some more coarsely crystallized specularite and magnetite. In some cases this zone is composed entirely of massive garnet with very few impurities. Beyond the garnet is an area in which epidote and quartz are prevalent. Mineralization of this type is the most widely diffused, and merges gradually into partially chloritized and epidotized country rock. The above description holds only for the typical deposits in igneous rocks. The complete sequence is rarely well-exposed. It can best be observed in the East Mine and in West Five Mine.

Those deposits which are obviously in limestone show two kinds of mineral distribution. One is shown in the Chicharron Mine. It is a dike-like mass of magnetite and specularite, with interstitial wollastonite. The contact with the marble is sharp, and no contact minerals are found. The other form of orebody in limestone is a central mass of almost pure specularite, in rosette-like clusters, or granular masses, surrounded by an intimate intergrowth of quartz well-crystallized calcite, garnet and epidote. This form of orebody is best exposed in the North Mine. The transition to unaltered marble is not exposed there. In the Ocania Mine this transition is exposed and shows a sharp change from garnet rock to unaltered marble.

Apatite is so rare that it is a curiosity in the district. Fig. 14 shows a microphotograph of the one section in which it was found in appreciable amount. It appears in the usual hexagonal basal, and elongated prismatic sections. Later than the apatite in time of crystallization are magnetite and the minute garnet-epidote intergrowth. The scarcity of apatite and so of phosphorus in the ore is of the greatest economic importance. It is also an indication that phosphorus had little or no share in the mineralization.

The secondary calcite, the kaolin, and the amorphous and hydrated hematite, are found either very close to the surface or in channels to which meteoric waters have obviously had access. Pyrite is common in all the mines and lies close to the surface in an unaltered, well-crystallized condition.

### *Interpretation of the Mineralogy*

The first inference to be drawn from the mineralogy of the ore deposits is that they were formed at a high temperature. The temperature of

mineral formations is not yet so well known that accurate conclusions can be drawn from mineralogy as to the exact temperature at which ore

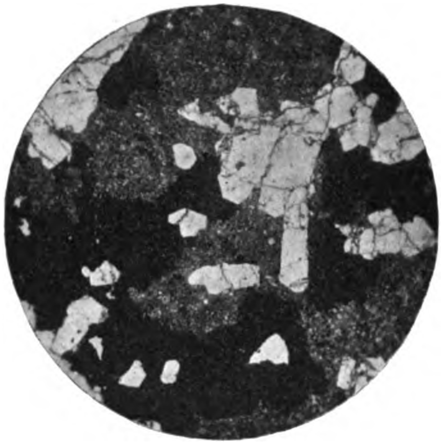


FIG. 14.—SPECIMEN FROM WEST FIVE MINE. × 18. Magnetite—black. Epidote and garnet—gray. Apatite—clear white.

deposits were formed, but the occurrence of lime-iron garnet and of wollastonite indicates high temperatures. The association with quartzose pegmatitic dikes, which, as will appear later, are believed to be con-

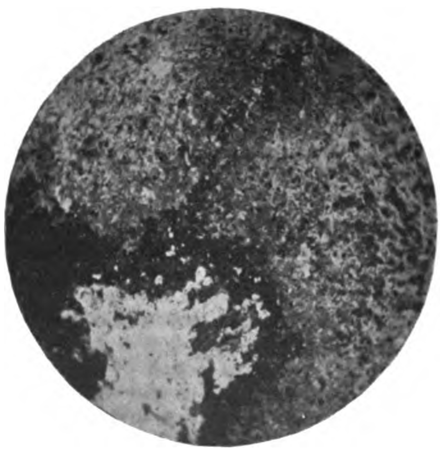


FIG. 15.—DIABASE PORPHYRY, SHOWING INTRODUCED QUARTZ AND EPIDOTIZATION. FROM EAST MINE. × 18. Quartz—white. Epidote—dark.

temporaneous with the ore formation, places the temperature at from 575° to 800°C.[20]

[20] F. E. Wright and E. S. Larsen: Quartz as a Geologic Thermometer. *American Journal of Science*, Ser. 4, vol. 27, p. 421 (1909).

The second inference drawn from the mineralogy is that the iron, lime, and quartz are either entirely, or in part, introduced material. The large quantity and great concentration of the iron oxides makes their formation from the rock in which they are found highly improbable, and it can be assumed that they represent, almost entirely, introduced material.

The lime silicates, epidote, wollastonite, and garnet are so abundant that an addition of lime seems extremely probable. Because the rock, now entirely replaced by ore, was a calcic igneous rock, it is impossible to determine whether the lime, which was added to form garnet zones, came from the magma or from the replaced rock. Since, in places like Estancia Hill, the garnet zones are out of all proportion to the size of the orebodies, an addition of lime from the magma seems probable. The major part of the lime is deposited in a zone beyond the iron. In this section the lime silicates always show a later crystallization than the iron oxides. This makes it appear as if the mineralizing agent deposited most of the iron oxides while still able to retain much lime in solution. Further evidence of the addition of lime from the magma is afforded by the apatite, intergrown with the magnetite, in the specimen from West Five Mine described above (Fig. 14). The epidotization and silicification in the outermost zone seems to be largely recrystallization, and represents a hydrothermal alteration, more or less *in situ*, rather than an addition of material. The chloritization is also an effect of this hydrothermal metamorphism. Six samples of diabase showing more or less epidote were analyzed for lime. They were selected at various distances from a limestone inclusion, in order to show a supposed absorption of lime. The variation in the lime content is so slight that no addition of lime in this zone can be postulated. The quartz is probably largely a byproduct of the alteration of hornblende to epidote. It is also in part introduced, as is shown in the microphotograph of a specimen from the diabasic porphyry in the East Mine (Fig. 15).

In regard to the chemical form of the mineralization, inferences are largely negative. Evidence of the participation in the mineralization of any of the halogens is practically lacking. The apatite and dubious scapolite indicate some chlorine, but the occurrence of chlorine-bearing minerals is so rare that the introduction of the iron as chloride cannot be shown. $CO_2$ would be expected to show its presence in the formation of carbonates in the zone of hydrothermal alteration. No appreciable amount of carbonates was found, except in the occurrences in limestone. The presence of much water is shown, however, by the extensive hydrothermal alteration in the outer zone of mineralization. If the inferences in regard to the temperature of the formation of the ore deposits is correct, it is more than 200° above the critical temperature of water (358°C.). The fact that this water carried material in solution makes it impossible

to prove that it was in the form of vapor; but the high temperature and the fact that the mineralizers were able to permeate dense rocks so readily makes it probable that they were in the gaseous rather than the liquid phase. The mineralizers are, therefore, believed to have been dominantly water vapor, carrying with it iron, lime and silica. The variation in mineralization with decreasing intensity is believed to be as shown in the diagram, Fig. 16.

A later mineralization is represented by the pyrite and chalcopyrite. Field observation shows that the pyrite in the ore lies in channels along which solutions have passed. In the ore it is usually accompanied by chlorite. Where the pyritization has taken place in the wall rocks, as at the west side of the Ocania Mine, both chlorite and sericite occur. The pyrite seems to represent a very much later stage of mineralization than the magnetite and hematite. It is impossible to prove any definite

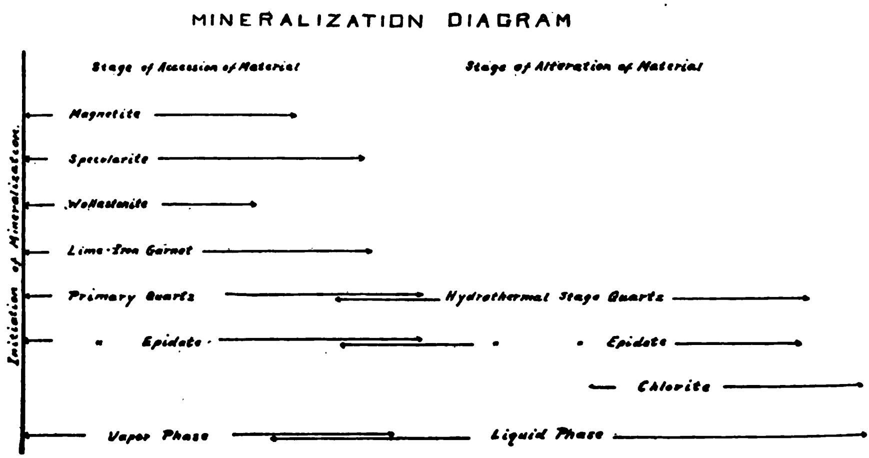



Fig. 16.

connection between the pyritization and the mineralization that produced the orebodies. It seems probable that the solutions that brought the pyrite came in at the time of the intrusion of the later dikes.

## *Superficial Alteration*

W. Lindgren believes that the hematite is the result of the alteration of magnetite by surface agencies. He says:

"——the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters."[21]

The occurrences in the Firmeza district are in entire accord with those

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[21] W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vlo. 53, p. 52 (1916).

recorded by Lindgren at Daiquiri. There is an increase of hematite in the higher levels of the mines. A polished specimen of the massive ore from the Chicharron Mine shows very clearly the intimate intergrowth of specularite and magnetite, and some of the specularite forms a later veinlet through the magnetite; but to accept without question specularite as an oxidation product of magnetite is difficult. The writer has been unable to find any record of the synthesis of specularite except at high temperatures, or of the occurrence of specularite as the result of surface oxidation except under superimposed regional metamorphism. It is not the crystalline facies of hematite that forms under surface agencies, as the result of dehydration of limonite, in the ores of Mayari. Lindgren does not state that the process of hematitization produces the crystalline facies of hematite (specularite). But since it is the crystalline phase that is abundant, and the earthy phase rare, it must be the specularite to which he refers. He does say:[22]

> "The pits in the hematite are probably caused by the local development of a softer or earthy facies of the mineral."

In spite of the fact that all the observations accord with the theory advanced by Lindgren, the writer desires to suggest another hypothesis—that is, that both the magnetitization and the hematitization are due to a primary mineralization by gaseous solutions deficient in ferrous oxide.

F. W. Clarke says:[23]

> "Ferric oxide can crystallize out as hematite only when ferrous compounds are either absent or present in quite subordinate amounts, for ferrous oxide unites with it to form magnetite."

The crystallization of specularite as a later phase, from a gaseous solution containing insufficient ferrous oxide to produce all magnetite, is in accord with Clarke's observation. Deficiency of FeO could account for a change from magnetite to specularite about centers of crystallization. It could also account for a diffusion of the hematitization to a higher level, further from the centers of mineralization than the magnetitization. Local variations in the mineralizing solutions account for the variations in the proportion of hematite to magnetite in bodies equally close to the parent magma.

The only important effect ascribed to surface waters by the writer is the oxidation of the pyrite near the surface, and the resulting decrease in the undesirable sulphur content.

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[22] *Op. cit.*, p. 52.

[23] F. W. Clarke: Data of Geochemistry. *Bulletin* No. 616, *U. S. Geological Survey*, p. 347.

## VI. GENESIS OF THE ORE DEPOSITS

### Previous Theories

The preceding description of the ore deposits, and discussion of their mineralogy, makes it possible to consider the larger features of their mode of formation. As this is purely a matter of interpretation, it is necessary to consider first the interpretation given by others. J. P. Kimball, writing in 1884–85, and F. F. Chisholm, in 1890, formed their conclusions from observations made in the Firmeza district. A. C. Spencer based his theories on work done at both Firmeza and Daiquiri in 1901. The papers by J. F. Kemp, and by W. Lindgren and C. P. Ross, were written more largely from evidence gathered at Daiquiri, with some specimens and a brief visit by Kemp to the Juragua mines in 1914. J. T. Singewald and B. LeRoy Miller made a brief visit to the Firmeza and Daiquiri districts in the fall of 1915.

J. P. Kimball[24] recognizes two types of ore deposit in the Juragua district. These are "replacements" of the coral limestone by iron-bearing solutions, and "concentrations" of ferric oxide in the diorite, "almost *in situ.*" Both replacement and concentration are ascribed to the action of circulating meteoric waters.

F. F. Chisholm[25] discusses the theory advanced by J. P. Kimball, and disagrees with it. He states his own opinion as follows:[26]

"My conclusions, after going in detail over most of the exposure made by the Union cut, were that, whatever the exact character of the ore deposit, the present position of the ore cannot properly be considered the result of local metamorphism of limestones by the action of surface waters containing iron leached from the overlying mass of iron-bearing diorite. I am much more strongly inclined to consider the ore here either the result of concentration within a diorite dyke which was originally characterized by the presence of a large percentage of iron, or else a distinct band forming a portion of a larger dyke. In other words, I am strongly of the opinion that the source of the ore is from below, and consequently that the loyalty of these deposits may be relied on below the limits of atmospheric action. I regret that I was unable to go into the question in detail, and get positive facts in support of my belief, but that I am obliged to admit that my examination was too superficial to enable me to prove my views."

A. C. Spencer,[27] after considering several possible explanations of the genesis of the ore deposits, comes to the conclusion that they are parts of

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[24] J. P. Kimball: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Sciences*, Ser. 3, vol. 28, p. 426 (1884).

The Iron-Ore Range of the Santiago District of Cuba. *Trans.*, vol. 13., pp. 613–634 (1884–85).

[25] F. F. Chisholm: Iron-Ore Beds at the Province of Santiago, Cuba. *Proceedings of the Colorado Scientific Society*, vol. 3, part 3, pp. 259–263 (1888).

[26] *Op. cit.*, p. 262.

[27] C. W. Hayes, T. W. Vaughan, and A. C. Spencer: *Report on a Geological Reconnaissance of Cuba*, 1901.

an older limestone-schist series which has been completely involved in the later igneous intrusions. The following quotation shows how completely Spencer supposed this older series to have been immersed in the igneous rock.[28]

"A mass of many million tons weight floated upward by the buoyant effect of molten rock in motion from the interior toward the surface of the earth, is the only conception which adequately accounts for the mode of occurrence of the orebodies of the Magdalena and the Lola Mines at Daiquiri; while, though less strikingly shown, at Firmeza it is likely that the masses of schist, marble and ore have been likewise actually suspended in the molten lava."

J. F. Kemp[29] divides the orebodies into two types—the "Distinctive Contact Zones" in limestone, and the orebodies in the diorite. The former he regards as due to contact-metamorphic effects produced in the limestones by the granite. Of the orebodies in the diorite he says:

"One is led to the conclusion that while the diorite mass was still hot in the depths or after it had consolidated and had been penetrated by some other and still hot intrusive in depth, a pronounced northwest and southeast fissured zone was formed, up through which came the emissions, fluid or gaseous, which brought the iron for the ore, the pyrite, the garnet and the epidote; the sulphur for the pyrite; and the silica for the quartz, the garnet and the epidote. The lime required by the garnet and the epidote may have been derived from the plagioclase and hornblende of the diorite, or srom included blocks of limestone, or deep-lying limestone, or from several of these fources."

Another quotation from the same article might indicate that Kemp suspected what has become the conviction of the writer:[30]

"One cannot help associating the granitic or pegmatitic dikes with some large parent body. The natural one is the intrusive granite mentioned at the outset. Yet this granite has produced contact zones on the older limestone with orebodies, whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies. We can only suspect the possible connection without being able to prove it."

Kemp apparently bases his conclusion that the aplite dikes represent a later intrusion into the ore on the fact that the dikes appear to cut the ore. That another hypothesis is possible will be shown later.

W. Lindgren and C. P. Ross,[31] on evidence gathered by the senior author, deduced the following conclusions:[32]

"From the above it is clear that the primary iron oxide of the deposits at Daiquiri is a magnetite, which subsequently has been altered more or less completely to a

[28] *Op. cit.*, p. 82.

[29] J. F. Kemp: The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. *Trans.*, vol. 53, p. 30 (1916).

[30] *Op. cit.*, p. 21.

[31] W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vol. 53, p. 52 (1916).

[32] *Op. cit.*, p. 53.

hematite. The discussion of genesis may therefore be divided into two parts: the first relating to the origin of the magnetite; the second to its subsequent hematitization.

The mineral association of the magnetite, particularly the presence of much garnet, shows that it originated under high-temperature conditions.

On the other hand, the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters."

The problem of hematitization has already been discussed.

Further on three theories discussed by A. C. Spencer are rediscussed by the authors. After dismissing the other two theories, the second theory is accepted, as follows:

"There remains the second theory, accounting for the large masses of magnetite included in the diorite by contact-metamorphic limestone, by which the latter has become almost entirely replaced by magnetite derived from magmatic emanations rich in iron. While it is freely admitted that the genesis of the Daiquiri deposits is difficult to explain, it will be shown that the view outlined in the previous sentence has much in its favor."

J. T. Singewald and B. LeRoy Miller discuss the theories advanced by J. F. Kemp, and by W. Lindgren and C. P. Ross in the papers cited above and come to the conclusions summed up in the following paragraph:[33]

"To sum up our opinions, the Cuban iron ores are contact-metamorphic deposits localized about engulfed blocks of limestone in diorite. In such cases, where there was a limited supply of magmatic emissions, there resulted the contact metamorphism of only a part of the limestone block. Where the supply was ample and the action most intense, not only was the block of limestone completely replaced, but complete endomorphism of the igneous rock on a large scale occurred in the vicinity."

The above are the interpretations offered by others who have studied the field more or less closely. But a study of different exposures and a different viewpoint, have led the writer to conclusions which are somewhat at variance with those quoted.

## Hypothesis of the Genesis of the Ore Deposits

Any comprehensive theory concerning the formation of an ore deposit in a rock of which it is not an original part—*i.e.*, an epigenetic deposit—must account for four things:

I. The source of the material forming the deposit.
II. The vehicle that introduced the material.
III. The channel through which the material was introduced.
IV. The cause for the deposition of the material.

### *Source of the Iron Ore*

The ultimate source of the iron ore was the diabasic magma. The extruded part of this magma, with the involved limestones, together with

[33] J. T. Singewald and B. LeRoy Miller: The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, p. 73 (1916).

the chilled upper part of the magma (the diabases and diorite) form the host rock of the ore deposits. It has already been shown, in the discussion of the areal geology of the district, that this magma was differentiated, that the earlier igneous rocks represent the parent magma and its early differentiation products, and that the latest rock differentiate was the granite massif with its aplitic apophyses. These aplites, which are, under anhydrous conditions, extremely viscous, must have contained large quantities of crystallizers, to keep them fluid and enable them to penetrate the basic rocks as sills and dikes. It is believed that these crystallizers were mostly composed of water vapor well above the critical temperature, and that they held in an ionized state the ore minerals.

### *Vehicle to Carry the Material*

These concentrated crystallizers were also the vehicle to carry the mineral burden into the rocks which act as host for the ore deposits, *i.e.*, the crystallizers plus the minerals are the mineralizers. Their nature has already been discussed under the interpretation of the mineralogy.

### *Channels through which the Materials Were Introduced*

One of the striking features of the larger Firmeza deposits is that the ores have almost obviously avoided the masses of marmorized limestone. This can be explained by the fact that the channels, through which the mineralizers entered, were not lines of weakness or shear zones produced by structural faulting, but were cooling cracks in the igneous rocks.

### *Causes of Deposition*

The causes of deposition may be of either a chemical or physical nature, or a combination of the two. It has been the tendency of geologists to ascribe to chemical action the dominant rôle in contact-metamorphic actions. This makes it difficult to account for ore deposits in two rocks as chemically different as diabasic igneous rock and marmorized limestone, except by postulating different theories. If it is believed that the dominating cause of deposition is physical, and is inherent in the mineralizing solutions, whether liquid or gaseous, this difficulty is obviated. It is the writer's opinion that such was the case in the formation of these deposits. The mineralizers were under such great pressure and at such high temperature that they had the power to diffuse through, and either partially or wholly absorb, any rock with which they came into contact. This diffusion and partial absorption would rob the mineralizers of a large part of their activity, and would cause deposition of the minerals in a definite order. The order would be dependent on the

saturation of the gases or solutions with the material to be deposited. The field evidence for such an order is clear, and as shown in discussing the mineralogy of the deposits, magnetite and specularite are the first minerals deposited. As a result, the magnetite and specularite are localized in and near the fissures through which the solutions had access, while the epidote, which is probably only in part due to accession of material from the magma, and is more largely a recrystallization of material present in the host rock, is much more widely diffused. Joseph Barrell, in conversation with the writer, suggested that such a hypothesis may also account for the association of aplitic and pegmatitic dikes with basic mineral segregations. The dikes are regarded as the residual, acidic rock material left in fissures, from which the mineralizers had gone into the rockbody, and in the process had effected a separation of acidic and basic material.

Under the hypothesis advanced, the orebodies owe their localization in igneous rock or limestone almost entirely to the accessibility of the rocks to the mineralizing solutions. The same causes govern all the cases.

Briefly stated, it is the writer's belief that the orebodies in the Firmeza district are due to mineralizers concentrated by further differentiation from the granitic massif, which is itself a differentiate of the diabasic magma. These mineralizers diffused from the igneous-rock material into those rocks to which fissures gave access, and, in diffusing, deposited their mineral burden, and left the rock material with which they had been mixed in the fissures, as aplites and pegmatites.

Under this hypothesis the mineralization becomes a definite event in the igneous cycle, which cycle may be restated as follows:

1. Extrusion of diabasic rock material.
2. Intrusion of diabasic magma.
3. Formation by differentiation of more acidic residual liquid in magma reservoir, with concentration of mineralizers.
4. Batholythic invasion of granitic material, together with mineralizers.
5. Further differentiation of the granitic batholith and escape of mineralizers with some acidic rock material, into cooling cracks in country rock.
6. Separation of basic mineralizers, and acidic rock material, with contemporaneous formation of ore deposits and aplitic dikes and pegmatitic apophyses.

The hypothesis submitted above is given prior to a detailed description of the ore deposits. It must be tested by its application to the phenomena observed in the field, and by its ability to meet the objections raised by other writers on the same subject.

## Discussion of the Different Hypotheses

The hypotheses of the different geologists who have visited the region have been given, either as abstracts or by quotations. The earliest of these, that by J. P. Kimball, does not account for the formation of what are now known to be high-temperature minerals: garnet, wollastonite, etc. Kimball recognized, however, that some of the ores lay in diorite and could not be accounted for by a hypothesis which depended entirely on limestone as a precipitant. F. F. Chisholm does not attempt to formulate a complete theory of the genesis of the ore deposits. With his conclusion that the mineralizing solutions came from below, and were not of meteoric origin, the writer entirely agrees.

The theory accepted by A. Spencer requires an older limestone-schist series. Limestone was found in the Firmeza district in considerable abundance, but schist could not be found by the writer. The occurrence of tuff-limestones interbedded with the volcanic extrusive material makes it seem probable that the period of sedimentation, and the initiation of the igneous cycle, were not separated by a time interval sufficient for the forming of surface orebodies and their entombing in igneous rock. This theory also fails to account for the forming of ore in diorite. In a tunnel under West Five Mine the transition from fresh diorite to granular magnetite can be traced. The transition is gradual and there is no contact which could possibly be interpreted as the margin of a partly assimilated block.

Later writers all agree in ascribing an igneous origin to the mineralizing solutions. The principal divergence is in the amount of influence ascribed to the limestone. Lindgren and Ross believe the ore to be definitely localized in limestone. Singewald and Miller regard the ore as in, and about limestone. Kemp believes that some of the deposits are independent of limestone. In regard to this feature the writer agrees entirely with Kemp. The conception of the orebodies as metamorphosed, engulfed limestone masses is difficult. The shape and position of the ore-masses demand that the entombed blocks should have been comparatively thin, and that they should have come to rest in every conceivable position. This theory completely fails to account for the gradual transition from ore to diorite shown at its best in West Five Mine. Such a deposit could be accounted for only on the basis of complete assimilation of the limestone, and a deposition of magnetite about the locus of assimilation. Any theory that places the burden of producing the orebodies upon the diorite, is open to one great objection. It demands that a magma of almost gabbroic basicity shall have produced contact-metamorphic phenomena of the most intense variety. Kemp recognized this, and ascribes the orebodies in limestone to the granite, and the orebodies in the diorite to solutions from "some other" intrusive. But Lindgren and

Ross, and Singewald and Miller regard the diorite as the source and as the cause of the mineralization. Another objection is the fact that, under that theory, the diorite must have exercised a strong selective tendency. Unaltered marble within 20 ft. of massive ore, and separated from the ore by recognizable diorite, can be seen at West Four. Such a case is not rare, it is a common feature. In one of the old cuts above West One Mine there is exposed a small block of marble, completely surrounded by diabase containing small ore masses. But the marble is entirely unaffected. Contact-metamorphic phenomena are undoubtedly capricious, but if another explanation will solve the problem in such a way as to show that the capriciousness is more apparent than real, it must at least be considered.

The great objection to ascribing the source of the ores to the granitic batholith lies in the interpretation of the aplitic and pegmatitic dikes. Field observation clearly establishes the direct connection between these rock types and the granitic massif. If these most acidic facies represent a later intrusion into the ore, the ore cannot come from the granite. Kemp believes they are later. He says:[34]

"——whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies."

Lindgren and Ross[35] agree with this.

"The dikes of igneous rock[35] intruded into the iron ore appear to be somewhat different from the diorite and are either granite porphyry or aplite; that is, complementary dikes of a later generation."

F. Klockmann, writing on contact-metamorphic magnetite deposits in general, puts the matter even more strongly:[36]

"Wiederholt wird angegeben, dass aplitische Gänge und Granitapophysen die Magnetitlager stätte durchsetzen. Gibt es denn für den Unbefangenen dafür noch andere Deutung als die, dass der Granit in eine von ihm vorgefundene Lagerstätte eine Apophysen entsendet hat?"

As a general proposition, the very fact that aplitic dikes and granite apophyses are repeatedly reported as traversing magnetite deposits is the strongest possible argument against the purely fortuitous intrusive nature of such apophyses. It was this constant relationship observed in the field study, that first convinced the writer that there must be definite connection between the forming of the orebodies and of the granitic apophyses.

J. H. L. Vogt recognizes a connection between the granite and its apophyses, and the magnetite deposits of Kristiania. He says:[37]

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[34] *Op. cit.*, p. 21.

[35] *Op. cit.*, p. 54.

[36] F. Klockmann: Über Kontaktmetamorphe Magnetitlagerstätten. *Zeitschrift für Praktische Geologie*, vol. 12, p. 81 (1904).

[37] J. H. L. Vogt: Problems in the Geology of Ore Deposits. *Trans.*, vol. 31, p. 138 (1901).

"A study of the Kristiania contact-deposits indicates that the formation of the ores preceded the solidification of the granitic magma. Even when the ores occur in slates immediately adjacent to the granite, or in the small Silurian fragments completely surrounded by granite, they are never found also in the granite itself. This is to be simply explained by the supposition that from the still liquid magma the ores were 'blown into' the adjoining rigid rocks. —The presence in these deposits of granitic apophyses, already mentioned, is another proof that they were formed before the solidification of the granite."

He regards the occurrence of the apophyses with the ore in the Kristiania district as proof that the granitic magma had not yet solidified.

The absence of ilmenite in the Firmeza occurrence is also an indication of their intimate connection with the granitic magma. Vogt has given some very complete studies of the relationship between the different


FIG. 17.—APLITIC GRANITE BORDERED BY MAGNETITE AND EPIDOTE. TUNNEL AT ELEVATION OF 299 FT. UNDER WEST FIVE WORKINGS. MAGNETITE CAUGHT IN APLITE = M. ACTUAL SIZE. Aplite—white. Magnetite—dark.

magnetite deposits and their sources. A quotation that shows some of his conclusions follows:[38]

"Die Erfahrung ergiebt, dass bei der 'oxydischen' Erzaussonderung es namentlich das 'Eisenoxyd-mineral'—bei den Gabbros Titaneisen oder Titanomagnetit, bei den Peridotiten Chromit—ist, welches concentrirt wird. Es ist somit a priori anzunehmen, dass etwaige 'oxydische' Erzausscheidungen in den Graniten einen ähnlichen niedrigen $TiO_2$-Gehalt führen müssen wie die in dem Granit normal ausgeschiedenen (wohl namentlich Magnetit und Eisenglanz, untergeordnet Titaneisen.)"

The strongest evidence of the intimate relation between the granitic apophyses and the mineralization is based on field observation. Some of the small dikes are reproductions in miniature of the larger occurrences. Figs. 17, 18 and 19 are photographs of two of these small dikes.

Fig. 17 is of a specimen from the ore in the tunnel under West Five

[38] J. H. L. Vogt: Zur Classification der Erzvorkommen. *Zeitschrift für Praktische Geologie*, vol. 2, p. 394 (October, 1894).

Mine. This is as deep-seated an ore deposit as has been exposed, and is the one most clearly independent of limestone. It is noticeable how entirely clear-cut is the contact between aplite and magnetite in the lower part of the picture. At the same time the other contact is less sharply defined and clearly shows some magnetite trapped in the aplite.

FIG. 18.—PHOTOGRAPH OF SPECIMEN FROM EAST MINE, SHOWING RIM OF GARNETS DEVELOPED BETWEEN APLITIC GRANITE (WHITE) AND DIABASE PORPHYRY (DARK). ACTUAL SIZE.

Fig. 18 is a photograph and Fig. 19 a microphotograph of a specimen from the East Mine. This specimen comes from the area of diabasic rocks, that lies between the ore in the East Mine and the ore in the pit above, known as North Mine. The specimen shows garnet, merging

FIG. 19.—GARNET BAND BETWEEN APLITIC GRANITE (LOWER RIGHT) AND DIABASE PORPHYRY (UPPER LEFT). FROM EAST MINE. × 18.

on one side into diabase porphyry, on the other into aplite. The writer can see only one explanation for this specimen, and that is a change in the mineralizing solutions from intermixed mineralizers and acidic rock material to aplite.

It is on the basis of such occurrences, and for the reasons given, that

the writer believes: that the aplite and ore are phenomena produced at approximately the same time; that both were formed from a mixture of mineralizers and rock material derived from the granitic magma; and that the iron oxides, lime and the major part of the silica, held in aqueous solutions above the critical temperature, diffused into the wall rock, leaving the residual rock material in the fissures as aplitic dikes.

## VII. DETAILED DESCRIPTION OF THE ORE DEPOSITS

In order to test the hypothesis stated above in regard to the genesis of the ore deposits, it must be applied to the different occurrences of ore in the district. To do this, a detailed description of the more carefully studied mines will be given, and the mode of formation of the orebodies discussed.

### Ocania Mine

The Ocania is the most westerly of the Juragua Iron Co.'s mines. It lies on a steep hillside, at an elevation of from 860 to 1,085 ft.

Fig. 20.—Specimen from Ocania Mine, Level 2. The Dark Material is Garnet. The Light Material is Wollastonite. The Structure is Attributed to Diffusion. Actual Size.

The wall rocks are coarse, diabase porphyry, volcanic agglomerates, and marble. The contact between the granitic and diabasic rocks lies on the south slope of the hill, at an elevation of about 700 ft. A quartz-bearing, andesitic dike is found in the mine itself. The ore forms a series of lenses that strike in a northwesterly direction, and have a steeply inclined dip to the southwest. These lenses are made up almost entirely of hematite and quartz, with little or no magnetite. There is considerable pyrite through the ore in various places, and, at the edge of the limestone, chalcopyrite and manganese oxide are found. Some fragments of diabase, with native copper in them, were observed near the limestone. Between the ore and the limestone are masses of garnet rock. Near the bottom of the orebody, as exposed on the southeast side of the mine, was

found a rock (Fig. 20) which shows, in part, alternating layers of brown garnet, and a greenish mineral which the microscope shows to be wollastonite. As the accompanying microphotograph (Fig. 21) of the contact between the garnet and wollastonite shows, the two bands have an interlocking structure, perpendicular to the boundary. The black crystals in the wollastonite band are magnetite. The garnet does not show crystal edges. Toward the center it is very dusty, with undeterminable inclusions. The wollastonite forms an aggregate of poorly developed, prismatic crystals. A somewhat similar structure, made up of magnetite and actinolite, is shown in a specimen from the Juragua Mines described by W. Lindgren. He says:

> "The texture of this specimen strongly suggests diffusion banding, such as might form in a material, a limestone perhaps, freely penetrated by hot iron-bearing solution."

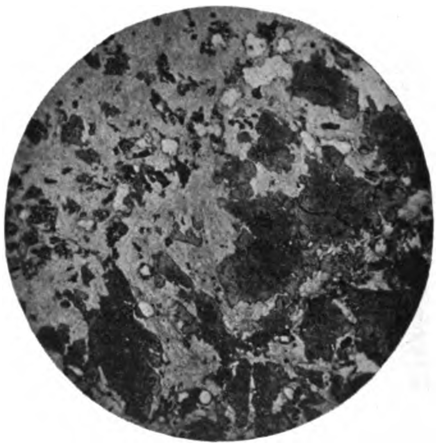


Fig. 21.—Contact between Garnet and Wollastonite in Rock Showing Diffusion Structure. From Ocania Mine. × 18. Magnetite—black. Garnet—dark. Wollastonite—grey. Holes in thin section—white.

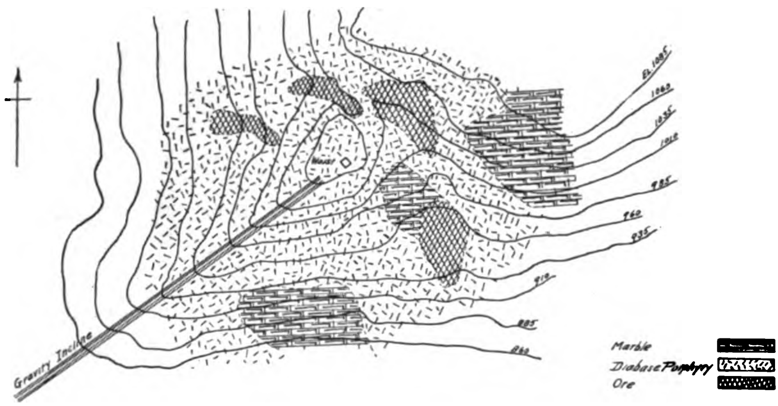
In the case of the Ocania specimen the solutions were probably lime-bearing and lower in iron. Whether the original rock was limestone or diabase porphyry, it is not possible to tell. It is true of this mine that the mineralization effects have spread further into the diabasic rocks than into the marble. The clean white marble is separated from the ore by only a few feet of garnet rock, while the diabase is garnetized, near the ore, and epidotized for tens of feet from the ore. The ore itself is so complete an alteration of whatever wall rock it replaces that there is no *prima facie* evidence of what it originally was. The pyritization is later than the principal period of ore deposition, the pyrite forming in fractures and cavities in the ore. The chalcopyrite accompanies the pyrite.

Surface action, producing partial hydration of the ore and considerable infiltration of secondary calcite, obscures the evidence.

Northeast of the mine workings is the largest exposure of marble found in the district. It is cut by sills of an entirely epidotized, basic, igneous rock. It is separated from the orebodies by massive garnet rock, or highly garnetized and silicified diabasic rock. The map (Fig. 22) shows the relation of orebodies to wall rock. The commercial ore is sharply bounded, but the mineralization is not.

The striking features in this mine are the absence of magnetite and of highly acid wall rocks, and the clear evidence that the diabasic rocks are more diffusely mineralized than the marble. At the same time the resemblance of the blocks of ore to engulfed limestone blocks or to roof pendants, is noticeable.

If the Ocania Mine were the only one studied, it would be difficult for the writer to defend his hypothesis. There are no aplitic or pegmatitic




OCANIA MINE
Fig. 22.

rocks in evidence. Except for one dike of doubtful age, no quartz-bearing igneous rock is exposed in the mine. There is abundant limestone.

There are also some features that are not explained by any theory involving the diabase as the ore-bringing rock. The diffuse mineralization of the diabasic porphyry indicates a hydrothermal effect, later than the consolidation of the diabase. It is difficult to ascribe to the diabase porphyry a hematitizing effect on included limestone, or even a magnetitization free of chrome or titanium minerals, with a superimposed hematitization. The fact that the larger part of the marble has suffered no mineralization is also unexplained by any theory that makes the overwhelming of the limestone by the diabase the cause of mineralization.

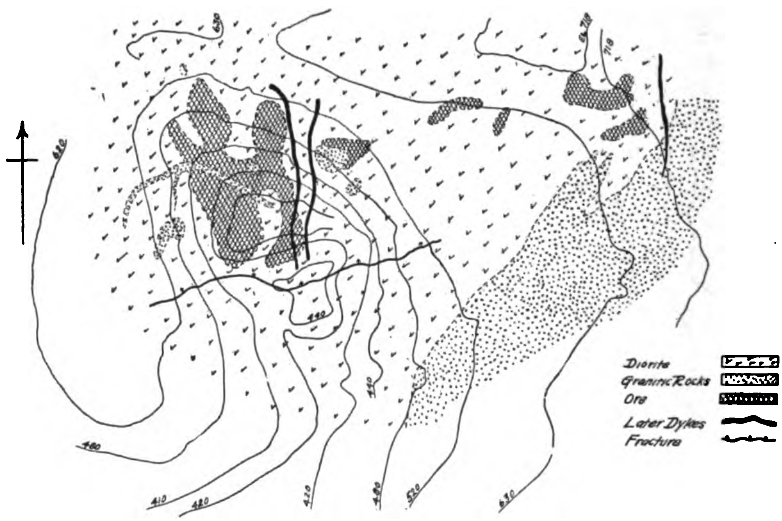
On the other hand, if the solutions are supposed to come from the granite, the difficulties of explanation are not so great. A fissure in the diabase near the limestone, localized the ore and the mineralization in

both formations. The distance of the ore exposed from the parent magma accounts for the preponderance of hematite over magnetite.

### West Five Mine

This mine is the furthest west of the mines in the vicinity of Firmeza, and lies at an elevation of from 420 to 718 ft. There is a tunnel under the mine at a level of 299 ft.

The ore lies in flat or in steeply inclined lenses. It is almost entirely magnetite, the enclosing wall rock diorite, with considerable, more or less aplitic, granite through and near the ore. The diorite varies from




WEST 'S' MINE
FIG. 23.

the even, granular type shown in Fig. 8 to a porphyritic type. No fragmental diabasic rocks occur, nor is there any trace of limestone.

The rocks from the aplitic dike, in the west side of the mine, were described under the granitic rocks. The connection between aplite and ore is shown on a small scale in the photograph (Fig. 17) of a small dike of aplite from the tunnel under the mine. The magnetite and epidote lying immediately next to the aplite, merge gradually into the granular diorite. There can be no question of any rocks being involved other than the diorite and the aplite. Their inter-relationship has been discussed in considering the question of genesis. Considerable garnet is found in the mine workings, and its occurrence is similar to the magnetite and epidote. In one small exposure the position of the garnet, as a contact effect in the diorite, next to the granite, is clearly shown. This

is on the north side of the bottom level. The epidotization and silicification are more widely diffused from the granite than are the magnetite and garnet. One specimen, from the lowest level, showed an intergrowth of apatite with magnetite, and garnet and epidote, cut by later veinlets of epidote (Fig. 14). This is the only case in which apatite amounted to more than a very minor accessory.

On the south side of the mine is an east-west fault steeply inclined to the north. It is obviously post-mineral, and appears not to be of very great throw.

Summing up the evidence from this mine we find: no limestone; much aplitic or porphyritic granite; almost no specularite; but much magnetite, garnet, epidote, quartz—all in diorite.

The ore deposits of this mine offer the clearest evidence that limestone could not have been the dominant cause determining the locus of their deposition. There is no limestone in the immediate vicinity. The transition from ore to diorite is gradual. The mineralogy shows none of the well-crystallized calcite that is so abundant in the ore deposits that were obviously formed in limestone.

The intimate relation between ore and aplite is also well shown here. The dike exposed in the west side of the workings has led other observers to believe the granitic apophyses later than the ore, but as shown under the discussion of the various theories, the same phenomena are interpreted by the writer to indicate the contemporaneous formation of the ore and the aplite.

All the phenomena can be explained on the hypothesis advanced. That is that the aplite, the magnetite, and the lime silicates are all the effect of the one mineralizing period.

## Loma Alta Mine

The Loma Alta workings lie at the top of a hill, just east of West Five Mine. They are at an elevation of from 900 to 1,000 ft. As they have been practically abandoned for some time, weathering agencies have obscured much of the evidence. Cropping out on the east side and pitching under the workings at a flat angle, is a sheet-like mass of porphyritic, pegmatitic granite. In the southwest corner of the workings is recognizable diabase porphyry. What is left of the ore is earthy hematite lying on the granite with some kaolin and chlorite. All around the workings on the hill are rounded boulders of more or less pure magnetite. The association of the granite, diabase and ore in this mine is suggestive, but the evidence is too obscured to be conclusive. The earthy hematite undoubtedly owes its origin to the weathering of the primary, crystalline iron oxides. There is no reason to believe that the process did not involve the formation of limonite and its dehydration.

### West Three Mine

An exposure in this mine, which lies on the same hill, south of and below Loma Alta Mine, is worth describing. It is a mass of magnetite, in diabase, directly next to a vertical pegmatitic granite dike. The rocks are all fairly unaltered, and the contact relationship well shown. There is no reason to suspect that any limestone was involved.

### West Four Mine

This mine is located northeast of Loma Alta, at an elevation of from 645 to 935 ft. At the north end of the mine, in the upper levels, is a mass of clear white marble, entirely unmineralized. The wall rock in the upper levels is diabase porphyry; in the lower levels it is diorite. The diabase and diorite are both cut by somewhat porphyritic and pegmatitic granite. The ore is magnetite, and hematite and magnetite, and lies in lenses, whose major axes vary from a vertical to a horizontal position, on top of and next to the granite. These lenses occur mostly in the diorite, partly in the diabase, but not, so far as exposed, in contact with the marble. The entire series, diabase, diorite, granite, ore, is cut by a large dioritic dike. The mineralization effects are no different from those in West Five Mine. The chief difference is that there is marble exposed in the West Four workings. So far as can be determined it has had no direct effect on the formation of the ore deposits.

### West One Mine

The West One workings are located north of Firmeza along the railroad tract. Exposed for a long time and not much worked of late years, they offer a poor field for study. Altered diabase and diorite, highly mineralized, are the wall rocks. Neither limestones nor granite are in evidence. From one of the cuts that is being worked, come the well-developed pyrite crystals, of which one is shown in Fig. 13. This exposure shows the pyrite all through the magnetite in cavities and along minute fractures. It establishes the age of the pyritization as definitely post-magnetitization.

Fig. 24 is a photograph of a drawing made with a camera lucida, from a thin section of a rock from West One Mine. The specimen is dark and dense, and shows patches of magnetite. In thin section the rock proves to be a diabasic porphyry, partly replaced by magnetite. The replacement is apparently independent of the mineralogy of the replaced rock. The magnetite occurs in patches in the groundmass, or in the phenocrysts, or partly in each.

West One Mine offers no new evidence in regard to the genetic problem. It does offer, however, an excellent field for the study of the pyritization.

## East Mine

The East Mine itself lies east of Firmeza, at an elevation of from 440 to 700 ft., but it cannot be considered apart from the small cut just to the north at about 800 ft. elevation, known locally as the North Mine.

The large orebodies of this mine are made up of specularite and magnetite, with much epidote quartz and garnet. They lie in the lower levels of the mine around bosses of aplitic granite, in diorite and diabase porphyry. Fig. 18 is a photograph and Fig. 19 a microphotograph of a lime-iron garnet rim along a small aplite dike, in recognizable diabase porphyry. It shows the entire ability of the mineralizers that accompanied the aplite to garnetize the diabase.

Fig. 24.—Photograph of a Drawing Made with a Camera Lucida, from a Thin Section of a Specimen from West One Mine. It Shows a Diabase Porphyry, Partly Replaced by Magnetite (Solid Black). The Phenocryst is Plagioclase. The Lath-Shaped Crystals Are Plagioclase. The Groundmass is Granular Magnetite and Chlorite.

(In the upper half of the drawing the interstitial magnetite has been left out.)

Fig. 26 is a microphotograph of a thin section of a completely epidoized and silicified diabase. The specimen comes from within 2 ft. to the one shown in Fig. 15. The two show successive stages in the hydrothermal metamorphism.

In the upper levels, extending toward the North Mine workings, are several included sheets of completely marmorized limestone. The North Mine workings, now abandoned, show remnants of orebodies. This ore is different from any found in the mines so far described. Well-developed rosettes of specularite blades in garnet quartz matrix; well-crystallized epidote in abundance; coarse quartz; much recrystallized calcite; in short all the components are found of a typical contact-metamorphic deposit in limestone.

In the East Mine, as in the others, later andesitic and dioritic dikes are in evidence. A northwest-southeast fault, pitching to the northeast, has cut the ore and aplitic granite as well as the older rocks. It has brought the most basic of the granitic rocks, the quartz-bearing diorite, in contact with the ore. That the specimen described as a wall rock of the East Mine by J. F. Kemp[39] came from this mass of quartz-bearing diorite seems probable. The rock in which the ore is found is of a much finer grain.

In the East Mine itself there is no evidence that limestone had any share in forming the orebodies. In the workings known as the North Mine, limestone is without doubt the host rock of the orebodies.

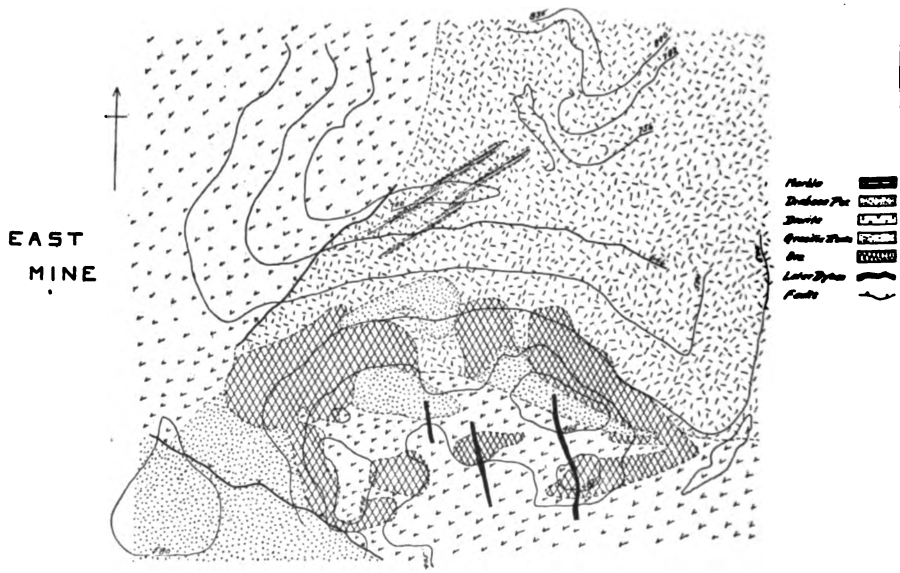



Fig. 25.

The ore in the main workings, with the associated granitic rocks, was formed at centers of intense mineralization. The ore deposits of the North Mine were formed at a greater distance from the centers of mineralization. The aplite dikes in the terrane between the two are the filling of the fissures through which the solutions traveled.

### Chicharron (North East) Mine

Located about ¾ mile northeast of the East Mine and now entirely idle, are the workings of the old Chicharron Mine. This mine is well north of the zone in which most of the mines lie, and in some respects differs from the others.

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[39] *Op. cit.*, p. 15.

The face of the mine is a cliff of diabase, over which the water of the Benevolencia River falls. This diabase is more or less mineralized, and

Fig. 26.—Diabase Porphyry Completely Replaced by Epidote and Quartz. From East Mine. × 18. Quartz—light. Epidote—dark.

replaced by magnetite, wollastonite, and epidote. The brook bed follows along the bottom of the cliff, and with an overgrown talus pile, separates the cliff from a mass of marble. Cutting the marble is a dike of

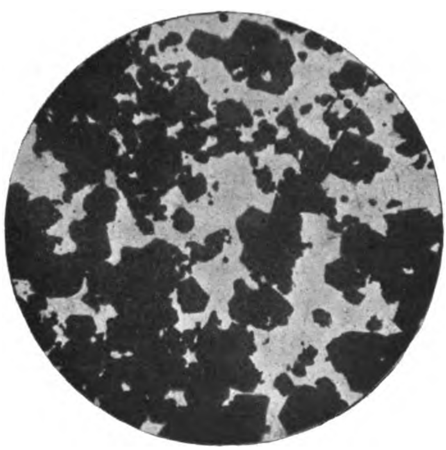

Fig. 27.—Ore from Chicharron Mine. × 18. Magnetite—black. Wollastonite—white.

magnetite, with interstitial wollastonite in small amounts (Fig. 27). A. C. Spencer describes this occurrence as follows:[40]

"At this place the magnetite seems to be an intrusive dike cutting across a mass of crystalline limestone into which it sends a short apophysis."

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[40] *Op. cit.*, p. 80.

The magnetite probably formed a dike, instead of a more diffuse form of deposit, because while the temperature was high, the pressure was not. As a result, the diffusive power of the mineralizers was not great, although they could form high-temperature minerals.

In the Chicharron Mine, also, the later dikes are in evidence.

### Estancia Mines

To the east of East Mine, in the mineralized zone, lies Estancia Hill. All around the hill at from 700 to 800 ft. elevation lie mine workings. The ore in these workings is either magnetite and specularite replacing diabase, or it is the hematite-garnet-calcite combination that is eloquent of contact-metamorphic limestone, though no limestone is left. Around most of the hill, and all of the south side, is a railroad track at an elevation of about 500 to 550 ft., and it is nearly all cut in a porphyritic granite, in which micropegmatite predominates. In the valley north of Estancia Hill granite is exposed along the trail. The dump of an old tunnel run into Estancia Hill for 600 ft. at an elevation of 695 ft. is granite, and A. W. Gaumer, the Juragua Iron Co.'s chief engineer, informed the writer that all the tunnel was in the same rock. South of the railroad track runs the lower trail to Concordia, and it is all in diorite, or coarse diabase, which has been mapped as diorite.

All this evidence seems to indicate that a sheet of porphyritic granite intruded into diabasic rock with included marble, to form a sill. The orebodies at the upper contact of the granite formed in that rock to which the mineralizers had access.

### Concordia Mine

This, the most easterly of the Juragua Iron Co.'s mines, offers no new evidence. It is the lowest of the mines, being at an elevation of from 300 to 450 ft. The rock and ore association is magnetite in diorite with granite near by.

The ore deposits are lenticular, and as a rule lie in an almost horizontal position. No limestone is in evidence.

## VIII. GEOLOGIC HISTORY

The evidence from which the geologic history of the Firmeza district must be deduced is scanty. Almost all the field work was done in the immediate vicinity of the mines, and little search was made in the surrounding country for evidence which might give completeness to the geologic history. Enough material was found to make possible a sequential arrangement of the geologic events that produced the present

topographic, petrologic, and mineralogic features of the district. These geologic events will be discussed in the order of their occurrence, as follows:

1. Sedimentation.
2. Igneous cycle (including ore deposition).
3. Uplift with tilting and erosion.
4. Deposition of coral limestone.
5. Emergence.
6. Submergence.

## Sedimentation

The formation which appears to be the oldest in the district is the marble. As has been shown in the discussion of the rocks, this marble is Mesozoic, and probably Cretaceous in age. The constituents of the floor upon which the limestones now represented by marble were deposited, cannot be determined; nor is it possible to estimate the thickness of these early sediments, because of the lack of any continuous section. The limestone found at the highest elevation is a tuff-limestone. Because of its position it may be considered as the youngest of these early sediments, and because of the volcanic fragments embedded in it at the time of its deposition, it may be considered as representing the connecting link between the period of sedimentation, and the initiation of the igneous cycle.

## Igneous Cycle

The igneous cycle began with a period of volcanic activity, to which the clastic igneous rocks of basic nature bear witness. These volcanic rocks form the southern slopes of the Sierra Maestra range, and the tops of some of the foothills. They contain remnants of interbedded limestones, and must have formed contemporaneously with the latter part of the period of sedimentation.

The next event was the invasion of these volcanic rocks and sediments by a magma of basic composition. Of this "parent" magma the present representatives are the diabasic and dioritic rocks. These rocks were left in their present position partly by the chilling of the magma, and to a lesser degree, by intrusion into the sediments and volcanic rocks.

The invasion of basic magma was followed by a long period of differentiation. During this period the liquid residue in the magma chamber became more acidic, and charged with mineralizers. This acidic liquid, with its burden of mineralizers, invaded the earlier, chilled, basic, igneous rocks in the form of a granitic batholith.

Further differentiation, resulting in a concentration of the mineralizers, ended in the injection of the mineralizers, with admixed rock material, into cooling cracks in the basic rocks above the batholith.

There followed a separation of the highly heated mineralizers from their admixed rock material, which resulted in the formation of the ore deposits and the aplitic and pegmatitic apophyses.

It is uncertain to what part of the igneous cycle the marmorization of the limestone should be ascribed, because all the marble exposures are near or in igneous intrusive rocks, which range from diabase to granite. Any one of these may have been competent to produce the marmorization. It seems probable, however, that because it was the earliest, the invasion of the diabasic magma is responsible for the greater part of the marmorization.

What events were taking place at the surface during the igneous cycle is a question to which no answer has been found. The formation of cooling cracks, however, in the basic igneous rocks, seems to indicate that they were not very far from the surface at the time of ore deposition.

The last episode in the igneous cycle was the intrusion of basic dikes into fissures formed in all the igneous rocks and their associated ore deposits.

## Uplift, Tilting, and Erosion

Because of the almost complete marmorization of the older limestones, their bedding is difficult to determine. But the pitch of the extrusive, and the few visible remnants of stratification in the marble, indicate a definite dip to the southeast. Evidence of uplift is found in the fact that limestone occurs at elevations up to 1,400 ft. above sea level.

As has been shown in the discussion of the areal geology, there is no conclusive evidence as to the exact time relation between the igneous cycle and the period of uplift and tilting, but because pressure is to a certain extent a function of depth, it seems probable that the ore deposits were all formed in an approximately horizontal plane. Their present position along an inclined plane would, in that case, be some measure of the differential elevation. Meager and inconclusive as the evidence is, it must be considered, and until evidence to the contrary is found, the uplift and tilting should be regarded as later than the igneous activity.

The coral limestones themselves show no evidence of tilting and this fact places the tilting as older than the coral limestones—that is pre-Pleistocene. Emergence has, however, continued.

Fragments of iron ore in the base of the later coastal limestones prove that this period of uplift was accompanied by erosion sufficient to cut into the ore deposits.

So far as the ore belt is concerned, the later record is one of continued erosion. Whatever else may have happened to that part of the district, no record remains except one of superficial alteration and erosion. The coral limestones and the valley topography show that dynamic forces were not entirely quiescent.

## Deposition of Coral Limestone

The deposition of the Pleistocene-Recent coral limestone began when the land was about 350 ft. lower than at present. Since then there has been oscillation with predominant emergence.

## Emergence

The sea-cut cliffs in coral limestone, rising in terraces from the coast to an elevation of 350 ft., are evidence of the periodic emergence of the land since deposition of the limestone began. What the extent of the emergence was cannot be determined on the evidence in the district, because it has been masked by the most recent geologic event—submergence.

## Submergence

A. C. Spencer,[41] judging from other parts of the island, places the amount of the submergence at from 40 to 70 ft. If the thickness of the fluviatile deposits in the east-west valley could be determined, it would give an accurate measure of the amount of submergence in this district.

For reasons that have been discussed in detail under the interpretation of the topography, the writer believes that this submergence represents a movement of the sea.

The geologic history of this district is a record of quiet activity. There is no regional metamorphism or profound fracturing, such as would be expected if vast diastrophic movements had taken place. The only metamorphism that has left a record was due either to igneous or to superficial causes. The only fracturing recorded is the result of the shrinkage of igneous rocks in cooling, and gives evidence of little or no local movement. Igneous forces, acting through long periods of time, dominated the rock formations. Slow uplift, alternating with periods of almost complete quiet, and accompanied by much erosion, formed the topographic features.

# IX. ECONOMIC APPLICATION

Any geological investigation of more than a purely academic interest must answer some questions of a commercial nature. The present study of the ore deposits of the Firmeza district, by its inquiry into their origin and extent, is able to answer eertain questions which are of economic interest. These questions refer to the continuation of the deposits in depth, a possible change in their composition if they do extend downward, and the favorable locus for the search for more ore.

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[41] *Op. cit.*, p. 34.

## Continuation of the Orebodies in Depth

Since the mineralization came from below, from the granitic massif, there is no reason to believe that the ore will not extend at least to the granite. On the other hand, there is no reason to believe that the separation of the mineralizers from the granite was incomplete, so that no ore can be looked for in the granite. Where the ore lies on top of sheet-like granitic masses, like the Estancia and Loma Alta deposits, a definite limit in depth is in sight, but in workings where the massive granite is not in evidence, extensive development in depth seems probable.

## Character of the Ore in Depth

The pyritization in the ores at Firmeza has made the owners of the ore deposits fear that the ores would change to pyrite in depth. However, the pyritization is later and independent of depth. There is no reason to believe that the ores found at greater depth will be any higher in sulphur than those now being mined in the lower levels of the larger mines. The lower sulphur content of the ores close to the surface is due to superficial oxidation of the pyrite. Below the level of surface oxidation the sulphur content may be expected to remain fairly constant.

The only change that seems probable is an increase in the proportion of magnetite to hematite, as the workings approach the deeper centers of mineralization.

## Favorable Locus for Exploration Work

The problem of exploration work may be considered in two ways—positively and negatively. Both are of fundamental importance, although frequently the question of where to look for more ore is considered, while the equally important question of where not to look for ore is neglected. It is just as necessary to discover in which localities the search for ore will be fruitless and therefore any exploration work would be valueless, as it is to know where the search for ore will be profitable.

Obviously, rocks formed later than the ore, like the coastal limestone, cannot be expected to contain ore deposits. The granitic massif, from which the ore solutions are thought to have escaped, is also to be regarded as barren ground.

On the other hand, the granitic apophyses, aplites and pegmatitic granite are the best guides as to the location of orebodies. It is their upper surface which merits the most careful search.

Since the orebodies contain much magnetite, the magnetic survey is undoubtedly the best guide to their location. But a careful study of

the formations is necessary to interpret such a survey, and to decide what areas of magnetic attraction are most valuable, and also what areas, in spite of a lack of this attraction, offer fields for development.

The economic application of geologic evidence will be successful only as the interpretation of the geologic phenomena is correct. At the same time the ultimate test of any geologic theory is its application in the field.

## Bibliography

F. F. Chisholm: Iron-Ore Beds in the Province of Santiago, Cuba. *Proceeding of the Colorado Scientific Society*, vol. 3, pp. 259–263 (1888).

C. Willard Hayes, T. Wayland Vaughan, and Arthur C. Spencer: *Report on a Geological Reconnaissance of Cuba*, made under the direction of General Leonard Wood, Military Governor, pp. 69–83 (Washington, 1901).

R. T. Hill: Tertiary and Later History of the Island of Cuba. *American Journal of Science*, Ser. 3, vol. 48, p. 203 (1894).

Geology of Jamaica. *Bulletin* of the Museum of Comparative Zoology at Harvard, vol. 34, pp. 92–100.

J. F. Kemp: The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. *Trans.*, vol. 53, pp. 3–38 (1916).

James F. Kimball: The Iron-Ore Range of the Santiago District of Cuba. *Trans.*, vol. 13, pp. 613–634 (1884–85).

Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Science*, Ser. 3, vol. 28, pp. 416–429 (1884).

W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vol. 53, pp. 40–59 (1916).

J. T. Singewald and B. LeRoy Miller: The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, pp. 67–74 (1916).

Arthur C. Spencer: The Iron Ores of Santiago, Cuba. *Engineering and Mining Journal*, vol. 72, pp. 633–634 (Nov. 16, 1901).

The Mayari and Daiquiri Iron-Ore Mines. Articles published in *Iron Age*, vol. 80, pp. 421–426 (Aug. 15, 1907) and vol. 81, pp. 1149–1157 (April 9, 1908).

D. B. Whitaker: Letter in regard to production. *Engineering and Mining Journal*, vol. 97, p. 677 (March, 1914).

H. Wedding: Die Eisenerze der Insel Cuba. *Stahl und Eisen*, vol. 12, No. 12, pp. 545–550 (June 15, 1892).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

**DISCUSSION OF THIS PAPER IS INVITED.** It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Effect of Sulphur on Low-Carbon Steel

BY CARLE R. HAYWARD,* S. B., CAMBRIDGE, MASS.

(New York Meeting, February, 1917)

SULPHUR has long been one of the banes of the steel manufacturer and often no effort and expense have been spared in order to reduce it to a small per cent. in the finished product. This condition is due to a general conviction that in many cases where steels have failed in service, sulphur has been the cause. But there has been a growing feeling in recent years that the verdict against sulphur has been unnecessarily severe. In cases of segregation it was present in augmented amounts along with other impurities, but it had not caused the segregation. High sulphur in pig iron is caused by poor furnace conditions and the sulphur is merely one indication of an iron that has not been properly reduced. No amount of subsequent treatment under oxidizing conditions in the open-hearth furnace can remedy the defects, although the per cent. of sulphur may be considerably reduced. In other words, the causes of bad steel can frequently be traced back to bad pig iron, and sulphur is merely one indication that the pig iron is bad. The writer recently visited a steel plant where a mass of evidence had been accumulated which substantiated this fact, and the superintendent was emphatic in stating that high sulphur was not harmful provided the steel was not otherwise poor due to insufficient reduction in the blast furnace.

The presence of a moderate amount of sulphur is desirable from the standpoint of the man who machines the steel. The low-sulphur material drags and the production of a smooth surface is very difficult. A slight increase in sulphur enables the machinist to produce a smooth surface without difficulty.

Since, therefore, such large quantities of steel are subjected to machining, it becomes highly important that the sulphur controversy should be settled, and if its presence is proved to be harmless the ban on it should be lifted.

Among the recent papers on the effect of sulphur on steel is one by Dr. J. S. Unger, Manager of the Central Research Bureau, Carnegie Steel

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* Assistant Professor of Mining Engineering and Metallurgy, Massachusetts Institute of Technology.

Co.[1] The results of an exhaustive series of tests are given and the conclusion states: "The author does not advocate paying no attention whatever to sulphur content in steel but believes firmly that a steel containing less than 0.100 per cent. is not necessarily bad, and that it will show little, if any, difference in quality when compared with the same steel of much lower sulphur, other conditions being the same."

The present investigation was undertaken at the suggestion of A. H. Annan of the Rhode Island Tool Co., who coöperated by furnishing the steel and machining the specimens. The results are presented as a contribution to the general knowledge on the subject.

### *Steels Used*

It was planned to use three steels of different sulphur content but with the other elements the same. The manganese was an exception, however, for with this element part is in the form of MnS existing free in the steel and the remainder is dissolved in the steel. It is evident, therefore, that the manganese should vary but that the amount in excess of MnS should be constant in the different steels.

The steels finally selected were in the form of ¾-in. round bars. Two bars of each grade were required to furnish sufficient specimens. The analyses are shown in Table 1.

TABLE 1

| Mark | Carbon, Per Cent. | Total Manganese, Per Cent. | Excess Manganese, Per Cent. | Phosphorus, Per Cent. | Silicon, Per Cent. | Sulphur, Per Cent. |
|---|---|---|---|---|---|---|
| 1 | 0.18 | 0.55 | 0.48 | 0.007 | 0.01 | 0.038 |
| 1A | 0.18 | 0.57 | 0.50 | 0.009 | 0.02 | 0.041 |
| 2 | 0.17 | 0.67 | 0.52 | 0.008 | 0.01 | 0.086 |
| 2A | 0.18 | 0.70 | 0.55 | 0.010 | 0.03 | 0.087 |
| 3 | 0.18 | 0.80 | 0.54 | 0.006 | 0.02 | 0.152 |
| 3A | 0.17 | 0.80 | 0.55 | 0.011 | 0.03 | 0.148 |

### *Heat Treatment*

In order to make a comparison of the steels under different conditions it was decided to heat all the specimens to a temperature just above the critical range, quench in water, and reheat different lots to 300°, 400°, 500° and 600°C. respectively. For this purpose the bars were cut into 7-in. lengths, which was sufficient for tensile specimens and specimens for microscopic examination.

The furnace used is shown in Fig. 1. The muffle is made of alundum

[1] *Iron Age*, No. 97, pp. 146–150 (1916).

and is 2 in. high, 9 in. wide and 16 in. long. It is wound with No. 15 excello resistance wire.

In order to obtain uniform heating of the specimens, they were supported on an asbestos rack, as shown. Asbestos shields were placed on the sides and ends of the rack so that the specimens were practically in a muffle within a muffle.

A platinum platinum-rhodium thermocouple was introduced through the back of the muffle into the center of the heating chamber and connected to a Siemens and Halske recording galvanometer.

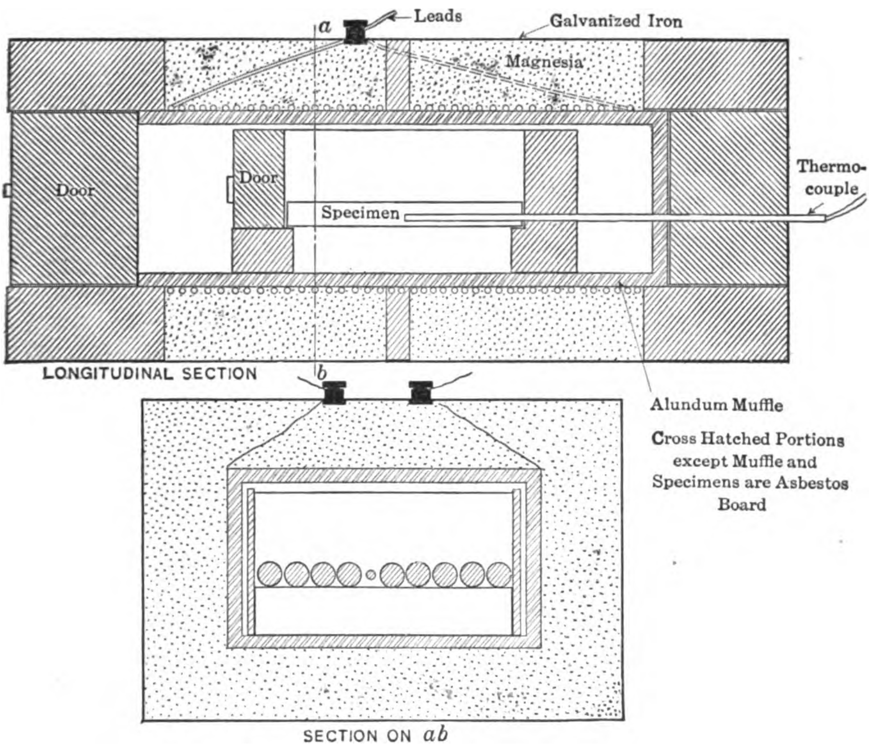
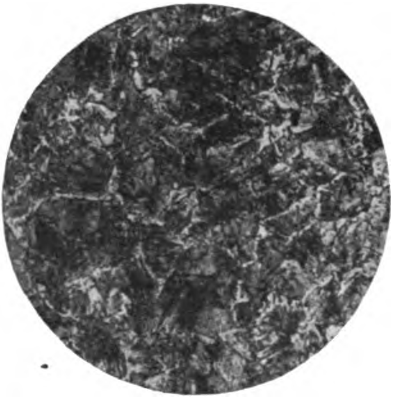
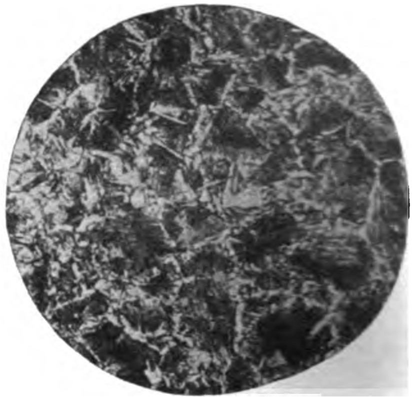



FIG. 1.—HEAT-TREATMENT FURNACE.

The procedure in heat treatment was as follows: The furnace was heated to 880° C. and nine specimens introduced. This caused the temperature to fall to 550° C. and it took about 40 min. to again reach 880°. When the latter temperature was reached it was maintained constant for 15 min., after which the specimens, except as noted below, were quenched in water and a new lot introduced into the furnace. This was continued until all but 18 of the specimens had been treated. Nine of these, consisting of three high-, three medium- and three low-sulphur steels, were removed from the furnace and allowed to cool in air. The



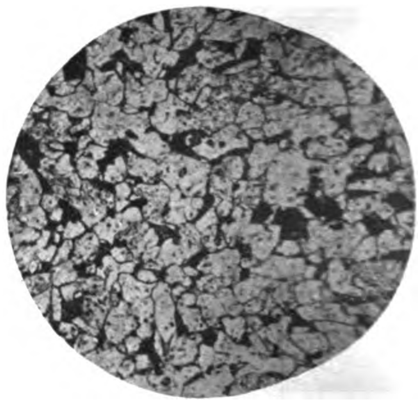
*a*.—High-Sulphur Steel.



*b*.—Low-Sulphur Steel.

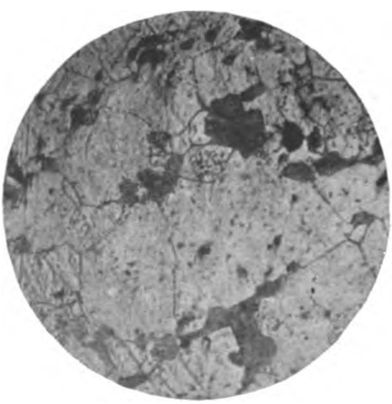
Quenched in Ice Water.

*c*.—High-Sulphur Steel.

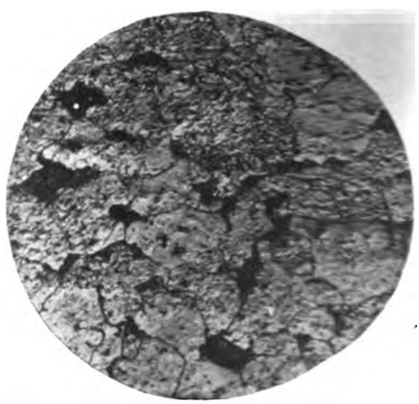


*d*.—Low-Sulphur Steel.

Cooled in Air.



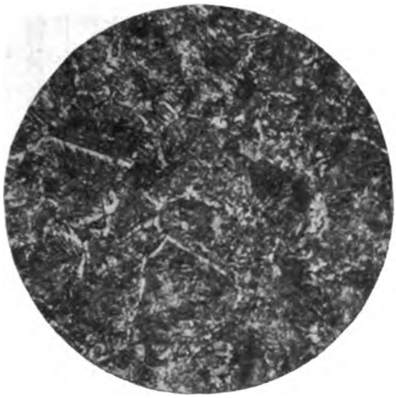
*e*.—High-Sulphur Steel.



*f*.—Low-Sulphur Steel.

Cooled in Furnace.

PLATE I.



*a.*—High-Sulphur Steel.

*b.*—Low-Sulphur Steel.

Reheated to 300°.

*c.*—High-Sulphur Steel.

*d.*—Low-Sulphur Steel.

Reheated to 500°.

PLATE II.

## Table 2

| No. | Mark | | Yield Point, Lb. per Sq. In. | Breaking Load, Lb. per Sq. In. | Elongation, Per Cent. in 2 In. | Reduction of Area, Per Cent. |
|---|---|---|---|---|---|---|
| 1 | 1A | Heated to 880° C. and quenched in ice water. | 58,500 | 86,250 | 23.0 | 64.7 |
| 2 | 1A | Heated to 880° C. and quenched in ice water. | 58,000 | 85,250 | 23.5 | 62.4 |
| 3 | 1A | Heated to 880° C. and quenched in ice water. | 57,250 | 85,500 | 23.5 | 62.4 |
| Average...... | | ........................................ | 57,900 | 85,600 | 23.3 | 63.1 |
| 4 | 2A | Heated to 880° C. and quenched in ice water. | 53,000 | 79,750 | 26.0 | 69.3 |
| 5 | 2A | Heated to 880° C. and quenched in ice water. | 56,250 | 77,000 | 29.5 | 71.4 |
| 6 | 2A | Heated to 880° C. and quenched in ice water. | 51,500 | 76,250 | 27.5 | 69.3 |
| Average...... | | ........................................ | 53,600 | 77,700 | 27.7 | 70.0 |
| 7 | 3A | Heated to 880° C. and quenched in ice water. | 56,500 | 84,250 | 22.0 | 59.9 |
| 8 | 3A | Heated to 880° C. and quenched in ice water. | 79,250* | 89,250 | 22.5 | 59.9 |
| 9 | 3A | Heated to 880° C. and quenched in ice water. | 57,300 | 84,500 | 22.0 | 62.4 |
| Average...... | | ........................................ | 56,900 | 86,000 | 22.2 | 60.7 |
| 10 | 1A | Heated to 880° and cooled in still air...... | 41,000 | 58,500 | 41.0 | 67.0 |
| 11 | 1A | Heated to 880° and cooled in still air...... | 46,750 | 58,500 | 41.5 | 69.3 |
| 12 | 1A | Heated to 880° and cooled in still air...... | 41,750 | 58,250 | 41.5 | 67.0 |
| Average...... | | ........................................ | 43,150 | 58,400 | 41.3 | 67.8 |
| 13 | 2A | Heated to 880° and cooled in still air...... | 45,750 | 58,000 | 41.0 | 67.0 |
| 14 | 2A | Heated to 880° and cooled in still air...... | 46,250 | 58,000 | 40.5 | 69.3 |
| 15 | 2A | Heated to 880° and cooled in still air...... | 39,750* | 58,000 | 41.5 | 69.3 |
| Average...... | | ........................................ | 46,000 | 58,000 | 41.3 | 68.5 |
| 16 | 3A | Heated to 880° and cooled in still air........ | 40,000 | 59,500 | 38.0 | 62.4 |
| 17 | 3A | Heated to 880° and cooled in still air........ | 41,500 | 59,500 | 39.5 | 64.7 |
| 18 | 3A | Heated to 880° and cooled in still air........ | 46,250 | 59,250 | 39.5 | 64.7 |
| Average...... | | ........................................ | 42,600 | 59,400 | 39.0 | 63.9 |
| 19 | 1A | Heated to 880° and cooled in furnace........ | 38,000 | 52,500 | 40.5 | 62.4 |
| 20 | 1A | Heated to 880° and cooled in furnace........ | 37,750 | 52,750 | 40.5 | 64.7 |
| 21 | 1A | Heated to 880° and cooled in furnace........ | 33,750 | 52,500 | 40.0 | 62.4 |
| Average...... | | ........................................ | 36,500 | 52,600 | 40.3 | 63.2 |
| 22 | 2A | Heated to 880° and cooled in furnace........ | 37,000 | 53,000 | 40.0 | 62.4 |
| 23 | 2A | Heated to 880° and cooled in furnace........ | 38,750 | 53,000 | 40.0 | 62.4 |
| 24 | 2A | Heated to 880° and cooled in furnace........ | 34,500 | 53,000 | 41.0 | 59.9 |
| Average...... | | ........................................ | 36,750 | 53,000 | 40.3 | 61.6 |
| 25 | 3A | Heated to 880° and cooled in furnace........ | 37,500 | 55,000 | 37.5 | 59.9 |
| 26 | 3A | Heated to 880° and cooled in furnace........ | 35,500 | 55,000 | 38.5 | 59.9 |
| 27 | 3A | Heated to 880° and cooled in furnace........ | 32,250 | 55,000 | 38.0 | 69.3* |
| Average...... | | ........................................ | 35,080 | 55,000 | 38.0 | 63.0 |
| 301 | 1 | Quenched at 880°. Reheated to 300°........ | 53,750 | 76,000 | 25.5 | 69.3 |
| 302 | 1 | Quenched at 880°. Reheated to 300°........ | 58,000 | 76,000 | 26.0 | 69.3 |
| 303 | 1 | Quenched at 880°. Reheated to 300°........ | 56,000 | 75,000 | 28.0 | 71.4 |
| Average...... | | ........................................ | 55,900 | 75,700 | 26.5 | 70.0 |
| 304 | 2 | Quenched at 880°. Reheated to 300°........ | 57,500 | 73,250 | 28.5 | 71.4 |
| 305 | 2 | Quenched at 880°. Reheated to 300°........ | 61,250 | 78,000 | 26.5 | 71.4 |
| 306 | 2 | Quenched at 880°. Reheated to 300°........ | 59,750 | 73,750 | 25.0 | 71.4 |
| Average...... | | ........................................ | 59,500 | 75,000 | 26.7 | 71.4 |
| 307 | 3 | Quenched at 880°. Reheated to 300°........ | 57,000 | 80,250 | 23.5 | 62.4 |
| 308 | 3 | Quenched at 880°. Reheated to 300°........ | 56,500 | 83,000 | 21.5 | 59.9 |
| 309 | 3 | Quenched at 880°. Reheated to 300°........ | 59,000 | 85,000 | 19.5 | 59.9 |
| Average...... | | ........................................ | 57,500 | 82,750 | 21.5 | 60.7 |
| 401 | 1 | Quenched at 880°. Reheated to 400°........ | 63,750 | 76,500 | 28.5 | 71.4 |
| 402 | 1 | Quenched at 880°. Reheated to 400°........ | 58,500 | 76,000 | 29.5 | 73.5 |
| 403 | 1 | Quenched at 880°. Reheated to 400°........ | 58,750 | 76,450 | 28.5 | 71.4 |
| Average...... | | ........................................ | 60,300 | 76,300 | 28.8 | 72.1 |
| 404 | 2 | Quenched at 880°. Reheated to 400°........ | 62,750 | 75,000 | 26.5 | 69.3 |
| 405 | 2 | Quenched at 880°. Reheated to 400°........ | 63,750 | 80,000 | 24.0 | 69.3 |
| 406 | 2 | Quenched at 880°. Reheated to 400°........ | 61,250 | 74,750 | 28.5 | 71.4 |
| Average...... | | ........................................ | 62,600 | 76,600 | 26.3 | 70.0 |
| 407 | 3 | Quenched at 880°. Reheated to 400°........ | 64,000 | 84,000 | 21.5 | 62.4 |
| 408 | 3 | Quenched at 880°. Reheated to 400°........ | 62,000 | 82,750 | 21.5 | 62.4 |
| 409 | 3 | Quenched at 880°. Reheated to 400°........ | 61,250 | 82,750 | 24.0 | 62.4 |
| Average...... | | ........................................ | 62,400 | 83,150 | 22.3 | 62.4 |
| 501 | 1 | Quenched at 880°. Reheated to 500°........ | 55,250 | 70,750 | 33.5 | 73.5 |
| 502 | 1 | Quenched at 880°. Reheated to 500°........ | 58,250 | 73,250 | 31.5 | 73.5 |
| 503 | 1 | Quenched at 880°. Reheated to 500°........ | 57,500 | 74,000 | 31.0 | 73.5 |
| Average...... | | ........................................ | 57,000 | 72,650 | 32.0 | 73.5 |
| 504 | 2 | Quenched at 880°. Reheated to 500°........ | 60,750 | 73,250 | 29.0 | 71.4 |
| 505 | 2 | Quenched at 880°. Reheated to 500°........ | 61,250 | 74,000 | 32.0 | 71.4 |
| 506 | 2 | Quenched at 880°. Reheated to 500°........ | 57,000 | 72,500 | 26.0 | 71.4 |
| Average...... | | ........................................ | 59,650 | 73,250 | 29.0 | 71.4 |
| 507 | 3 | Quenched at 880°. Reheated to 500°........ | 61,500 | 78,750 | 26.0 | 67.0 |
| 508 | 3 | Quenched at 880°. Reheated to 500°........ | 60,500 | 77,500 | 26.0 | 67.0 |
| 509 | 3 | Quenched at 880°. Reheated to 500°........ | 60,750 | 77,750 | 26.0 | 67.0 |
| Average...... | | ........................................ | 60,900 | 78,000 | 26.0 | 67.0 |
| 601 | 1 | Quenched at 880°. Reheated to 600°........ | 53,500 | 69,250 | 35.0 | 75.5 |
| 602 | 1 | Quenched at 880°. Reheated to 600°........ | 51,000 | 67,750 | 35.0 | 75.5 |
| 603 | 1 | Quenched at 880°. Reheated to 600°........ | 52,000 | 69,500 | 32.5 | 75.5 |
| Average...... | | ........................................ | 52,150 | 68,850 | 34.2 | 75.5 |
| 604 | 2A | Quenched at 880°. Reheated to 600°........ | 54,500 | 70,500 | 35.5 | 73.5 |
| 605 | 2A | Quenched at 880°. Reheated to 600°........ | 52,500 | 72,000 | 32.5 | 73.5 |
| 606 | 2A | Quenched at 880°. Reheated to 600°........ | 53,750 | 70,750 | 31.0 | 73.5 |
| Average...... | | ........................................ | 53,550 | 71,100 | 33.0 | 73.5 |
| 607 | 3 | Quenched at 880°. Reheated to 600°........ | 55,000 | 74,250 | 31.0 | 71.4 |
| 608 | 3 | Quenched at 880°. Reheated to 600°........ | 55,000 | 73,750 | 30.5 | 71.4 |
| 609 | 3 | Quenched at 880°. Reheated to 600°........ | 55,500 | 75,750 | 30.0 | 69.3 |
| Average...... | | ........................................ | 55,150 | 74,600 | 30.5 | 70.7 |

* Results abnormal. Not included in average.

remaining nine, consisting of three of each sulphur content, were allowed to cool in the furnace.

In drawing the quenched specimens at the various temperatures, the procedure was as follows: The furnace was heated to 600° C. and nine specimens (three of each sulphur content) were introduced. The temperature fell and it required about 30 min. to come back to 600°. It was maintained here for 10 min. and then the specimens were withdrawn and quenched in water. The furnace was cooled to 500° C. and another set of nine specimens was introduced. When the temperature had regained 500° it was maintained constant for 10 min. and then the specimens were withdrawn and quenched. Following a similar procedure, sets of nine specimens were treated at 400° and 300° C. respectively.

### *Tensile Tests*

After sawing off ¾ in. from each piece for microscopic examination, standard test specimens were prepared with 2-in. gage length, 0.505-in. diameter and threaded ends. These were pulled in an Olsen machine in the testing laboratory of the Massachusetts Institute of Technology. The results obtained are given in Table 2.

For more ready comparison, the averages are retabulated in Table 3. The designations *L*, *M* and *H* refer to low, medium and high sulphur content.

TABLE 3

| | Yield Point, Lb. per Sq. In. | Breaking Load, Lb. per Sq. In. | Elongation, Per Cent. in 2 In. | Reduction of Area, Per Cent. |
|---|---|---|---|---|
| *L* Heated to 860° C. and quenched in ice water | 57,900 | 85,600 | 23.5 | 63.1 |
| *M* Heated to 860° C. and quenched in ice water | 53,600 | 77,700 | 27.7 | 70.0 |
| *H* Heated to 860° C. and quenched in ice water | 56,900 | 86,000 | 22.0 | 60.7 |
| *L* Heated to 860° C. and cooled in still air | 43,200 | 58,400 | 41.5 | 67.8 |
| *M* Heated to 860° C. and cooled in still air | 46,000 | 58,000 | 41.3 | 68.5 |
| *H* Heated to 860° C. and cooled in still air | 42,600 | 59,400 | 39.0 | 63.9 |
| *L* Heated to 860° C. and cooled in furnace | 36,500 | 52,600 | 40.5 | 63.2 |
| *M* Heated to 860° C. and cooled in furnace | 36,800 | 53,000 | 40.3 | 61.6 |
| *H* Heated to 860° C. and cooled in furnace | 35,100 | 55,000 | 38.0 | 63.0 |
| *L* Quenched at 860° in ice water. Reheated to 300° | 55,900 | 75,700 | 26.5 | 70.0 |
| *M* Quenched at 860° in ice water. Reheated to 300° | 59,500 | 75,000 | 26.7 | 71.4 |
| *H* Quenched at 860° in ice water. Reheated to 300° | 57,500 | 82,750 | 21.5 | 60.7 |
| *L* Quenched at 860° in ice water. Reheated to 400° | 60,300 | 76,300 | 28.8 | 72.1 |
| *M* Quenched at 860° in ice water. Reheated to 400° | 62,600 | 76,600 | 26.3 | 70.0 |
| *H* Quenched at 860° in ice water. Reheated to 400° | 62,400 | 83,200 | 22.3 | 62.4 |
| *L* Quenched at 860° in ice water. Reheated to 500° | 57,000 | 72,700 | 32.0 | 73.5 |
| *M* Quenched at 860° in ice water. Reheated to 500° | 59,700 | 73,300 | 29.0 | 71.8 |
| *H* Quenched at 860° in ice water. Reheated to 500° | 60,900 | 78,000 | 26.0 | 67.0 |
| *L* Quenched at 860° in ice water. Reheated to 600° | 52,200 | 68,800 | 34.2 | 75.5 |
| *M* Quenched at 860° in ice water. Reheated to 600° | 53,600 | 71,100 | 33.0 | 73.5 |
| *H* Quenched at 860° in ice water. Reheated to 600° | 55,200 | 74,600 | 30.5 | 70.7 |

There is not sufficient variation in the results to make an effective plot, but Table 4 summarizes the figures by giving the order in which they occur from high to low.

TABLE 4

| | Yield Point | | | Breaking Load | | | Elongation | | | Reduction of Area | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | *L* | *M* | *H* | *L* | *M* | *H* | *L* | *M* | *H* | *L* | *M* | *H* |
| Quenched in ice water.. | 1 | 3 | 2 | 2 | 3 | 1 | 3 | 1 | 2 | 3 | 1 | 2 |
| Cooled in still air...... | 2 | 1 | 3 | 2 | 3 | 1 | 1 | 2 | 3 | 2 | 1 | 3 |
| Cooled in furnace ..... | 2 | 1 | 3 | 3 | 2 | 1 | 1 | 2 | 3 | 2 | 3 | 1 |
| Reheated to 300°...... | 3 | 1 | 2 | 2 | 3 | 1 | 3 | 1 | 2 | 3 | 1 | 2 |
| Reheated to 400°...... | 3 | 1 | 2 | 3 | 2 | 1 | 1 | 2 | 3 | 1 | 2 | 3 |
| Reheated to 500°...... | 3 | 2 | 1 | 3 | 2 | 1 | 1 | 2 | 3 | 1 | 2 | 3 |
| Reheated to 600°...... | 3 | 2 | 1 | 3 | 2 | 1 | 1 | 2 | 3 | 1 | 2 | 3 |
| Totals.............. | 17 | 11 | 14 | 18 | 17 | 7 | 11 | 12 | 19 | 13 | 12 | 17 |
| Order of totals...... | 3 | 1 | 2 | 3 | 2 | 1 | 1 | 2 | 3 | 2 | 1 | 3 |

### *Photomicrographs*

One specimen from each set was polished, etched in a solution of 10 per cent. $HNO_3$ in alcohol and examined under the microscope. The photographs show a representative spot on each specimen magnified 300 diameters.

The specimens were studied at higher magnifications than those given in the micrographs, but no further information was obtained. The lower magnifications gave a better idea of the general structure and, therefore, only these were reproduced in the paper.

### *Shock Tests*

After concluding the tensile tests, it was thought desirable to determine the effect of varying sulphur content on specimens subjected to shock.

Through the kindness of the testing department of the Watertown Arsenal, the use of their Charpy machine was obtained for this purpose.

The remaining ¾-in. stock of bars marked 1, 2 and 3 was sawed into 2-in. lengths and the specimens subjected to heat treatment in sets of three under the same conditions that obtained in the case of the tensile specimens. After heat treatment, the specimens were machined to conform to the following specifications:

Length 55 mm., cross-section 10 by 10 mm., notch across one side, midway between the ends, 1 mm. wide. Radius of cutter edge, ⅔ mm.

The Charpy machine consists of a heavy pendulum which drops from a fixed height, strikes the specimen supported at each end and breaks

TABLE 5

| Treatment | No. | Mark | Breaking Shock, Ft.-Lb. per Sq. In. | Average |
|---|---|---|---|---|
| Heated to 480° and quenched in cold water...... | 1 | 1A | 494 | |
| Heated to 480° and quenched in cold water...... | 2 | 1A | 545 | |
| Heated to 480° and quenched in cold water...... | 3 | 1A | 540 | 526 |
| Heated to 480° and quenched in cold water...... | 4 | 2A | 523 | |
| Heated to 480° and quenched in cold water...... | 5 | 2A | 488 | |
| Heated to 480° and quenched in cold water...... | 6 | 2A | 542 | 518 |
| Heated to 480° and quenched in cold water...... | 7 | 3A | 432 | |
| Heated to 480° and quenched in cold water...... | 8 | 3A | 507 | |
| Heated to 480° and quenched in cold water...... | 9 | 3A | 470 | 470 |
| Heated to 480° and cooled in still air............ | 10 | 1A | 356* | |
| Heated to 480° and cooled in still air............ | 11 | 1A | 460 | |
| Heated to 480° and cooled in still air............ | 12 | 1A | 452 | 456 |
| Heated to 480° and cooled in still air............ | 13 | 2A | 512 | |
| Heated to 480° and cooled in still air............ | 14 | 2A | 521 | |
| Heated to 480° and cooled in still air............ | 15 | 2A | 524 | 519 |
| Heated to 480° and cooled in still air............ | 16 | 3A | 418 | |
| Heated to 480° and cooled in still air............ | 17 | 3A | 398 | |
| Heated to 480° and cooled in still air............ | 18 | 3A | 433 | 416 |
| Heated to 480° and cooled in furnace............ | 19 | 1A | 258 | |
| Heated to 480° and cooled in furnace............ | 20 | 1A | 284 | |
| Heated to 480° and cooled in furnace............ | 21 | 1A | 295 | 279 |
| Heated to 480° and cooled in furnace............ | 22 | 2A | 353 | |
| Heated to 480° and cooled in furnace............ | 23 | 2A | 357 | |
| Heated to 480° and cooled in furnace............ | 24 | 2A | 259* | 355 |
| Heated to 480° and cooled in furnace............ | 25 | 3A | 283 | |
| Heated to 480° and cooled in furnace............ | 26 | 3A | 256 | |
| Heated to 480° and cooled in furnace............ | 27 | 3A | 279 | 273 |
| Quenched at 480°. Reheated to 300°............ | 301 | 1A | 633 | |
| Quenched at 480°. Reheated to 300°............ | 302 | 1A | 447* | |
| Quenched at 480°. Reheated to 300°............ | 303 | 1A | 578 | 605 |
| Quenched at 480°. Reheated to 300°............ | 304 | 2A | 562 | |
| Quenched at 480°. Reheated to 300°............ | 305 | 2A | 537 | |
| Quenched at 480°. Reheated to 300°............ | 306 | 2A | 530 | 543 |
| Quenched at 480°. Reheated to 300°............ | 307 | 3A | 462 | |
| Quenched at 480°. Reheated to 300°............ | 308 | 3A | 422 | |
| Quenched at 480°. Reheated to 300°............ | 309 | 3A | 434 | 439 |
| Quenched at 480°. Reheated to 400°............ | 401 | 1A | 613 | |
| Quenched at 480°. Reheated to 400°............ | 402 | 1A | 597 | |
| Quenched at 480°. Reheated to 400°............ | 403 | 1A | 582 | 597 |
| Quenched at 480°. Reheated to 400°............ | 404 | 2A | 540 | |
| Quenched at 480°. Reheated to 400°............ | 405 | 2A | 541 | |
| Quenched at 480°. Reheated to 400°............ | 406 | 2A | 556 | 546 |
| Quenched at 480°. Reheated to 400°............ | 407 | 3A | 474 | |
| Quenched at 480°. Reheated to 400°............ | 408 | 3A | 447 | |
| Quenched at 480°. Reheated to 400°............ | 409 | 3A | 429 | 450 |
| Quenched at 480°. Reheated to 500°............ | 501 | 1A | 439* | |
| Quenched at 480°. Reheated to 500°............ | 502 | 1A | 694 | |
| Quenched at 480°. Reheated to 500°............ | 503 | 1A | 670 | 682 |
| Quenched at 480°. Reheated to 500°............ | 504 | 2A | 560 | |
| Quenched at 480°. Reheated to 500°............ | 505 | 2A | 572 | |
| Quenched at 480°. Reheated to 500°............ | 506 | 2A | 552 | 561 |
| Quenched at 480°. Reheated to 500°............ | 507 | 3A | 495 | |
| Quenched at 480°. Reheated to 500°............ | 508 | 3A | 456 | |
| Quenched at 480°. Reheated to 500°............ | 509 | 3A | 469 | 473 |
| Quenched at 480°. Reheated to 600°............ | 601 | 1A | 726 | |
| Quenched at 480°. Reheated to 600°............ | 602 | 1A | 728 | |
| Quenched at 480°. Reheated to 600°............ | 603 | 1A | 726 | 727 |
| Quenched at 480°. Reheated to 600°............ | 604 | 2A | 584 | |
| Quenched at 480°. Reheated to 600°............ | 605 | 2A | 604 | |
| Quenched at 480°. Reheated to 600°............ | 606 | 2A | 602 | 597 |
| Quenched at 480°. Reheated to 600°............ | 494 | 3A | 494 | |
| Quenched at 480°. Reheated to 600°............ | 502 | 3A | 502 | |
| Quenched at 480°. Reheated to 600°............ | 521 | 3A | 521 | 506 |

* Abnormal. Not included in average.

it at the notch. The pendulum then continues its swing and the height it reaches is registered. Knowing the weight of the pendulum, the height it falls and the height it rises, a simple calculation gives the energy consumed in breaking the specimen.

The results obtained are given in Table 5.

## *Conclusions*

Table 4, which expresses the summary of the tensile tests, shows that the high-sulphur steel has for each treatment the highest breaking load while the yield point ranks first for two treatments, second for three and third for two. From this we may conclude that the sulphur does not lower the tensile strength.

The figures for elongation and reduction of area show that there is little difference in ductility between the low- and medium-sulphur steels, but the ductility of the high-sulphur steel is slightly lower than the other two for most of the treatments.

The average figures for the shock tests, except for the air- and furnace-cooled specimens, are highest for each treatment in the case of the low-sulphur steels and lowest for each treatment for the high-sulphur steels. The widest difference appears in the steels which have been quenched and reheated.

It is difficult to draw definite conclusions from the results because of the newness of the shock test and the difference of opinion among engineers regarding its value. The tensile tests are not unfavorable to steels with moderate amounts of sulphur, while the shock tests show a decided falling off in strength as the sulphur increases. Until the interpretation of the results from the Charpy machine is more fully understood, it is impossible to say to which set of tests the most importance should be attached.

Further light might be thrown on the subject by making alternate stress or fatigue tests. It would be important to learn whether the results would confirm the tensile or shock tests. Unfortunately, however, the stock of steels used in the previous work was exhausted, and whatever the results of the fatigue tests there would be an uncertainty in their interpretation because of difference in stock. It was, therefore, decided not to include this series in the present investigation.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Recrystallization after Plastic Deformation

BY HENRY M. HOWE, LL. D.,* BEDFORD HILLS, N. Y.

(New York Meeting, February, 1917)

THIS paper is a discussion of the extremely valuable one of Mathewson and Phillips, The Recrystallization of Cold-Worked Alpha Brass on Annealing,[1] which not only gives us a wealth of important data reached with great intelligence, but also shows both uncommon powers of imagination and a perfectly fair spirit.

Let me try to analyze some of their results, and to enunciate some of the laws to which these results point. For brevity I refer to them as "The Authors." The page numbers in parentheses refer to their paper.

1. *The Visible Aspects of Recrystallization.*—Plastic deformation in a single direction, as in cold-rolling, draws the grains out in that direction, that is, inequiaxes them, without destroying their apparent individuality. Each grain seems to endure the drawing out without changing either its volume or its orientation materially, for the etching tint of each grain seems to be as uniform after as before the drawing out, save for the "etch bands,"[2] shown in Figs. *A* to *D* of the Authors' Plate I.

As the drawn-out metal is heated progressively, on reaching a certain "disintegration temperature" these old grains seem to break up, probably into submicroscopic fragments, and this apparent disintegration is followed by a coalescence of these fragments, so that their number decreases and their size increases, till they may become very much coarser than before the drawing out. Ruder[3] coarsened the grains of silicon steel in this way till single grains reached an area of 50 cm. (7.75 sq. in.).

The deformation itself probably breaks each grain up into many very small fragments, but the fact that each grain continues to have a uniform etching tint different from that of its neighbors indicates that these fragments retain the initial common orientation of the grain. The apparent disintegration which we observe, then, is rather an abandonment of community of orientation for individuality of orientation, by the fragments formed earlier during the deformation. Hence we

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* Emeritus Professor of Metallurgy, Columbia University.

[1] *Bulletin* No. 109, p. 1 (January, 1916).

[2] Rosenhain calls these "slip bands," and I have called them "X bands," to avoid any preconception as to their nature (*The Metallography of Steel and Cast Iron*, p. 452).

[3] *Trans.*, vol. 47 p. 569.

should speak rather of a "reorientation" than of a "disintegration" temperature. The reorientation and regrowth may jointly be called "recrystallization."

The recrystallization is accompanied by softening, that is to say, though the first raising of the temperature after the cold-rolling may increase the hardening which the cold-rolling caused, yet with further rise of temperature the hardness falls progressively to that of the normal metal.

While the Authors would explain the nature of disintegration and regrowth, their main objective seems to be to trace the connection between these two processes and this softening.

2. *Our natural conception of disintegration and grain growth* I take to be that the plastic deformation itself breaks each grain up into many fragments as already pointed out, of which the number increases and the size decreases as the deformation increases; that each fragment retains for the time the initial orientation of the grain from which it is broken, as is shown by the uniform etching tint of all these fragments of a given grain; but that each is sheathed completely with metal made amorphous by the friction during the act of deformation. This amorphous metal cements the whole together. On this hypothesis when the deformed metal is heated, five distinct changes occur side by side, though not necessarily simultaneously or in any fixed proportion to each other. These are, first, a progressive reabsorption of the amorphous sheaths by the crystalline fragments which they inclose; second, a suspected progressive but undescribed change in the condition of so much of these sheaths as has not yet become reabsorbed, or of the crystalline grain fragments thus sheathed, or of both; third, an apparent disintegration of the initial grains, which is really only a reorientation of the grain fragments already formed; fourth, their progressive growth by coalescence, with consequent lessening of their number; and fifth, the progressive release of stress as the mobility increases and the existing elastic limit falls correspondingly. The first three may be grouped together as "recrystallization." Let us in (3) to (7) glance at the evidence that these five agencies are at work.

3. *The reabsorption of the sheaths* with rising temperature is inferred from the observation that, by heating before etching, the etch bands, which seems to represent an accumulation of this same amorphous metal along the slipping planes, are made first fainter and finally undetectable.

4. *The reorientation of the grain fragments* is shown by the apparent disintegration which the grains undergo, for this disintegration means only that the several fragments of a given grain cease to have common orientation and hence to etch alike, and instead adopt each a new orientation, so that they etch differently and hence appear as distinct grains or grain fragments. It is not clear that this distinction between grains

and grain fragments need be insisted upon in what follows, for indeed each fragment seems for all intents and purposes to be an independent grain, once its reorientation has occurred.

5. *Grain Growth.*—The progressive growth of the grain fragments is familiar to all through the coarsening that occurs on heating cold-rolled or otherwise plastically deformed metal.

6. *The progressive release of stress* could no doubt be proved by noting how much, after reheating to various temperatures, the specimen bends out of shape spontaneously on cutting away a fixed part of it, so as to destroy the balance of internal stresses remaining after the initial cold-rolling itself. We may assume confidently that the degree of this spontaneous bending decreases progressively as the temperature of annealing, prior to cutting away, rises.

7. *A progressive change in the condition of the amorphous sheaths* or of the crystalline fragments is neither shown nor known. I have inferred it as a probable cause of the perturbations which the cohesion, as indicated by the elastic limit and hardness of cold-rolled iron, both pure and cementite-bearing (mild steel), undergoes on gentle heating. On either slight heating or mere rest such iron may soften materially or may harden greatly, according to the intensity of the previous deformation.[4]

8. *In what Proportion is the Softening Caused by Annealing due to Each of These Five Influences?*—Though the present evidence does not enable us to give a full answer to this question, some inferences concerning it may be drawn. Let us see, in (9) to (14), what importance we are inclined to assign each of these five supposed agencies.

9. *Stress.*—The relief of stress is not likely to cause a very large fraction of the softening induced by annealing; first because it is incompetent to explain the great hardening and embrittlement which gentle heating causes in cold-worked iron; and second because the progress of the soften-

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[4] *The Metallography of Steel and Cast Iron*, pp. 365 and 452 to 458. The Authors' explanation (*Bulletin* No. 109, p. 7 (January, 1916)) that the strengthening represents the removal of stress is not competent to explain either the tripling of the hardness which, as I find, mere rest after deformation may cause, or the simultaneous great loss of ductility. Thus Stromeyer (*Journal of the Iron and Steel Institute*, vol. 75, No. 3, p. 92 (1907)) finds that the ability of the outer surface of a nicked test-piece to elongate before rupture may fall from 60 per cent. to 6 per cent. on long rest, and in one case to 6 and in another even to 3 per cent. on heating to 100° for 15 min. Nor can we accept the explanation (Gulliver, *Journal of the Institute of Metals*, No. 2, p. 220 (1909)) that this hardening, strengthening, and embrittlement represent "a change from the instable $Fe_3C$, present in commercial steels, to the stable $Fe_3C$," because I find that practically carbonless steel, with only 0.01 per cent. of carbon, hardens like and perhaps even more than common mild steel of 0.147 per cent. of carbon, on heating to 100° after plastic deformation (*op. cit.*, p. 365). This strengthening and hardening which slight heating causes so markedly in iron occur only in a slight degree in brass, as is shown concordantly by the results of Bengough and the Authors (pp. 3 and 6). Indeed it is so slight as to have been overlooked by Grard (p. 3).

ing, both absolute and relative, in the Authors' results is not such as would be expected to be caused by stress-relief. Thus, on the theory that stress-relief is the chief cause of the softening, it is hard to see why a progressive increase in the degree of cold-rolling should lead either to the progressive and marked lowering of the temperature of most rapid softening, or to the marked narrowing of the temperature range in which most of this softening occurs, as recorded by the Authors (Fig. 3), or why, after 8 per cent. reduction by rolling, the softening should be only 1.4 units between 350° and 450°, yet should rise to 2.1 units between 450° and 550° and to 2.5 between 550° and 650° (Table IV).

10. *Reorientation or Disintegration.*—If we are right in conceiving that visible disintegration represents the breaking away from the initial orientation and substituting a new one in each of the several grain fragments which thereby become visible, then before any large fraction of the fragments have thus changed, the change should become visible, substituting in the earliest of the reorienting grains finer uniformly etching areas for the initial coarser ones. On this conception, disintegration is not likely to be the chief cause of the softening, because after the Authors' 4 and 12 per cent. reductions more than one-third of the total softening occurs from 100° to 200° before the reorientation becomes visible (Table IV).

Again, it is hard to reconcile the belief that most of the softening is due to reorientation with the fact that, after their 35 per cent. reduction, 80 per cent. of the total softening occurs after reorientation has become visible, and successive softenings each of about 10 per cent. of the total are found at temperatures, 200°, 300°, and even 400° higher, though the micrographs suggest no such temperature-distribution of the reorientation.

11. *Grain Growth.*—If we conceive, as seems reasonable, that the grain fragments cannot coalesce with each other to any important degree before they reorient themselves and thereby cause visible disintegration, then the great softening which, after the 4 per cent. and 8 per cent. reductions, precedes visible reorientation, cannot be referred to grain growth. This disinclines us to refer the softening as a whole chiefly to grain growth, in spite of the very great softening that accompanies the replacement of finer by coarser-grained gold in the important experiments of Fahrenwald,[6] a softening so great that we hesitate to refer it solely to difference in grain size. The Authors clearly share this belief (p. 31).

Another indication that grain growth is not the chief cause of the softening is the discordance between the curves of grain size and of hardness in Fig. 4. To judge from Plate XI there is a great retardation of grain growth from the 350°–450° to the 450°–550° range, and Fig. 4

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[6] *Bulletin*, No. 109, p. 122 (January, 1916).

tells us that there is such a retardation from the 450°–550° to the 550°–650° range, yet in passing these ranges the softening undergoes no corresponding retardation, nor does the tensile weakening.

12. *Reabsorption of the Amorphous Sheaths.*—Thin as these sheaths seem to be, if we refer to them the great hardening that cold work is capable of giving we must infer that their total volume forms a very appreciable fraction of the whole mass, an inference which is the more probable in view of the marked changes which cold work causes in the solution pressure, density, and other physical properties of the mass as a whole.

It is reasonable to hold that this amorphous metal is harder than the crystalline, because it lacks the cleavages and partings which enfeeble every crystalline mass. Hence its progressive reabsorption on heating and rest would itself naturally cause a corresponding softening of the mass as a whole. It may be that the rate of reabsorption of this amorphous metal on heating and rest varies in such a way as to contribute greatly to the variations observed in the rate of softening of the whole mass.

13. *Change of Condition of the Amorphous Sheaths or of the Crystalline Fragments which they Inclose.*—The great hardening, strengthening, and embrittlement which pure iron undergoes on rest or on slight heating after cold deformation are not readily referred to any of the four agencies which we have just considered in (9) to (12). In order to explain them I attempt to refer them to unknown changes in the nature of either the amorphous sheaths or crystalline fragments which they inclose, or both. If we grant that either or both these agencies are capable of causing these very striking changes, we naturally infer that they may also contribute to a very important degree to the softening which follows this hardening in iron on higher heating, and occurs without prominent hardening in the other metals.

14. *To sum up,* stress-relief and grain growth are not likely to contribute greatly to the softening caused by heating overstrained metal. Reorientation may contribute greatly, but it is not likely to be the chief cause. The reabsorption of the amorphous sheaths inclosing the grain fragments, and a progressive change in the condition of those sheaths of the crystalline metal in those fragments, are possible important causes. Beyond this preliminary speculation the present data do not justify our going.

It is probable that at least two agencies are at work, with effects which differ greatly, because, in the case of iron, rest and gentle heating, while effacing one of the effects of the cold work, the etch bands, simultaneously increase another and simultaneous effect of that same cold work, the hardening which it causes.

15. *Correspondence between the Authors' Conjectured Progress of*

*Recrystallization and Their Observed Progress of Softening.*—First we must understand the Authors' term "fragmental resolution," for which I venture to suggest "limiting grain size." They hold that, for each temperature, there is a certain "limiting grain size," "limiting" in the sense that all grain or grain fragments which are smaller than it start to grow toward it, but cannot grow beyond it, nor can grains of this or any greater size grow at this temperature. As far as I understand "fragmental resolution," it means this "limiting grain size," a

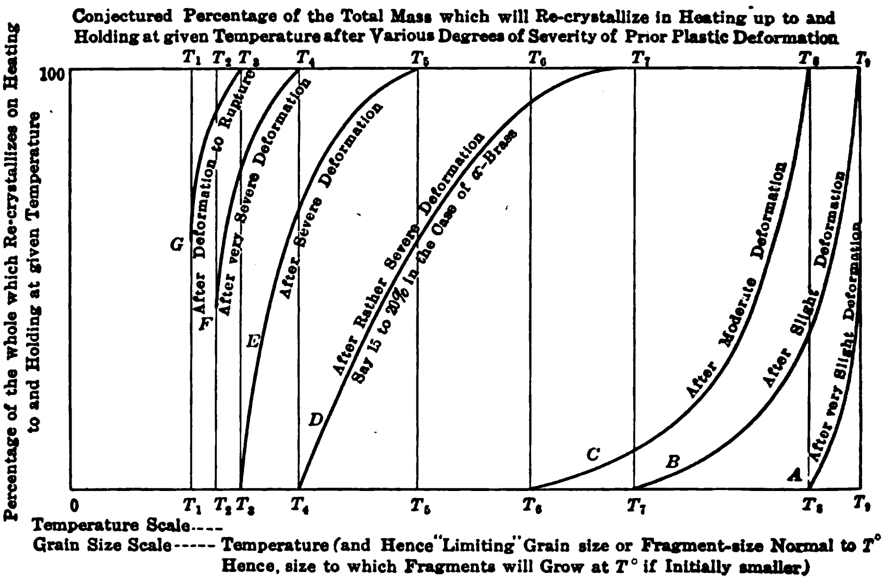


FIG. *A*.—CONJECTURED PROGRESS OF RECRYSTALLIZATION OF COLD-WORKED BRASS ON PROGRESSIVE HEATING.

term which I offer reluctantly and only because I find "fragmental resolution" confusing.

The Authors give in their Fig. 2 a most interesting generalization from their hypothesis. This, their conjecture as to the progress of recrystallization, that is of the return to the crystalline state of the metal made amorphous by the deformation, may perhaps be grasped more easily as relettered in my Fig. *A*. Here ordinates show what percentage of that part of the metal which has thus been made amorphous can recrystallize in rising to given temperature and staying there 30 min. Abscissæ measure both temperature and the limiting grain size for given temperature. Far from attempting quantitative accuracy, this sketch tries only to show the general family to which the curves are expected to belong.

According to this forecast, the greater the previous reduction by cold-rolling has been, (1) the lower is the temperature at which the curves leave the horizontal axis, that is the temperature at which recrystallization

starts, and (2) the steeper is the beginning of the curve. Further (3), whereas after severe deformation the curves are convex upward, after slight deformation they are concave upward. Hence the curves for great reductions form a sheaf of which the top fans out to the right, whereas those of slight deformation form a sheaf of which the bottom fans out to the left.

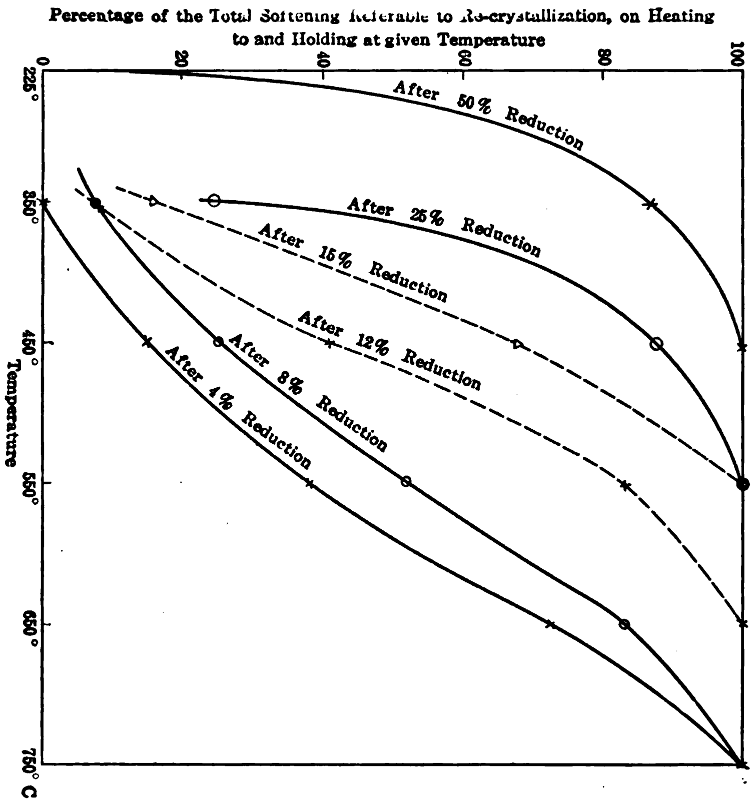


FIG. B.—OBSERVED PROGRESS OF SOFTENING OF COLD-ROLLED BRASS.

We have no data showing the true progress of this recrystallization, and hence we cannot check their forecast as to the relation of this progress to the temperature of annealing. But in my Fig. *B*, I attempt to check it indirectly by showing how the progress of that fraction of the softening which may be referred to recrystallization is related to the annealing temperature. Any such attempt must be very rough, because we do not know what fraction of the softening is due to recrystallization and what to grain growth. In view of the relatively small effect of grain size as such on hardness, I assume that the greater part of the softening is due to recrystallization, and for the purposes of this rough check I assume

that the temperatures given in line 2 of my Table *A* are those at which recrystallization becomes complete, on the ground that these are the ones at which the softening is nearly complete.

Because the heatings underlying their Table IV were of 30 min., because a temperature of 255° was needed in their Table II to cause an appreciable softening after 40 per cent. reduction in about this time; and because 30 min. at 200° caused a slight hardening after this reduction; I set $T°_1$, the temperature needed to cause the first softening after 50 per cent. reduction, at 255°.

The curves of Fig. *B*, plotted on these assumptions, tend to support the second and third of the Authors' predictions, that the steepness of the beginning of the curve increases with the previous reduction, and that the curves for great reduction are convex upward whereas those for light reductions are concave upward.

Their first contention, that the temperature at which recrystallization starts is the lower the greater the deformation has been, has such good direct experimental support that it is to be accepted in spite of its not being clearly indicated by these curves. Note, for instance, that the shape of the 8 and 12 per cent. reduction curves suggests that they leave the horizontal axis at temperatures rather below than above those of the 15 and 25 per cent. curves.

16. *Is There an Equilibrium Grain Size for Each Temperature?*—The Authors' assertion to this effect (p. 18) may well be taken rather cautiously in our present ignorance as to the mechanism of grain growth. That this growth accelerates with rise of temperature, and retards itself as it proceeds, seems clear; but that it ceases completely, so that the agencies resisting coalescence come in time to arrest it completely, at relatively high temperatures, has not yet been shown, I believe. A marked retardation as the process proceeds would naturally occur on almost any theory of grain growth, but that need not imply arrest. If it is true as reported that meteorites often consist of a single enormous grain, that would indicate that, at least in this case, if time is almost unlimited coalescence becomes almost unlimited, and that here the anti-coalescence agencies never become strong enough to cause complete arrest.

17. *Certain Laws of Annealing for Alpha Brass.*—The Authors' data point to the following laws:

(*A*) As the degree of previous deformation (cold-rolling) increases from a reduction in thickness of 4 per cent. to one of 40 or 50 per cent.,[6] (1) the temperature at which in subsequent heating reorientation becomes visible under a magnification of 85 diameters falls progressively from 650° to between 275° and 300° (Table III); (2) the temperature of most

[6] Both the 40 per cent. maximum reduction of Table III and the 50 per cent. reduction of Table IV and Plate XI are correct.

rapid softening falls progressively from between 550° and 650° to below 350° (Table IV); and (3) the range of temperature in which the greater part of the softening occurs becomes progressively narrower (Table IV).[7]

(*B*) From this it follows that the annealing temperature needed to cause nearly complete softening is the lower the greater the previous reduction has been, and the longer the annealing period. For a 30-min. annealing the relation is about as follows:

TABLE *A*

| | | | | | | |
|---|---|---|---|---|---|---|
| 1. Percentage of reduction by cold-rolling..... | 4 | 8 | 12 | 15 | 25 | 50 |
| 2. Temperature needed for nearly complete softening, degrees Centigrade.............. | 750 | 750 | 650 | 550 | 550 | 450 |
| 3. Percentage of total softening at that temperature.............................. | 90.4 | 98.8 | 90.0 | 90.6 | 79.6 | 79.2 |
| 4. Lowest temperature of visible disintegration, degrees Centigrade...................... | 650 | 550 | 550 | 450 | 350 | 350 |

(*C*) The temperature-range of most rapid softening merges into a higher range of progressively retarded further softening, which continues in every case even to beyond 750°, suggesting that the softening is due to at least two superposed changes, of which one, probably recrystallization, occurs rapidly at a relatively low temperature—the lower and narrower the greater has been the cold-rolling—and the other, probably grain growth, is spread out.

(*D*) *Decrease on Heating of the Excess of Hardness Caused by Greater Deformation.*—Here we may leave out of account the 2 per cent. reduction, because its effects are so slight compared with the accuracy of the Shore method, and the 50 per cent. reduction because this refers to a different alloy.

The excess of hardness caused by 25 per cent. reduction over that caused by 4 per cent., which is 14.7 Shore units before annealing, falls to 10.9 units on annealing at 350° and to 1.7 units on annealing at 450°. We may question whether this excess remains measurably great after annealing at 550° or any higher temperature, for though, if we consider the extreme cases, the 4 and the 25 per cent. reductions, a slight excess persists even on heating to 800°, as shown in line 4 of Table *B*, yet for

[7] This agrees with Muir's observation that the greater the hardening caused by overstrain, as shown by the rise of the yield point in steel, the lower is the temperature at which softening sets in (*Philosophical Transactions of the Royal Society of London*, vol. 193 A, p. 22, 1900), and with Chappell's that the greater the deformation the lower the temperature at which the recrystallization (disintegration and subsequent coarsening) of overstrained steel occurs (*Journal of the Iron and Steel Institute*, vol. 89, pp. 471, 496, 1914).

TABLE B

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| 1. Annealing temperature, degrees C.... | Unannealed | 350 | 450 | 550 | 650 | 750 | 800 |
| 2. Hardness after 25 per cent. reduction.. | 27.4 | 23.6 | 13.7 | 11.8 | 10.2 | 8.2 | 7.8 |
| 3. Hardness after 4 per cent. reduction... | 12.7 | 12.7 | 12.0 | 10.9 | 9.3 | 8.0 | 7.5 |
| 4. Difference, Shore units[8]............ | 14.7 | 10.9 | 1.7 | 0.9 | 0.9 | 0.2 | 0.3 |

temperatures above 450° this excess hardly exceeds the limits of error. Moreover, if we compare all five of the reductions, we find great anomalies. For instance, the specimens reduced by 4 per cent. are actually reported as harder than those reduced by either 8 or 12 per cent., after the 650°, 750°, and 800° annealings severally.

Again, for each of the four annealing temperatures, 550°, 650°, 750°, and 800°, there are five reductions that we are considering, those of 4, 8, 12, 15 and 25 per cent. If for each temperature we compare the hardness after any one reduction with that after the next greater reduction, we have four such comparisons at each temperature, or 16 for the four temperatures. Though in eight of these 16 cases the hardness given by the greater reduction exceeds that given by the less, yet in seven cases the reverse is true, and in the remaining case the greater and the less reduction give the same hardness.

(*E*) *The Temperature at which Disintegration First Becomes Visible may Suffice to Efface the Excess of Hardness Given by Greater Reduction.*—This refers to the temperature at which the reorientation becomes visible under a magnification of 85 diameters. For instance, this temperature is 550° for the reductions of 8 and of 12 per cent., and at this temperature the residual hardness is actually greater after the 8 than after the 12 per cent. reduction, though the difference hardly exceeds the limits of observational error.

(*F*) As the temperature rises, the softening accelerates greatly and continuously up to a maximum rate. For instance, a softening amounting to 3 Shore units is made 6.4 times as rapid by a 50° rise, from 225° to 275°, and about 1,000 times as rapid by a 100° rise from 225° to 325° (Table II, p. 7).

[8] By a "Shore unit" is meant a difference of 1 in the unit place of the Shore scale, hardness 14.3 being 2 units greater than 12.3.

# Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. HOWLAND, printed in *Bulletin* No. 111, March, 1916, pp. 627 to 650.

W. H. BLAUVELT, Syracuse, N. Y. (communication to the Secretary*).—I have read Mr. Howland's paper on Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces with much interest, and believe that his method of analysis of the subject will be of material assistance in our study of the action of coke in the blast furnace.

Mr. Howland quotes Professor Richards' statement that from the standpoint of the generation of the maximum quantity of heat in the furnace, Grüner was right in saying that all the carbon burned in the furnace should be first oxidized at the tuyères to CO and all reduction of oxides above the tuyères should be caused by CO, which thus becomes $CO_2$. It is clear that the maximum quantity of heat is generated in the furnace when the greatest amount of carbon is burned to $CO_2$ irrespective of whether it is burned at the tuyères or further up the furnace. So it would appear that Grüner's theory is sound regarding the maximum production of heat only because if all the carbon is burned to CO at the tuyères the reduction of the ores will have to be effected by the further oxidation to $CO_2$. Whereas, if the reduction of the ores is effected in part by solid carbon, part of this carbon might be oxidized only to CO, resulting in a higher percentage of CO in the escaping gases than from Grüner's "ideal conditions." But in studying the combustion of coke in the furnace, it is clear that the production of the maximum quantity of heat is not of the first importance in blast-furnace operation, or in the utilization of the fuel charged into the furnace. To my mind the production of a high thermal head at the tuyères is of the first importance, and the best coke is that which reaches the tuyères in proper condition to produce the highest temperature at the tuyères, and in just sufficient quantity to do the amount of work required there under the conditions produced by this maximum temperature. The combustion of a much larger amount of fuel at the tuyères, under conditions that will fall short of producing the highest possible temperature, cannot produce as good results, either in fuel economy or output. If this statement is correct, does it not correspond with Mr. Howland's statement that too much wind is inimical to low coke consumption? Nothing is more fatal to obtaining the highest temperatures than an excess of air for combustion. In the blast furnace an excess of air dilutes and cools the products of combus-

* Received June 30, 1916.

tion, reducing the maximum thermal head at the tuyères, and the larger volume carries the high-temperature zone too high in the furnace.

There has been a great deal of discussion as to what are the qualities which make the best furnace coke. The wider use of the byproduct oven has brought the control of the coke closer to the furnace man, and makes it possible to modify its structure in many ways, so that the question "what are the best specifications?" has become a very live one. It will probably be generally admitted that furnace coke should be of nearly uniform size, and many furnace managers are eliminating all coke below ¾ in. and above 4 or 4½ in. Also that the best coke is that which is sufficiently strong to resist undue abrasion and crumbling by attrition with the stock, and of an open porous structure that will permit the most rapid combustion when it reaches the tuyères. Many large users agree that the coke should never be overcoked, beyond the point of producing a sufficiently strong structure, as overcoking quickly reduces the combustibility.

If Grüner's ideal gives the best furnace operation, we should want a coke that is resistant to the oxygen in the ore but easily combustible at the tuyères, which is a contradiction of qualities. If my argument is correct, that the furnace man wants the greatest thermal head at the tuyères rather than the production of the greatest quantity of heat in the furnace as a whole, then he is willing to sacrifice some coke by solution in the oxidizing gases in the upper part of the furnace, provided he can obtain a sufficient quantity of coke at the tuyères, of a quality that will permit rapid combustion with the minimum amount of air, thereby giving him the maximum thermal head.

## The Rifling of Diamond-Drill Cores

Discussion of the paper of Walter R. Crane, printed in *Bulletin* No. 113, May, 1916, pp. 823 to 833.

H. M. Roberts, Minneapolis, Minn. (communication to the Secretary*).
—The rifling of drill cores is a frequent source of interest to men engaged in diamond drilling. Previous to the appearance of Dr. Crane's paper, there has been little record of connected observation or rigid speculation as applied to the cause of rifling. Replying to one of Dr. Crane's letters of inquiry, I ventured my opinion that the cause of rifling was a complex problem in physics, which I had not attempted to solve up to that time, except to ascribe it to the general cause of rotation and vibration of the rods. The question is worth attention not only as a matter of curious scientific interest but also for the reason that it stimulates thought on

* Received June 24, 1916.

the mechanical action which takes place at the end of a diamond bit. In discussing Dr. Crane's paper, I still recognize the complexity of the problem and merely note a few observations of fact with the inferences which may be drawn from them. I am indebted for suggestions from W. J. Mead, F. F. Fredlund and James W. Hunter among others on the staff of the E. J. Longyear Co.

It seems clear that the cutting medium which causes uniform threading must be the diamond bit itself. The deep, regular rifling of such dense rocks as granite and norite for long intervals admits of no other reasonable assumption.

Rifling occurs in many different rocks, both hard and soft, but in all instances noted there is one feature in common: the rock is homogeneous over the extent of the rifling.

### *Wall of Drill Hole Rifled*

The wall of the drill hole itself is rifled, as well as the core. This has been observed in the shaft of the Isabella Mine in Northern Michigan, which was sunk on a 2-in. hole in diabase.

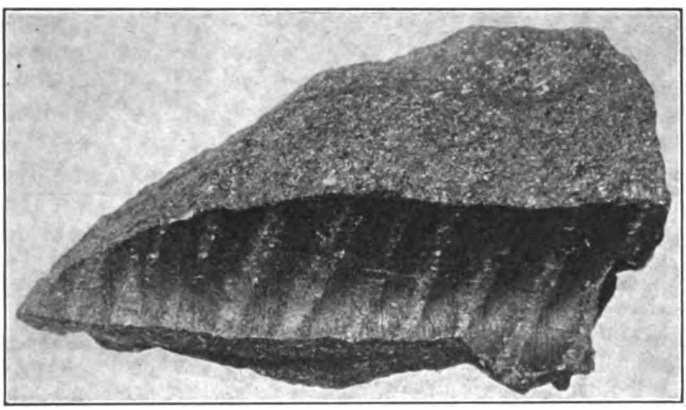

Fig. 1.—Specimen from Shaft of Isabella Mine, Cascade Range, Mich., Showing Rifling on the Wall of an "N" Drill Hole.

Fig. 1 shows a specimen taken from the walls of this hole. It is reasonable to suppose, therefore, that rifling of the core is also accompanied in most instances by rifling of the wall of the hole. This leads to the inference that perhaps the immediate end of the bit is responsible.

### *The Diamond Bit*

A consideration of the great pressures which bear upon a bit during the drilling makes it quite certain that all the stones work and engage the rock together. Examination of a worn bit shows that all the stones

have played their part. In this connection it is of interest to note that while diamond setters place outside stones very accurately to gage, they seldom use a gage in setting inside stones, which perhaps admits of one stone working alone on the inside for some interval of time. Those portions of the stones which project from the immediate interior and exterior edges of the face of the bit are no doubt responsible for the threading. It is apparent that any play in the bit would act with greatest effect at the extreme edges and would permit of greater penetration than the overset of that portion of the stone which extends up from the edge on the inside and outside of the bit. During the drilling operation the protuberance of the stones at the edges is much greater than when first set up, owing to the wearing away of the metal. Fig. 2 shows two worn diamond bits which have produced rifled core. There is no evidence to

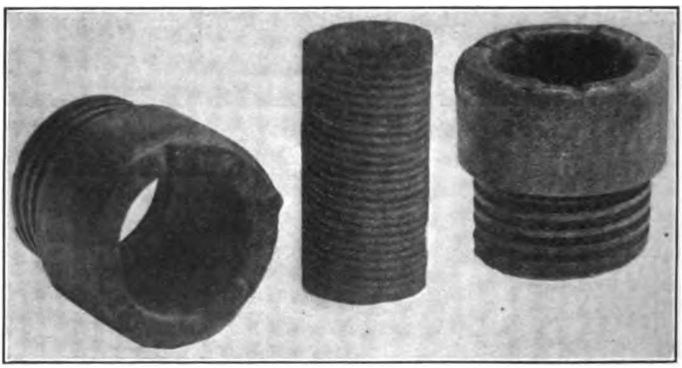

Fig. 2.—Worn Diamond Bits Which Have Produced Rifled Core and a Specimen of Quartzite Core Showing Left-Hand Threads. This Core Was Produced by a Bit Revolving to the Right.

show that a bit which has produced rifling was different in any material respect when first introduced to the hole than a bit which has produced smooth core. There is decided evidence to the effect that carbon wear and loss of metal are increased in any bit which has produced rifling. Thus the cause of rifling must lie in some force which acted upon the bit during the time when the rifled core was produced and which did not act during the time when the smooth core was produced.

### *Different Kinds of Threads and Varying Conditions*

The depth of the hole does not seem to be a governing factor. Deeply rifled cores have been found in norite drilled with a 2-in. bit 10 ft. from the machine. Threads of different size, different pitch, and of varying numbers to the turn are to be found on cores of the same material from one hole drilled with the same bit and with the machine making the same

number of revolutions per inch of advance in the bit. The threads are both left hand and right hand, although as far as my observation goes right-hand threads predominate. Fig. 1, showing the rifled wall of a drill hole, reveals both left-hand and right-hand threads in the same specimen. The bit producing these threads revolved to the right. I have counted both even and odd numbers of threads, varying from three to twenty, after which they become too fine to count; in fact, a smooth core always shows small variations from a true cylindrical shape.

### *Vibration of Rods Prevalent When Rifled Core is Produced*

It is an observed fact that vibration of the rods is prevalent in drilling formations where regular rifling occurs. This obvious relation led to the replies which drill operators gave to Dr. Crane's question. The production of threads on a vibrating pipe in a lathe bears a distinct analogy and renders the relation between the vibration of the drill rods and rifling of the core quite certain. Let us examine this relation further, since we are pursuing the inquiry largely as a matter of pure interest.

### *Harmonic Motion of the Bit*

The presence of threads over any interval of core indicates a constant play to and fro of the bit relative to the axis of the hole during its advance. Uniformity of the threading over any interval of core indicates that the swing of the bit operates according to some law. The fact that the number of threads and their character change from interval to interval, indicates that the factors causing the swing of the bit vary in considerable degree. We are evidently dealing with some type of harmonic motion. Possibly the homogeneous character of the rock found in most rifled cores is one of the principal factors which permits of harmonic motion in the bit. In considering the instance where many threads of low pitch appear on a cross-section of core, it is possible at first sight to conceive that the threads are due to spiraling which is directly proportional to the advance of the bit, but in an instance where corrugations of steep pitch are produced, as No. 8, Fig. 1, of Dr. Crane's paper, with perhaps 600 revolutions of the bit to an advance of 1 in., it is evident that the process is extremely complicated. The production of left-hand core by a bit revolving to the right is also a difficult matter to explain (see Fig. 2).

Consider the forces that operate on a flexible line of drill rods which are revolving rapidly. First, compression; second, torque. The play of these two forces sets up waves which have their lengths parallel but twisted with respect to the drill rods and with their crests and troughs at right angles to the axis of the hole. The twist in the rods is perhaps

not as great as might be supposed, for when a bit is blocked under great pressure, and the engine is stopped and released, the chuck seldom revolves back more than a turn or a turn and a half. During this process the bit is presumably fast on bottom.

As the rods revolve, transverse waves advance down the rods at every azimuth, like waves of light. That waves of this type are produced is shown by the whipping out of uncased holes when drilling soft iron formation. The samples are often vitiated in this manner. Anyone who will climb the tripod and look down on the water-swivel when the rods are vibrating will see there a distorted but nevertheless fairly true picture of what is taking place at the bit on the other end of the rods. It is possible to count distinct beats in the play of the swivel to and fro as the rods vibrate.

The regularity of the grooving in the threaded core indicates that the waves producing it have a definite time interval with a fixed relation to the revolution of the rod. The fact that all the stones are working at the same time does not present any particular difficulty when it is remembered that in any one instance of threading the stones are always at fixed intervals with respect to the axis of the bit and therefore have a fixed relation to any wave that affects the bit and a fixed relation to the period of revolution. In the instance cited by Dr. Crane where pentagonal core is formed, the time of vibration must be nearly a definite divisor of the time of revolution. That is, during one revolution of the bit there were approximately five major wave motions which produced a portion of five threads on the core for a longitudinal distance of less than $\frac{1}{100}$ in. In all other instances where the pitch is flatter the ratio must be more intricate. When right-handed threads are produced, the period of vibration is incommensurable and in this instance thereby deferred slightly each time with respect to the core as the rod revolves. When left-handed threads are produced, the period of vibration is also incommensurable, but in this case occurs just an instant earlier with respect to the core during each succeeding revolution.

### *Overtones*

In wave motions of this kind it is conceivable that there are overtones; that is, if the period of one wave length be $T$, then there are wave lengths of $\frac{1}{2}T$, $\frac{1}{3}T$, etc. The braided threads on many pieces of core are suggestive of secondary action of this kind (see Fig. 2 of Dr. Crane's paper). All of the laws of resonance and interference in wave motion would enter into the amount of play in the bit as it revolves. Again, when relatively smooth core is produced, it is probably the result of the impact of countless numbers of periodic waves offsetting each other.

There are many other apparently simple phenomena in nature which

are due to a combination of periodicities, as a beam of light or the general strike of a complexly folded iron formation.

The net result of the wave motions down the rods operating under the laws of resonance is translated into a wave whose length is nearly parallel to the circumference of the core and whose amplitude is along the radius, thus producing uniform threading. This is recorded in the rock like the mark on the indicator card of an engine. Only in rare instances do all the factors combine under the laws of resonance so that a deep definite tracing of the wave motion is formed.

### *Mathematical Interpretation*

We may attempt to arrive at a rough mathematical interpretation of wave motion at the end of a diamond bit as expressed by threads in the core.

Let $R$ = the number of revolutions of the bit during any interval of advance.

$N$ = the number of threads appearing on the cross-section of core

$F\ (p)$ = some continuous or discontinuous function of the pitch of the threads, $p$ having values from negative to positive infinity, $F\ (p)$ approaching the values $-1$ and $+1$ at these ends, and approaching the value zero as $p$ approaches zero, the value of $p$ being related to the value of $N$.

Let $W$ = the number of major wave motions during any interval of advance of the bit, as indicated by the threads.

Then $W = R\ N\ F\ (p)$

Thus in the instance where pentagonal core is formed, and $N$ equals 5, $p$ approaches infinity and $F\ (p)$ approaches the value 1, if it be assumed that 600 revolutions of the bit have been made during the delivery of 1 in. of core; then $W = 5 \times 600$ or 3,000, which is indicative of the number of major vibrations of the bit with respect to the axis of the core during its spiral advance of 1 in.

When the pitch $p$ is left-handed, $F\ (p)$ may be considered as varying from 0 to $-1$; when the pitch is right-handed $F\ (p)$ varies from 0 to $+1$. As $p$ becomes small in either case the value of $N$ increases and $W$ increases approaching the case where many fine threads of low pitch are produced; $W$ increases greatly, which is to say that a cylindrical core is formed when there are a large number of exceedingly small wave motions of the bit to and fro. Considering an instance when $N$ remains constant and the threads ultimately coalesce while "$p$" decreases in value, then the value of $W$ decreases. The ultimate end of this process is the production of a smooth core with slight wave motion of the bit.

Inspection of this equation shows that as $R$ increases, $W$ must increase.

This expresses the violence of vibration when harmonic motion of the resonant type is set up during a high speed of revolution.

This mathematical statement is merely an attempt to express some of the relationships which appear, by speculating on the nature of the phenomenon. It is doubtful whether it is possible to determine rigidly the value of $F(p)$ in any instance or to tell why $p$ should be left-handed or right-handed or to devise any means of telling why $N$ is 5 in one case or 9 in another. The strength of the rods, the number and character of the carbon, the way they are set with respect to each other, the water pressure in the hole as determined by the force of the pump, the clearance of the bit, etc., the amount and character of the deformation in the metal of the bit at any particular time, the kind of drilling machine, and the manner of its set up, the depth drilled, the nature of the rock, the presence of soft horizons above in the hole which may affect the vibration of the rods, the speed of rotation and the pressure on the rods—the factors which influence these values are numerous and therefore difficult to analyze into their true proportions.

*Practical Considerations*

Dr. Crane's practical application of the results of his research is discouraging in these days of keen competition for drilling contracts. I refer to the suggestion that it is possible to reduce "the vibratory action of the drilling mechanism by reducing the speed of rotation of the rods." An old drillman in commenting on this says that he can break up the vibration by increasing the speed of rotation. This suggestion might better be stated thus: The violence of vibration can be broken up by varying the speed of rotation.

This destroys one of the constant conditions essential to harmonic motion. The greasing of the rods dampens the wave and sets up interference by releasing friction at antinodes of the waves, thus destroying another constant. This may be inferred by examining the line of greased rods when pulled from the hole and by observing that the grease is worn clean at definite intervals.

In making this comment on Dr. Crane's conclusion I would not be understood as casting doubt on the utility of this type of research. An analysis of all the conditions which govern the advance of a diamond bit might enable operators to control the direction of diamond-drill holes. At present the hole usually takes its own direction.

Some practical considerations have occurred to me while dealing with the subject. They are no doubt commonplace to many old drillmen, but seem worth recording: When drilling deep holes in homogeneous formations, it will reduce the carbon wear and increase the footage to use stiff new rods and to use a blank bit which has the smallest possible excess

in diameter over the size of the rods. The use of eight stones in a bit rather than four or six will reduce the possibility of one stone carrying the burden of the work. The bit should be reset frequently so that deformation of the weakened steel in long-used bits will be avoided; thus the gage of the hole may be kept more accurately. These precautions will set up conditions adverse to the development of resonant wave motion to and fro in a bit and thus tend to keep it working straight ahead, which is the true business of a diamond bit.

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## The Emerald Deposits of Muzo, Colombia

Discussion of the paper of JOSEPH E. POGUE, printed in *Bulletin* No. 113, May, 1916, pp. 799 to 822.

EDGAR T. WHERRY,* Washington, D. C. (communication to the Secretary†).—Dr. Pogue's presentation of the facts concerning the emerald deposits is very clear and convincing, and the only addition that I can suggest is a summary of previous theories of origin. He makes it evident that the pegmatite theory is the only one capable of explaining the existing relations, but upon certain details there may be some difference of opinion. If I understand the term pneumatolytic, it does not imply that all the elements concerned in a given deposit were transported as gases, but rather that the crystallization of these elements into the various minerals was favored by the presence of certain gaseous substances, notably $H_2O$, $CO_2$ and HF. It is highly improbable that the oxides of glucinum, aluminum, chromium, and silicon, which enter into the composition of the mineral emerald could have been transported in the gaseous form. The same is true of the metallic constituents of the parisite and other associated minerals. The explanation suggested, that solutions separated into liquid and gaseous portions, the latter ascending and forming the emerald in the upper portions of the rock only, therefore, seems to me untenable.

When two formations exist side by side and one, *A*, is mineralized while the other, *B*, is barren, the possible explanations may be classed as (1) chemical, and (2) physical.

1. Some chemical feature of *A* not found in *B* might have caused crystallization of certain minerals in the former, which did not appear in the latter. In the present instance both rocks appear to be so similar chemically that no such effect can be looked for. Pogue mentions carbon as a possible precipitating agent, but describes both formations as carbonaceous, so that the difference in the minerals of the two can not be thus explained.

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* Assistant Curator, Division of Mineralogy and Petrology, U. S. National Museum.

† Received June 28, 1916.

2. The physical condition of $A$ might have permitted or encouraged the passage of the solutions, while that of $B$ retarded or prevented it. In the present deposit some mineralization occurs in both formations, calcite veins, albite, and pyrite being found in both, whereas emerald, parisite, and a number of minerals of minor importance occur only in the upper, $A$. It seems to me that this difference may have been produced by a change in the composition of the solutions during the progress of mineralization; at first these brought in only the constituents of albite and pyrite and deposited them with calcite dissolved from the wall rock, in both formations; the openings in $B$ became completely filled, while the more numerous or larger ones which would naturally have developed in $A$, since it was the uppermost formation, remained partially open. Then, when during later phases of the mineralizing activity the constituents of emerald and parisite appeared, they were deposited only in $A$ because $B$ had become impermeable.

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## History of the Flotation Process at Inspiration

Discussion of the paper of Rudolf Gahl, printed in *Bulletin* No. 117, September, 1916, pp. 1627 to 1681.

David Cole, El Paso, Texas (communication to the Secretary*).—I have read with great interest Dr. Gahl's painstaking paper giving us the details of development of flotation at Inspiration, and it seems to me that he has covered the ground so completely that there is little to discuss or criticize in connection with his subject. While it was in operation, the test mill in which the campaign was carried on was the Mecca of mill men and metallurgists. There was much to interest students of milling methods in addition to the flotation experiments, and readers of Dr. Gahl's paper will miss the information they hoped to get when the "pilot" mill results were finally compiled, and I hope that the following remarks outlining from a distance some of the things that happened there will bring forth the rest of the story of the performances recorded in these important experiments.

Prior to the use of flotation methods in concentration it had long been recognized that the "unavoidable" losses of sulphides were in the slime which is inevitably produced in the grinding operations required to free the minerals to be separated. Classifying the feed prior to its final stage of treatment had long been in style in milling. This assisted the sand-handling machines and resulted in lowering the tailings made by them, but at the expense of the slime-handling department; and while little was expected of the latter, the wisdom of complicating the process by hydraulic classification was being seriously questioned. Indeed, at the

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* Received July 8, 1916.

time that the gravity method flowsheets were being developed for Inspiration ores, there was a minor revolution in milling methods impending involving the elimination of the hydraulic classifier. This change gave no promise of higher extractions but did promise greater simplicity and much less floor space per unit of capacity. Such mills promised to be less costly to build, less expensive to operate, and equally efficient. Wholesale concentration in relatively small space would get as good results as piecemeal concentration had been getting in the multiple operations of the spreadout plants into which the porphyry treatment mills had degenerated.

The one great drawback was the slime. The desiderata of the engineer and the manufacturer was, on the one hand, to provide grinders which would produce the minimum of slime, and, on the other hand, improved "slimers" which would give the maximum recovery from the finest products. The technical press of the period reflected this state of things in the advertisements of those having the latest novelties to sell. There was much revamping of old ideas with some refinement, but nothing new. Tube-mill grinders were "taboo" for concentration, because they had a bad reputation as slimers. Automatic canvas plants were being exploited and very ingenious multiple-deck table devices were being offered as the remedy—the only possible remedy.

Looking backward no further than 1912, when Inspiration began to study its milling problems, we now see that we were without effective resource in treating real sulphide slimes. Mr. Callow's investigations and experiments had apparently demonstrated that some departure from the usual gravity practice would be advantageous and that a high recovery for that method would be possible on the granular material, but he threw small light on slime treatment. This was the situation when the Inspiration company was endeavoring to determine its mill treatment scheme.

After reviewing from every angle the results of experiments on the ore and other information available, a modified flowsheet was finally crystallized by Mr. Burch and adopted by the management. Mill plans were drawn for what was to be a most highly developed 7,000 tons per day gravity process plant, and work was immediately inaugurated to carry out these plans. The mill site was selected, and much active work had been prosecuted before flotation (by this time being hastily tried in the old experimental plant) had so far won its battle that results could be viewed otherwise than probably too good to be true.

The apparent promises of flotation were extremely attractive, because the process would be simple and would solve the all-important slime problem. The process would have a greatly reduced number of stages; the plant would be much smaller per unit of capacity; the cost of construction per unit of capacity would probably be very much less; the

use of water would be minimized; grinding would have to be carried further than usual, and would be the main item of milling cost, but this was not a very great handicap because sliming did not matter. What system of grinding would be best to use and what is the best machine? Would it be possible to parallel the small test-mill metallurgical results on a full-tonnage basis, and finally would it not be too hazardous to accept so revolutionary a process with so much experiment in its makeup?

The mine preparation program which had been settled upon would produce about 600 tons daily of freshly broken ore directly from the headings. This happened to be the rated capacity of a full-sized Minerals Separation unit which the flotation people were urging as a means of improving their extraction. If this ore from the headings were put in stock pile in the usual way, pending the completion of the milling plant, the ore would oxidize to some extent and besides would involve reclaiming expense later on. Why not mill it all as fast as produced, in a real "pilot" mill, wherein not only flotation problems, but grinding problems, power consumption, use of water, preparation of sticky concentrates, and other vital questions which might come up could be definitely threshed out under what would be regular commercial conditions on full-sized machines? In this size of "pilot" operations, labor would be used economically and production would almost, if not quite, pay all of the current expenses, except the mining cost.

This program was adopted. That the decision to carry it out was a wise one is shown in Dr. Gahl's paper, and that it paid its way is shown by Mr. Mills' annual report for 1915, in which he says: "Contrary to the usual experience, this test mill paid the cost of its construction, its operation expense, the present average mining cost on ores treated, and something besides, and has been written off the books."

Anybody with a real idea applicable to the problem could get a hearing and a tryout in this extremely practical commercial sized laboratory where sampling was in the hands of engineers and the results were compiled in a way to make them comparative and valuable. Thousands of dollars were spent by the company, by inventors, and manufacturers in demonstrations. Expense was subordinated. Heavy shipments by express were made when necessary to hasten the work, and much more than flotation was developed in this "pilot" plant.

Four different types of Symons crushers and pulverizers were tried. Three of these were marked and very interesting departures from ordinary practice. The company had purchased the Hardinge mill patent rights for Arizona and four forms of this mill were installed to determine the best to use, and these were kept busy during nearly the whole campaign. A long parallel tube mill was installed and run in competition with the Hardinge mill. A high-speed Huntington-type grinder at one time attracted much interest. Hammer pulverizers of two different makes

were also tested. Various linings and grades of flints were tried in the pebble mills. Steel balls in place of pebbles were advocated and a carload purchased.

In the latter part of the testing period the Marcy type of ball mill, especially designed for using iron balls larger than usual, and adapted to crush from breaker sizes to 48-mesh in closed circuit in one operation, was installed and perfected. This grinder proved capable of a greater range of reduction than had previously been thought possible, from an initial feed as coarse as 3-in. cubes. It is a ball mill pure and simple, having large capacity in small space. It makes use of a perforated diaphragm to keep the balls and charge inside of the mill until the latter will pass a 3/16-in. opening, and it has the equivalent of a peripheral discharge. An overflow classifier determines the finished size and the oversize is continually returned to the grinding chamber. This mill uses little water in the grinding chamber, so that its charge of ore is mortar like in consistency. It was quite successful. There was nothing in the Hardinge equipment to parallel it because the Hardinge mills were built for pebble mills and did not have feed scoops or openings adapted to handle as coarse a feed, and the linings would not stand up under the ball load. Would the Hardinge machines when built as a ball mill with the required strength, type of lining, size of feed and discharge openings, do as well? To wait for a mill to be made over or a new one manufactured would take too long, so the Marcy type was adopted and, contrary to what I think is the popular impression, the conical-type steel-ball mill *vs.* the Marcy-type steel-ball mill did not receive a tryout.

Recording electric instruments were installed so that accurate power records could be continually made while the various machines were being operated.

Several varieties of drag and rake classifiers and two types of vacuum filters were installed and records made. The efficacy of high-reduction herringbone gears driving ball and tube mills became a matter of interest on account of the troubles that developed in them, and the reasons for these troubles, which would make a story by itself when discovered.

This testing work grew into a process of elimination and the scrap pile grew steadily. Some blasted hopes may be buried there, but it does not follow that all of the machines or materials that were returned to the sponser or that found their way to the scrap heap were entirely unfit. It was necessary to choose and to discard, and that there is no acrimony in connection with the matter speaks well for the type of justice and judgment that prevailed. Doubtless the use of some discarded things might have answered as well, but it would be hazardous indeed to say that anything vital on the score of cost or recovery of values failed to receive recognition in the final selection. One of my mental offspring was among the fallen. It held out for a long time and I greatly appre-

ciate the favorable mention which Dr. Gahl has made of it. I would have been pleased to pursue its development further, but I can not at this time see where its adoption would have saved any more copper or any more money in getting the copper than the ones that were selected, and I think this will apply to all the "late lamented" concerned in the campaign that was carried on. In other places and on other ores the conditions would not be the same. The ratio of concentration, or the crushability factors particularly, may so change conditions as to indicate the use of ideas that were discarded at the Inspiration test mill.

When the comprehensive plan of testing on a large scale was decided upon, I think Dr. Ricketts and Mr. Mills recognized that the records made might be extremely valuable to engineers and metallurgists, and I believe the matter of writing the story of the campaign was considered and that records were started with that ultimate end in view. Experience added to theory is more valuable than theory alone; the scrap heap more eloquent than the machine promotor, and the wornout unit in the "bone yard" is often more interesting than the one shown on the original blue print. It is seldom indeed that experiments are possible under such auspices and on such an extensive scale as those carried out in the Inspiration "pilot" mill, and I hope that Mr. Burch or Dr. Gahl, or both of them, will, with the approval of the Inspiration company, find time and inclination at an early date to prepare a paper or papers which will extend, through the *Transactions*, the value of these experiences to our members.

Referring to the final flowsheet adopted, I note that hydraulic classification had no place in the 600-ton "pilot" mill experiments which Dr. Gahl has so interestingly described, and I note that he has not referred to the reason for retaining this remnant of the old system of concentration in the flow sheet.

I note that 3 tons of water per ton of ore handled is required in the flotation operation and that 3 tons more is added in the subsequent table treatment which, of course, includes the hydraulic classification operation practiced, and I presume that something more than one-half of the last 3 tons is added in the classifiers themselves; and since the water is clarified and returned, the addition of unnecessary water would entail expense for clarification and pumping which would not be required if less water were used.

In my work in the concentration of ores I have not been able to become very enthusiastic over hydraulic classifiers, and since flotation has come to the rescue of the slimed sulphide I find myself less enthusiastic than ever over their use.

When I was a lad in the Black Hills I was fascinated in watching a certain mountain spring in which polished micaceous particles glistening in the sunlight would be caught in the current rising from an orifice in the bottom of the sand funnel and be flirted to the surface, sail across the

crater and fall upon the conical sides to be again methodically returned over the same route. It was interesting to watch the disturbance caused by dropping a handful of foreign sand and silt into the funnel and have it "classified" and washed clean, a new form of crater finally being established with the changes of average sizes retained. My first contact with hydraulic sizing in concentration was studied from that foundation. The spring took its time to do a good job; it worked the charge over repeatedly. All the silt went out quickly and the fines gradually went overboard with a rapidly decreasing ratio, until the crater would settle down again to its regular work of turning the mobile contents over and over in a new condition of equilibrium. But I soon learned that the beauty was all taken out of the process in its commercial application.

The process witnessed in the action of the spring was balanced, precise, and definite, and quite at variance with what we witness in watching through glass the operations going on in a "teeter chamber" of the metallurgical hydraulic classifier, which seems to me to be of little value except in its office of washing out the slimes which used to be the main source of the loss in treating unclassified material upon concentrating tables.

Following the thorough combing out of the slimed values as effected by the splendid flotation treatment that the pulp has previously received in the Inspiration final flowsheet, I am inclined to question the value of the subsequent classification by hydraulic means. I note that the flotation tailing is split into slime and sand at the drag belts, that there is very little slime left in the sand portion, and that what little there is (on account of the previous frothing) is completely devoid of value which the tables can save, as indicated by the fact that the main slime overflow of the drag-belt separators is discharged to waste without further treatment. It seems to me that it ought to be possible to eliminate the classifiers, and I would like to ask Dr. Gahl if there has been any trial to determine what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation.

## Comparative Friction Test of Two Types of Coal-Mine Cars

Discussion of the paper of P. B. Liebermann, printed in *Bulletin* No. 114, June, 1916, pp. 1057 to 1065.

Edwin M. Chance, Wilkes-Barre, Pa. (communication to the Secretary*).—I have read with great interest Mr. Liebermann's paper reporting dynamometer tests of mine cars and wish to express my appreciation of this investigation. The coal-mining industry needs just such precise study of the problems with which it is confronted.

* Received June 24, 1916.

It would seem, however, that the writer has demonstrated that a high-grade grease is superior to black oil as a mine-car lubricant rather than that roller-bearings are superior to plain. While it is reasonable to believe that roller bearings will cause less friction than plain when used on the trucks of mine cars, still this point can not be conceded on the strength of the tests quoted, for in these tests grease was the lubricant of the roller bearings, while the usual black oil, of low lubricating value, was the lubricant of the plain bearings. Under these conditions, a case is made for the lubricant rather than for the bearings.

Some time ago certain of the coal mining companies with which I am associated, called upon me to have a high-grade grease prepared for them for use in plain mine-car bearings. This problem was successfully solved and this lubricant has now been in use for some years. The fact has been established that the ordinary plain mine-car journals are especially susceptible to such a lubricant and that its use greatly decreases the drawbar pull of these cars. The fact must be borne in mind, however, that the exigencies of this service require a special grease and that the use of the ordinary grades of cup grease will surely spell disaster.

It would be indeed interesting to have tests of the same degree of precision as those reported made upon cars with plain bearings lubricated with black oil and special mine-car grease and to compare the latter with tests under like conditions made upon cars equipped with roller bearings.

Charles Legrand, Douglas, Ariz. (communication to the Secretary*).—Mr. Liebermann's paper on friction tests of two types of coal-mine cars is very interesting. It is a pity that the dynamometer car was not designed for use on tracks of 18 to 24 in., as this is the most usual gage in metal mines in this part of the country.

I have tried to make tests by means of an ordinary spring dynamometer and find, as stated by Mr. Liebermann, that this method is very unsatisfactory, as the conditions of mine tracks are too variable to produce a steady pull on the dynamometer. It would be very valuable to the mining industry to have accurate tests taken on narrow-gage track, to determine not only bearing friction, but the total track resistance. Such tests would convince the mine managers that it pays to lay heavy tracks and keep them in good condition.

Personally, I have always been in favor of the roller bearing, not only on account of its lower friction but also on account of its smaller maintenance and lubricating costs. To be successful, however, this type of bearing has to be dustproof. On a narrow-gage car this is usually done by inclosing the whole axle in a casting which also supports the two bearings. This casting acts as an oil reservoir and permits the car to be run a whole month with one oiling.

* Received July 5, 1916.

I hope that Mr. Liebermann will continue his work and make tests of the narrow-gage cars.

JOHN PHILLIPS, Pittsburgh, Pa. (communication to the Secretary*).—I am familiar with the test made by the Greensburg Coal Co. and described in Mr. Liebermann's interesting paper, although I was not personally present. Mr. Liebermann's results on the Hockensmith plain-bearing wheels compare closely with tests which have been made elsewhere and show a considerable saving in favor of the roller bearings. However, as the grade increases, the ratio in favor of the roller bearings decreases until, on a 4 per cent. grade, their efficiency is only 11 per cent. greater than that of the plain bearings, as shown by the following official figures of the Hyatt company:

*Test, Oct.* 3, 1915

| | Pounds |
|---|---|
| Weight of car | 2,650 |
| Weight of coal | 4,500 |
| Total | 7,150 |

| | |
|---|---|
| Weight of 20 cars loaded, 2.4 per cent. grade average.. | 71.5 tons. |
| Correction | 48 lb. per ton. |

$$71.5 \times 48 = 3{,}432 \text{ lb.}$$

| | |
|---|---|
| 4,280 | 5,190 |
| 4,350 | 5,070 |
| 4,410 | 5,250 |
| 3)13,040 | 3)15,510 |
| 4,347 Hyatt | 5,170 plain |
| 4,347 | 5,170 |
| 3,432 | 3,432 |
| 915 | 1,738 |

Frictional resistance for Hyatt, $\frac{915}{71.5} = 12.8$ lb. per ton.

Frictional resistance for plain bearings, $\frac{1{,}738}{71.5} = 24.3$ lb. per ton.

| Drawbar, Pounds per Ton | | Grade, Per Cent. | Per Cent. Saving | Percentage Cars Possible to Add in Case of Roller Bearings |
|---|---|---|---|---|
| Roller Bearings | Plain Bearings | | | |
| 12.8 | 24.3 | level | 47⅓ | 90.0 |
| 32.8 | 44.3 | 1 | 26.0 | 35.0 |
| 52.8 | 64.3 | 2 | 17.9 | 21.8 |
| 72.8 | 84.3 | 3 | 13.6 | 15.8 |
| 92.8 | 124.3 | 4 | 11.0 | 12.4 |

It will be noticed that all the readings were taken on a straight track on an average grade of 2.4 per cent. Had any of the readings been taken on a curve, we are confident that the drawbar pull on both the roller

* Received July 10, 1916.

bearings and the plain bearings would have been considerably higher when the train was rounding the curve, because the axles of the Hockensmith trucks are of straight cold-rolled steel without collars, and in rounding the curve the car naturally shifts against the inside wheel. The rear hubs of the wheels are necessarily large in order to admit the bearings, and the large hub grinding against the box creates considerable friction, which would increase the drawbar pull on a curve.

Records of three tests conducted by the Engineering Department of the Pittsburgh Coal Co., on three different types of wheels, are given below. At Essen No. 3 mine, the Jarvis Adams roller bearings were used; at the Champion mine, Eureka wheels and angle-bar trucks (the wheels being similar to the plain-bearing wheels used in the Greensburg test), and at the Dickson mine, our open-cap wheel trucks. Inasmuch as the drawbar pull on the Hockensmith plain-bearing wheels at Champion agrees very closely with the results on the Hockensmith plain bearings at Greensburg, the results may be taken as approximately correct, and show the following comparison:

| | |
|---|---|
| Hyatt roller bearings at Greensburg | 12.8 lb. per ton. |
| Jarvis Adams roller bearings at Essen No. 3 | 15.14 lb. per ton. |
| Open-cap trucks, Dickson mine | 18.48 lb. per ton. |
| Hockensmith plain bearings, Champion mine | 24.40 lb. per ton. |
| Hockensmith plain bearings, Greensburg | 24.30 lb. per ton. |

The figures on the Hyatt bearings were lower than we anticipated, as we had always supposed that the Jarvis roller bearings when new and in good condition were the easiest running on the road. Our open-cap truck costs about the same as the Hockensmith angle-bar truck, but runs about 25 per cent. easier. The cost of the roller-bearing truck is about $10 to $12 per car more than our truck, and on the basis of the Greensburg figures runs about 30 per cent. easier.

From the above data, I think that it is safe to say that it would be well for Mr. Liebermann, or some person equally well qualified to carry out this class of tests, to pursue it much further, both as to different kinds of wheels, and to the character of roads, which would include different grades and varied curvature, before a thorough comparison could be made and a conclusion arrived at that would be entirely satisfactory.

## History of the Flotation Process at Inspiration

Discussion of the paper of RUDOLF GAHL, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1627 to 1681.

RUDOLF GAHL, Miami, Ariz.—Since I wrote the paper on flotation which is in your hands, important developments have taken place, and, for this reason, I will try in a few words to bring it nearer up to date as far as the Inspiration plant is concerned.

You may have noticed that in the Inspiration concentrator, flotation is not solely relied upon for the recovery of the coarser material, but that flotation tailings are split into a sand and a slime product on drag classifiers. The sand product is treated on tables which thus supplement the work of the flotation machines.

Extensive tests have shown us that, if we wanted to, we could leave out these tables and substitute additional flotation machines, as they will make fully as good, if not a better recovery than tables on the deslimed feed, but the treatment would be more expensive, especially in view of the fact that oils are required which cost more than those which we are now using in our main flotation plant. We have, however, decided to apply flotation treatment to our table middlings.

I would also like to add a few words regarding the treatment of oxidized copper ores. Although our experience shows that the addition of hydrogen sulphide and other soluble sulphides effects a very good recovery of copper carbonates with certain ores, we have not been able to prove that we could use it advantageously for the ore mixture which we are treating in our concentrator, and for this reason, have looked toward leaching for extracting the carbonate and silicate copper that we are losing now. Experiments in this direction are going on and are giving very encouraging results. It may interest you to hear that we intend to use limestone for the precipitation of the copper which goes into solution, as electrolytic precipitation seems to be out of the question on account of the diluteness of the resulting solutions, and precipitation by iron was rightly objected to on account of the unavoidable contamination of the water supply. We feel very hopeful about the success of our limestone precipitation which, if it holds what it seems to promise, will develop into a novel feature of copper metallurgy.

FREDERICK LAIST, Anaconda, Mont. (written discussion).—Dr. Gahl is assuredly to be complimented on the preparation of so well-written and complete a paper as well as upon the excellent work done under his supervision. One cannot but be impressed by the care and thoroughness with which all possible combinations were investigated,

and the paper leaves one with the impression that the equipment finally chosen is without a doubt the best obtainable for the conditions prevailing.

I was specially interested in Dr. Gahl's description of the development of the Inspiration flotation machine, and its final perfection is certainly a credit to his keenness and powers of observation. The machine is simplicity itself and strikes me as the logical development of the pneumatic type.

The relative merits of the impeller and pneumatic types of flotation machines have been the subject of much discussion and the selection of type is doubtless dependent largely on local conditions and on the characteristics of the ore undergoing treatment. We have always been of the opinion at Anaconda that whenever a neutral or alkaline treatment was used and the oils could be added to the pulp going to the ball mills, the pneumatic type had an advantage both as regards power consumption and installation cost. When, however, the treatment requires the use of acid as is the case on some copper ores and most zinc ores, the pneumatic machine loses much of its advantage. Obviously, the acid cannot be introduced into the ball mill and it is generally necessary to add it ahead of, or at the same time as, the oil.

Thus it becomes necessary to insert an agitator between the ball mill and the flotation machine as the pneumatic treatment alone is not sufficiently vigorous. The early pneumatic machine installations contained Pachuca tanks for this purpose. These, however, did not prove effective, for the reason that an emulsification of the oil is required, and not merely agitation. It therefore becomes necessary to use an impeller or some form of mechanical emulsifier, and the power required to operate this machine must be added to the power consumed by the flotation machines proper.

In Montana we find that our power consumption for emulsifying and flotation is about 0.24 hp. per ton as compared with 0.15 hp. for flotation alone at Inspiration. The capacity of an impeller-type machine is materially greater when the emulsification of the oil in the pulp is done in the ball mill. In this connection an interesting suggestion was recently made by Dr. Cottrell. He suggests making an emulsion of oil and water in a special emulsifier, such as made by the De Laval people, consisting of two disks running almost in contact. The oil and water is fed in at the center and is thrown out at the circumference. Thus the work of spreading the oil through the pulp might be done more efficiently than is now the case.

For some time it seemed to us that the main point to be considered in choosing between the two types of machines was power, and that this depended largely upon whether acid or neutral or alkaline treatment were decided on. Of late, however, it has seemed to us that the impeller type of machine has another advantage, which, I recall quite distinctly,

was cited as a disadvantage when the first Callow machines were brought out. I refer to the toughness of the froth. Most of you doubtless recollect that one of the advantages of the pneumatic machines was supposed to be that the froth breaks down quite readily, thus rendering the mineral content of the froth easier of collection.

It is becoming more evident to us, however, that this apparent advantage is actually the reverse for the reason that the coarser mineral grains tend to fall back before they can be skimmed off and are thus lost or must be recovered by tabling. We are beginning to believe that the tougher froth is a distinct advantage of the impeller machine, and our belief has been considerably strengthened of late by tests made on a disseminated copper ore from South America. It was impossible to make as lean a tailing on the pneumatic machine as on the impeller machines, owing to falling back of the coarser mineral grains.

FRANCIS S. SCHIMERKA, Clifton, Ariz.—Regarding the proposed scheme of Dr. Gahl, to precipitate the copper from a sulphate solution by means of ground limestone, I had a discussion with him a few days ago. I called his attention to the fact that the result of this operation would be a low-grade, very slimy and voluminous precipitate consisting of the rather insoluble gypsum and basic copper carbonate both in highly hydrated form, which is difficult to handle and can be worked up only by matte smelting in the blast furnace.

Dr. Gahl has proposed limestone as a precipitant for the copper in the leaching solutions to avoid contamination of the water supply to the mill which would result from returning the exhausted liquors into the mill system. I think the difficulty could be overcome by applying the acid to a de-watered thickened pulp; this procedure would assist the leaching operation and the exhausted liquors could be run to waste.

RUDOLF GAHL.—I would like to say that I agree completely with everything that Mr. Laist says, except, of course, his compliments. It occurred to me through Mr. Laist's remarks regarding the relative character of the froth produced by the different types of flotation machines, it might be worth while trying to modify the froth of the pneumatic machines by reducing the air supply, and perhaps also by reducing the frothing agent in the flotation oil mixture. I feel sure that the character of the froth can be modified to some extent, although I doubt very much if without retreatment it ever would approximate the froth of the standard-type M. S. machine. Regarding Mr. Schimerka's remarks, I might say that I know very little about smelting, and have not considered the smelting problem very carefully. All I know is that Mr. Wallace, who used to be smelter superintendent of the International Smelting Co. here assured me that he could smelt that stuff, and could pay for the lime also. Regarding the other point Mr. Schimerka brought out, about add-

ing the acid to a thickened pulp, my impression is that still a large part of the water supply would be contaminated because we have to figure on leaching mainly very fine slime as it contains most of the oxidized copper; and it is the experience of every mill man that it is impossible to settle such slime to a consistency exceeding 3 to 1—that is, 3 parts of water to 1 part of solid matter. That would mean, if we added the acid to the thickened pulp, we would spoil 3 tons of water for each ton of leached ore.

C. E. MILLS, Miami, Ariz.—It seems to me that one of the members might tell the experiences at Chino in leaching oxidized ore.

H. W. MORSE, Los Angeles, Cal.—We have not gone far enough to say anything about what we are going to do, but I think at the next meeting of the Institute we will be able to give some very interesting results on this class of work. Dr. Ricketts has always turned down my proposals because they involved the handling of a thin pulp, settling and washing, and a great many other miseries which would certainly be involved in this class of work. But, when he saw the proposed scheme at Chino, he expressed himself, fortunately, as being satisfied that we had, provided we could make it go, decided on a scheme that met his approval, which involved no thinning of material or washing, but simply the ordinary passage of mill pulp through an agitator and flotation machine. We have not any results worthy of consideration as yet, but I feel encouraged by the laboratory work and very confident of the future of this scheme. If it will go, it will be an important process because it will enable us to treat mixed ores, provided that they contain a sufficient amount of soluble ores to precipitate on iron, and provided that they contain sufficient sulphides to pay for flotation operation.

DAVID COLE, El Paso, Texas.—I understand that Dr. Ricketts "turned down" Dr. Morse when advocating leaching processes for low-grade tailings by methods which involved first getting the copper into solution and then separating the solution from the pulp in a clarified condition, which is difficult and involves much washing with clear water, etc., resulting in thin solutions, the latter entailing difficulties or "miseries" in effecting precipitation and recovery of the metal.

On the other hand, Dr. Ricketts approves the present process which Dr. Morse is using at Chino, in which the oxidized copper is first taken into solution by the use of about 3 lb. of sulphuric acid per pound of copper digested; this copper is then precipitated upon iron in an agitated mass, which results in metallizing the copper at the expense of about 2 lb. of iron per pound of copper precipitated. The metallized copper is then in a very fine state of division in the pulp, and the remaining copper sulphides which were not attacked by the acid are also in the pulp in a very fine state of division, and the whole is subjected to flotation treatment for the removal of both the sulphide and metallized contents, thus doing

away with the necessity of removing the solution in a clarified condition from the pulp under treatment, avoiding the washing of the pulp for complete removal of solutions pregnant with copper, and avoiding most of the "miseries" previously unavoidable.

The exhibit of this new scheme of treatment operating on a small tonnage basis, which was so kindly thrown open to examination by our party while at Chino, was very interesting.

Miami has been experimenting for some time along the same line and has achieved success in a laboratory scale. Flotation has been successfully used in the separation of ultra-fine native copper in Michigan, and there seems to be much promise in this new scheme of treatment for the recovery of copper in low-grade ores when in a mixed condition of oxide and sulphide form. As Dr. Morse has suggested, the development promises to be important, and we may be on the threshold of another important step in copper metallurgy. I congratulate him.

(Here Mr. Cole read an abstract of his discussion of Dr. Gahl's paper which was printed in the October, 1916, *Bulletin*, pp. 1870 to 1875.

Rudolf Gahl.—Mr. Cole's question, why hydraulic classifiers were installed in the Inspiration mill, is very pertinent, and his doubt about their usefulness for the classification of the deslimed flotation machine tailings seems well justified. In defense of the installation, I will say this:

1. The expense of operating these classifiers is very low, the principal cost item being labor for their attendance.

2. It requires a very small settling capacity to settle the table tailings. At the Inspiration mill it is accomplished by three 60-ft. Dorr settling tanks, with an additional tank installed for the purpose of keeping the coarsest sand out of the Dorr tanks. This small installation handles more than 7,000 tons per day. As the reclamation is accomplished right at the foot of the mill, the reclaimed water does not have to be lifted very high, and the reclaimed cost for this water is, therefore, low.

In other words, the benefit derived from the installation of hydraulic classifiers does not need to be very great to make them pay.

It is true that when we operated our test mill, we succeeded in obtaining quite satisfactory recoveries in spite of the fact that we put the whole of the flotation machine tailings on the tables without preliminary classification. We realized, however, that the table tailings (at least the coarser sizes) were not as low in their copper contents as we hoped to have them some day. We knew, furthermore, that by grinding finer we could reduce the coarser screen sizes in copper. As grinding finer than to a certain point is an economical impossibility, we had it in view to make different sizes and to send them to separate tables. This separation we intended to accomplish in hydraulic classifiers. The coarser tailings, or

perhaps only middlings from the tables, could then be reground and reconcentrated.

To Mr. Cole's question as to what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation, I have to reply:

That we have not made the test which he suggests, but that I am inclined to agree with him that the tables might do as well on an unclassified as on a classified feed. The hydraulic classifiers were put in as I have indicated before, in anticipation of further development of our flow-sheet.

We are doing a great deal of experimental work on this special point, but have not gone far enough to predict just what the details of the additional sand treatment will be.

R. S. HANDY, Kellogg, Idaho.—I would like to ask if anyone has determined the relative economic efficiency of flotation as compared with gravity treatment on granular material.

C. E. MILLS.—I don't know whether I could answer that, but would say that through some experiments which Dr. Gahl has mentioned on the coarse feed to the tables, we have come to the conclusion that we do slightly better by flotation than by tabling of that material.

E. P. MATHEWSON, Anaconda, Mont.—I would like to say, in regard to Mr. Handy's question, that it is the practice in Montana to take out everything that is possible by means of tables or other water concentration machines; and at the Anaconda plant we take out concentrates 1½ in. size, and keep on taking out finer and finer material by water concentration until we get to the tables. What is left from the tables and not saved is then ground up and put through the flotation machines. We find by our experiments that it is better to keep the slime separated from the sand.

R. S. HANDY.—In experimenting on lead ore, I found it advisable to separate the granular material, tabling the sand material; and I wondered if that same experience had been gone through in copper ore.

C. E. MILLS.—That is similar to what we are doing at Inspiration in our callow sections. The pulp is first subjected to a primary flotation, then sent to drag belts which separate slime from sand. The sand is tabled while the slime is given another flotation treatment.

DAVID COLE.—I consider it unnecessary to first remove the slime, for the reason that after the slime has been subjected to flotation treatment there is nothing in it that a table treatment can save—no sulphides in a sufficiently fine state of division to be transported by the slime, because the previous frothing operation has removed it all, and when slime is

"denatured" in this manner it is no longer harmful upon the table, and does not interfere with the working of the sands upon the table. The table will do exactly the same work upon the sands that it would do if the slimes were not going across and off in the rear of the sands with the excess water. If no previous division or washing out of slime is practiced we have gained to the extent of the trouble and cost that would be entailed in making the division. That is the way I view the matter.

RUDOLF GAHL.—I would like to express a doubt as to Mr. Cole's statement. Our experiences distinctly show that we get a better sand recovery by flotation if we remove the slime beforehand.*

E. P. MATHEWSON.—I beg to confirm Dr. Gahl's statement as to the practice at Anaconda.

R. S. HANDY.—I would like to say that I have separated a minus 200-mesh lead ore into granular and flocculent matter, treating the former on tables and the latter by flotation. By this treatment I have recovered 92 per cent. of the total lead, in a concentrate assaying 63 per cent. lead. By flotation of the total ore I could recover only 85 per cent. of the lead when the concentrates assayed 63 per cent. lead and in order to get 92 per cent. extraction, the best grade of concentrates I could get was 50 per cent. lead.

C. E. MILLS.—I wish to say, in response to Mr. Cole's question, that I think Dr. Gahl overlooked something that we have at the Inspiration mill—a section that is running tables on an unclassified feed. The results from that section are just as good as in the others. I think that may be due to the fact that we do not do very good classification.

F. G. COTTRELL, San Francisco, Cal.—There is one point in connection with what Mr. Handy was saying. I think what he has in mind is the distinction between sands under 200-mesh. It is more of a physical distinction than a mill distinction. I mention it because it might be something for the practical mill man in drawing a distinction between clay material and the very finely ground crystalline material. It is customary to lump under slimes everything, irrespective of its chemical properties.

C. E. MILLS.—I think Mr. Cole has done a lot of work along that line. He might give us some information about it.

DAVID COLE.—In my paper "The Advent of Flotation in the Clifton-Morenci District," I referred to that. In devising the early flow sheets for

* NOTE by Dr. Gahl: I judge from Mr. Cole's remarks in the form in which they appear on the stenographer's transcript of the discussion after being corrected by Mr. Cole that very likely he did not refer, as I understood him to do in the discussion, to the flotation treatment of a mixture of slime and sand. If this is so, my answer does not fit the case.

Morenci we thought it would be advantageous to make the separation because we did not at the time consider flotation as at all applicable to the problem. In my paper I have alluded to the methods devised and reasons for making the divisions.

We knew, of course, that the separation could not be made upon screens. Hydraulic or washing separation I considered to be impracticable after trial of some of Mr. Overstrom's methods, and spitzkasten or pointed box or Callow tank separation is only partially successful. A better result would be accomplished by shortening the period of settling. To go into an explanation of why this is so would take too much time, but those who have been dealing with slime and know its peculiarities in settling and coagulation—its power to arrest the falling of successively larger particles of the ultra-fine sands which Mr. Handy has in mind—will realize what I mean by hastening the removal of the sands, and we developed a machine which was probably misnamed a "colloid separator," the office of which was to remove the millman's type of "colloid"—the very much diluted clay—before it should have time to settle into a mass that would seize and hold the granular material which we knew we *could* treat successfully on tables or vanners, and at the same time allow the dilute clay, which it did no good to treat by table or vanner, to escape to the tails. In the light of our present knowledge of flotation treatment, we realize that this overflow, the dilute clay material referred to, is the part best prepared for flotation treatment, and outside of the oxidized copper content would yield the lowest tails and the most complete recovery, and that it is not necessary to remove the ultra-fine sands for flotation treatment, for these are best removed by flotation also.

The colloid separator that was devised for making the separation consisted of wide thin belts dragging upon the bottom of a shallow basin (2 in. deep), across which the pulp was fed in a wide, thin, gently moving stream. The belts covered nearly the whole of the bottom of the basin (2 in. apart) and therefore practically the whole floor of the basin was (figuratively speaking) moving toward the discharge, where it rose gently above the surface of the water, received a gentle washing from clear water sprays like the sprays used upon vanners, and discharged its load upon vanners for gravity treatment, where a good grade of concentrate and a low tail were made. I think the illustrations in my paper will make the matter clear.

I have noted Mr. Mathewson's remarks concerning the present Anaconda practice, and that he has the impression that they really are desliming the feed before treatment on tables. The Anaconda flow sheet shows such a separation and the Anaconda type of conical deslimer is installed with that end in view, but like Mr. Mills' admission of a few minutes ago as to Inspiration's poor classification practice, Anaconda

doesn't do good classification. The feed to the tables is not deslimed as it was intended it should be, the reason being that there are not enough of the Anaconda classifiers to do the work put upon them, and since the only office of these classifiers is to prepare feed for the Butchart riffle treatment, and since these tables have no office but to impoverish the ore treated by them, the complete separation of the slime from the feed is of little consequence, for all reject from the tables is taken at once to the regrinding mills where the cleaning-up work is most thoroughly accomplished by the flotation process. If the tables were making a reject to tailing, the Anaconda classifiers would have to do their full duty in desliming the feed to them, because the slimes going across the Butchart tables would result in serious losses, but since it is immaterial whether the primarily made slimes reach flotation treatment over the top of the classifiers or through the spigot, the classifiers' inefficiency and the results as to the reject from the tables are tolerable and there does not seem to be any reason to change. Obviously, it does not matter at Anaconda where the copper is taken out so long as a minimum amount of it is allowed to get away with the final tailing, and it is also obvious that with Anaconda's present practice wherein the ore is reduced by their splendid treatment scheme from a 60-lb. copper content to less than a 3-lb. copper content in the final tail on a ratio of practically 3 into 1, the chance for improvement in mill practice through modification or more perfect slime classification is extremely remote.

Guy H. Ruggles, Miami, Ariz.—To get back to the subject of classification at the Inspiration: In the early part of the year tests were made showing that the section which treats the pulp from the flotation machines directly upon the tables, that is, without classification, did as good work as the other sections in which the pulp was classified. Since that time we have improved the settling in our drag classifiers The average of the general tailings on a number of sections for the month of July and August shows that the section which treats the flotation pulp directly upon the tables is, if anything, not quite as good as the sections in which we have classification. There are some sections which beat this section by 0.02 per cent. copper and others which beat it by only 0.01 per cent. copper. Every 0.02 per cent. which we reduce the general tailings means 180 lb. of copper per 24 hr. per section, and when this is multiplied by 18 or 20 it means quite a lot. While our classification is not very good, it has helped out quite a little and if improved we might get more out of it.

R. C. Canby, Wallingford, Conn. (communication to the Secretary*).—At the time I was working upon flotation with the late Robert S. Towne, he told me of his interesting experience in the use of iron balls in a sort of ball mill, with iron balls used simply as an oiling device in connection with the Murex process.

Now in the Murex process there is no question of modification of surface tension, etc., simply the oiling by preferential affinity of the sulphide surfaces that the pulverized magnetic oxide may be made to adhere to the sulphides and not to the gangue.

Mr. Towne believed that the iron balls readily became coated with the oil and that *then* the sulphide surfaces very readily became coated as the particles—through the ball-mill action—came into contact with the surfaces of the oil-coated iron balls.

May it be that metallic iron has a somewhat similar assisting function in the cases referred to by Dr. Gahl? If so, it would suggest that an oil film upon the sulphide surfaces may be an assistance, if not altogether a necessity, in the frothing flotation process.

I had not intended to give Dr. Gahl the impression, as expressed in the footnote on page 1637 of the September *Bulletin*, that the experiments which I had conducted had given to Mr. Towne the idea of the porous medium for disseminating air bubbles. It was furthest from my thought to make such a claim.

What I suggested was the use of a Frenier pump spiral for incorporating the air bubbles, having in mind the converse of the Elmore vacuum, in which I proposed to produce the bubbles by pressure instead of by a vacuum. Upon investigation, however, I found that Norris had first suggested using air bubbles in solution.

While the manner of releasing the discharge from the Frenier into a separating compartment gave possibly an effect somewhat that of the "bubble column," still it was not actually the "bubble column" in the sense the word is now understood, and I did not intend to claim it as such.

I wish I felt it proper to take the space here to more fully attest the remarkably analytical manner in which Mr. Towne studied into all of the problems which he encountered, taking as he always did, the lead in the constructive thought of his staff, with whom he worked in close touch.

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* Received Sept. 14, 1916.

## Flotation Concentration at Anaconda, Mont.

Discussion of the paper of FREDERICK LAIST and ALBERT E. WIGGIN, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 111, March, 1916, pp. 549 to 581.

O. C. RALSTON, Salt Lake City, Utah.—I have merely glanced over this paper, consequently, I am hardly in a position to discuss it intelligently. There is one thing, however, that is of interest, that is, the claim that the oil mixture which they have chosen is the best adapted to the work at hand. I would like to ask Mr. Mathewson if they still feel that that is the case—if that really does give the best grade of concentration and the best extraction for the same money, compared to any particular oil mixture which they could get.

E. P. MATHEWSON, Anaconda, Mont.—We are still using that mixture, and we consider it the most economical mixture used.

DAVID COLE, El Paso, Texas.—I would like to ask Mr. Ralston if there are some other mixtures that would be better?

E. P. MATHEWSON.—We are open to suggestions.

O. C. RALSTON.—My own paper on oils* calls attention to the fact that certain hardwood oils, especially the tars, are now being burned or wasted. A great deal of hardwood tar is available as low as 4 or 5 c. per gallon, that being its value as fuel under the boilers. As a matter of fact, a great deal of testing has been done with only the commercial products that are as a usual rule sold on the market. Very little hardwood tar is sold on the market. In asking a great many people who have tested flotation oils, I have not heard hardwood tar mentioned very often. I think I would add to that that the hardwood tars can probably be bought in quantity comparable with the sludge acid at Anaconda, and should contain a greater proportion of the frothing constituent, and at a price fairly commensurate with what they pay for the sludge acid.

E. P. MATHEWSON.—I would like to make a correction in one statement I made that there has been no change made in the mixture. This change has been made: We found that the amount of wood creosote in treating the sand from tables was extremely small, and we tried some experiments on a large scale, dropping it out and using simply the sludge acid and sulphuric acid. We found that this gave practically as good results as wood creosote. We use wood creosote in treating the slimes. We find it necessary in that operation.

DAVID COLE.—I would like to ask, what is sludge acid?

* Some Miscellaneous Wood Oils for Flotation, *Bulletin* No. 116, August, 1916.

E. P. MATHEWSON.—Sludge acid is refuse from the refining of oil. It contains sulphuric acid and some greasy material from petroleum.

DAVID COLE.—I believe that you are making sulphuric acid very cheaply at Anaconda. Since sludge acid consists of coal oil and sulphuric acid, I have wondered if you could not compound it at Anaconda more cheaply than you can buy it.

E. P. MATHEWSON.—We have made sludge acid at Anaconda, but it is more expensive than that on the market. We got good results with the acid we manufactured.

RUDOLF GAHL, Miami, Ariz.—Some of the wood oils have the drawback that the concentrates produced with them have the tendency of retaining some of the gangue very firmly and it is difficult to settle such concentrates. We have found very few we could use. We have found some pine oils of the more refined character, but we cannot do anything at all with pine tar. Hardwood tar may have different qualities, and I would like to ask Mr. Ralston and Mr. Allen if they have given some attention to the physical characteristics of the froth made by the wood oils which they have tested.

O. C. RALSTON.—Of course, the tests reported in our paper on wood oils are laboratory flotation tests, and give the flotative value of the oil and do not discuss later mechanical treatment of the concentrates. We cannot tell what will happen later in that part of the mill. That is another point. Sometimes you cannot tell even when you get to the mill what you can do with such a froth. I think Mr. Cole's patent for breaking froth is a rather interesting sidelight on that subject. To answer Dr. Gahl directly, our suggestions are only based on laboratory experiments on the flotation problem alone.

RUDOLF GAHL.—Mr. Ralson has done so much work on the theoretical end of flotation, he might be able to answer a question which I have often put to myself. Why is it that at Anaconda they can float chalcocite ore (Is it not chalcocite ore?) with acid and when we add acid here we spoil our flotation altogether?

O. C. RALSTON.—Some day when I understand what flotation is—I will answer that.

NORMAN CARMICHAEL, Clifton, Ariz.—One point which has not been touched on in this discussion today. In going through the Old Dominion concentrator, I noticed that they were using what appeared to be caustic soda. I think if there is anyone here who can give us any explanation in regard to the use of the caustic soda, it would be interesting.

W. B. CRAMER, Globe, Ariz.—The point which Mr. Carmichael has just spoken of, I shall try to answer. I was interested in what Mr.

Mathewson said about the use of acids at Anaconda. I believe the analyses of the Anaconda and Old Dominion slimes would be much the same. We find, as Dr. Gahl finds, that the moment we use acid in flotation, our flotation suffers considerably. In fact, it almost ceases. If we use a pulp slightly alkaline, the flotation work is satisfactory. Caustic soda is expensive, and we are using about 1 lb. to a ton. A distinct advantage that caustic soda has at Old Dominion, at present we are running on high flotation feed, averaging 3 per cent., is in the matter of settling. We are installing a new Dorr thickener, and our capacity so far as thickening concentrates are concerned is very limited. Soda has the effect of flattening out the froth considerably, allowing us to cut down the water in the launders at machine less than half, which allows us to handle the concentrates in the Dorr thickener very well. Without the use of caustic soda, we may lose 4 or 5 tons of concentrates a day. It goes over and is recovered later in the secondary tanks. Why the soda should have this effect at the Old Dominion and the opposite at other places is hard to explain. We cannot use acid. We get better results with the caustic soda. In other places the opposite is found. At Nacozari a few years ago we were not able to get a satisfactory froth without acid. If we put acid in small amounts, the pulp still being non-acid, the froth was light in character; but increasing the amount of sulphuric acid until pulp was acid to methyl orange, a splendid froth was obtained.

C. W. Merrill, San Francisco, Cal.—But the use of caustic soda is solely for the purpose of decreasing the amount of water from the overflow?

W. B. Cramer.—Caustic soda does increase our extraction. So, in a matter of dollars and cents, it pays us to use caustic soda, but it brings the cost per ton for treatment up about 5 c. per ton. It does increase our extraction, increases the copper content of the froth and lowers the insoluble content.

C. W. Merrill.—I understand that the purpose is threefold; to cut down the water, increase the extraction, and clean the concentrates.

R. S. Handy, Kellogg, Idaho.—I would like to suggest that in the treatment of refining petroleum, acid is used, which before the treatment is finished is neutralized by caustic soda, so that the residue is sludge acid. If I am not mistaken, it contains a considerable quantity of caustic soda. That has been my experience in California. I would like to know if that is true.

O. C. Ralston.—The acid sludge from petroleum refining is usually a product obtained by the addition of strong sulphuric acid to the partially refined petroleum products in order to remove certain impurities,

and is usually removed before sodium hydrate is added to the remaining oil to neutralize the small amounts of sulphuric acid held by the oil. Hence it consists of strong sulphuric acid combined with some of the dark tarry constituents of the oil. Good flotation acid sludge comes from the California oil fields and titrates about 50 per cent. $H_2SO_4$. The other constituents are apparently the sulphonated asphaltic compounds contained in the oil. On that account sludge acid does not contain sodium hydroxide. Turning to the question of the use of sodium hydroxide in the flotation pulp and the reason for the good results obtained, I think that an explanation is easy. This substance, as well as other substances of alkaline reaction, like sodium silicate and sodium cyanide, is used in a number of mills which have a great amount of clay and other finely divided material in the ore pulp treated. Such an ore pulp is usually partially flocculated and sodium hydrate is known to be a powerful reagent for deflocculating certain gangue slimes. That is, its hydroxyl ions are adsorbed into the film of water in contact with the surfaces of individual particles, in greater proportion than are the sodium ions. This amounts to giving these particles a stronger electric charge of the negative sign and they fly apart (are deflocculated). In case any sulphide particles had been entrained in the flocs of gangue, this deflocculation liberates these sulphide particles for flotation. This explains the statements by Mr. Cramer that he gets better extraction and higher grade of froth by the use of caustic alkali in the pulp. I am informed by other operators that it also reduces their consumption of flotation oil as much as 50 per cent. in some cases. This seems reasonable, as the sulphide particles are all liberated and ready to be oiled so that oil will not have to penetrate to the center of a floc to oil the entrained particles of sulphides. Further, the great number of oils used are more easily emulsified in alkaline solutions. In the cases where the use of sodium cyanide has proven beneficial I am not sure that the alkaline reaction of the reagent is the sole contribution of this reagent to improvement of the flotation, but the case is interesting because Charles Butters and associates claim that it is an impossibility to float in the presence of cyanide. In at least two instances coming under my observation, the flotation of the sulphide minerals was improved by the addition of cyanide. It is just another instance of being told that certain things cannot be done in flotation, only to discover later that with proper care and right conditions they can be done. Don't believe any one who tells you that you can't float under some particular conditions. He may be partially right but the statement of negative results should always receive the qualifying phrase, "as far as I know, and to the best of my experience." For instance, Dr. Gahl claims that acid spoils flotation at Inspiration. I feel confident that the right conditions for an acid flotation of that ore could be found.

David Cole.—When we were at Chino a few days ago, we saw flotation machines handling vanner concentrates in which they were using alkali and rosin as the flotation agent. Mr. Ralston looked at this, and I would like to have him tell us what we saw.

O. C. Ralston.—That solution is a solution of sodium resinate, the idea of the alkali being to get the rosin into solution. The rosin has a very good effect on the froth, giving a particularly stable froth. They were working under conditions where the oil froth wanted to die, and the addition of the rosin was the proper thing to bring up its strength and allow it to rise until it got over the discharge board. I think there was no intended significance in the additional alkalinity. In that case the alkali was added for a particular purpose, simply as a solvent for the rosin.

The Chairman.—I understand that Mr. Ralston wishes to get some questions answered by the members of the Institute, and I will ask him to kindly present those questions.

O. C. Ralston.—A great many of the questions I am interested in have been asked and answered this afternoon, so that you will all welcome the fact that the list can be cut down. This list was presented by D. A. Lyon, who sent them out in a circular letter all over the country. So far I have not heard of anyone who has prepared a list of answers for publication. I don't see why people need to be afraid to answer them. The other day somebody said, "it is safe to talk about theories of flotation, because none of us know anything about them." I will only pick out a few questions. The first one is this: What is the effect of dilution of pulp with water on the flotation of the minerals contained? The reason for asking that question is, of course, obvious. There are enough large-scale operators who must have tested this question out. So, it will be interesting to get the consensus of opinion. I think that Dr. Gahl might help us.

Rudolf Gahl.—Well, I would say offhand that the effect of dilution is to make a cleaner concentrate, and to make it more difficult to produce concentrate. I think it takes more oil to produce the same amount of concentrate from a dilute pulp than it takes to produce it from a thicker pulp.

O. C. Ralston.—Might I supplement by asking you what determines the amount of oil necessary for flotation? Is it the amount of water you are using, making a certain strength of solution of frother in water, or is it the amount of mineral in the ore? If you had an ore consisting of 50 per cent. mineral, would you use more oil than if it contained 5 per cent.? What amount of oil must be used in flotation?

Rudolf Gahl.—What amount of oil must be used in flotation? I do not feel qualified to answer Mr. Ralston.

O. C. RALSTON.—The Butte and Superior ore, containing 30 per cent. of mineral, as compared with the Inspiration ore with 5 per cent. or 10 per cent. of mineral, nevertheless uses less oil, if anything. So, the proposition would be that the amount of mineral in the ore is not determinative of the amount of oil necessary. It must be the amount of water in the pulp which is determinative of the amount of oil necessary.

DAVID COLE.—I think the kind of oil has a lot to do with the amount used.

O. C. RALSTON.—That is a question not of theory but of experience. For instance, the amount of pine oils being used in the Coeur d'Alene district is often less than 1/5 lb. oil to the ton of ore, while at Anaconda they use several pounds of oil per ton. At Anaconda they use a considerably less expensive oil.

A. P. WATT, St. Francois, Mo.—Mr. Ralston has asked the question whether the amount of oil used in flotation is proportional to the water or to the solids in the feed to a flotation machine. My experience with the flotation of lead ores may be of interest. In one particular case a flotation machine was receiving a feed with a ratio of solids to liquid of 1 to 7. This ratio was later changed to 1 to 3.5 and the amount of oil used was decreased approximately 50 per cent. This particular case would seem to indicate that the amount of oil used in flotation is proportional to the water rather than to the solids in the feed. I also found that the extraction was increased by the use of a thicker pulp. This result would be expected as the volume of pulp passing through the machine was decreased, thus permitting it to be acted upon for a longer time than would be the case with a thinner feed.

DAVID COLE.—Speaking of the small amounts of oil—we carried out some experiments the other day at El Paso, and on a density of 7 to 1, approximately, on an ore that carried 6 per cent. lead, 9 per cent. zinc, and 1.3 per cent. copper, with the use of 2/10 lb. of cresylic acid per ton, we were able to take out a large part of the lead. Then, by the addition of 3/10 lb. of No. 350 pine oil per ton, we were able to take out nearly all of the zinc. The products made assayed as per the accompanying table.

| | Per Cent. of Weight | Ounces Au | Ounces Ag | Per Cent. Pb | Per Cent. Cu | Per Cent. Fe | Per Cent. Zn |
|---|---|---|---|---|---|---|---|
| Lead concts. | 12.86 | 0.110 | 12.85 | 35.7 | 7.32 | 7.7 | 13.0 |
| Zinc concts. | 25.08 | 0.025 | 2.67 | 4.4 | 1.30 | 3.9 | 23.6 |
| Tailing | 59.68 | 0.005 | 0.45 | 0.3 | tr. | 3.0 | 1.9 |

NOTE.—This was done with a single operation without the use of cleaner for the concentrates.

There was very little oil used—a total of ½ lb. per ton of ore. If we had used a larger amount of oil, we would not have been able to get that separation. We put in just enough cresylic acid primarily to get that result.

C. E. MILLS.—I think Mr. Gottsberger might contribute something. The fact is that the Miami Co. and the Inspiration Co. are treating similar ores, except we have not as much copper as they have. We use quite a little more oil in flotation than Mr. Gottsberger does; and I think, perhaps, they have got more water in the pulp. Is that about the condition, Mr. Gottsberger?

B. B. GOTTSBERGER, Miami, Ariz.—I am sorry to say I could not give the density of the pulp.

C. E. MILLS.—We are using 1:3.

B. B. GOTTSBERGER.—Our oil mixture is much thinner because we are using less coal tar. I think we have found that with a thick mixture composed very largely of coal tar it is really necessary to add the oil in the grinding mill. We do find, however, that it is not essential to add these oils in the grinding mill in order to get good flotation work. At present the necessary mixture is obtained by adding the oil in a bucket elevator.

C. E. MILLS.—Can you give us another question, Mr. Ralston?

O. C. RALSTON.—I might ask this question, which would supplement Mr. Handy's question: Are ores which contain very much fine colloidal material harder to treat successfully than granular ores? As far as I know, they usually are. Dr. Gahl has mentioned something about the treatment of them. I would like to pick on Dr. Gahl again to answer that.

RUDOLF GAHL.—I would say that colloidal ores are harder to treat than granular ores.

O. C. RALSTON.—There is a question in my mind. Is it possible to treat by any method of flotation colloidal material? Would it be possible to deflocculate that material and separate the granular material? In our experimental laboratory we did work on material sent us by Mr. Watt which approximated that condition. It was from Missouri, in the disseminated lead section, where the leady limestone had weathered away, leaving practically nothing but clay. The galena oxidized to lead carbonate, and Mr. Allen's attempts to sulphidize and float that ore were completely unsuccessful. It was practically nothing but a clay—about 100 per cent. colloidal material. So, a question arose: How much colloidal material is allowable in an ore?

C. W. MERRILL.—Do you mean only colloidal gangue or the whole ore to be of colloidal nature?

O. C. RALSTON.—This question is very well put. No one seems to have studied the question as to whether only the gangue in an ore is colloidal. "Colloids" are blamed for a lot of trouble and it occurred to me that whether the flotative minerals are reduced to colloidal size and condition, or not, colloidal gangue might cause trouble.

W. B. CRAMER.—Our experience at the Old Dominion on primary slimes differs from that at other plants. Our mill feed contains about 15 per cent. minus 200-mesh material which may properly be called "primary slime." This slime was separated in the first compartment of our fine jigs, thickened in a Dorr thickener and treated separately from the secondary mill slimes by flotation. An extraction of 84 per cent. was obtained in the flotation treatment of this purely primary slime. Hence it may be stated that the primary slime at the concentrator of the Old Dominion Copper Mining and Smelting Co. is as readily adapted to flotation as the secondary slimes.

C. W. MERRILL.— I can conceive of colloidal ore where the mineral values were extremely finely divided that could not be treated.

A. P. WATT.—Referring to the ore from Missouri of which Mr. Ralston has just spoken. I believe that Mr. Ralston and Mr. Allen were unable to make any extraction by flotation on the sample of slime I sent them. When, however, we took a more granular sample we were able to make some separation. We found that a good extraction could be made on the granular portion if the colloidal portion were first eliminated. The colloidal portion of the ore, however, appeared to prevent flotation.

The sample of slime submitted to Mr. Ralston was extremely fine. Although it contained 2 per cent. lead no signs of cerusite could be detected when the slime was treated on a vanning plaque. Whether or not the cerusite was in a "colloidal condition" I do not know.

F. S. SCHIMERKA, Clifton, Ariz.—In connection with the question brought up a short while ago as to the different quantities needed of oil to successfully float, I wish to ask one question. Has it ever been noticed by the flotation experts present today whether it makes a difference as to what density the pulp is in when the oil is added in the grinder or mixing machine—whether a pound of oil goes farther at a high density?

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## A New Flotation Oil and A New Source of Flotative Agents

Discussion of the papers of MAXWELL ADAMS AND G. H. CLEVENGER presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1683 to 1684 and 1685 to 1692.

OLIVER C. RALSTON, Salt Lake City, Utah.—We are indebted to these two gentlemen for the work which they have done with the oils derived from sage brush. Their work should not be confused, however, as the work of Mr. Adams deals with steam-distilled oils and that of Mr. Clevenger with destructively distilled oil, largely. In other words, Mr. Adams' oil is probably the original essential oil that existed in the brush before distillation while Mr. Clevenger's oil not only contained this essential oil but also some products of destructive distillation of the wood constituents.

I had noticed the work of Mr. Adams on attempting to separate the various constituents in the oils of some of the Western conifers and had sent to him to ask if he could send me small samples of some of these pure products. This he very kindly did, and with them included some samples of oil from the common sage brush, whose characteristics he has here described, as well as a sample of oil from the common "rabbit brush." Only the sage brush oil proved of much value for flotation, but it was very powerful, and I so informed Mr. Adams. Even in amounts as small or smaller than $\frac{1}{10}$ lb. per ton of ore, it gave very quick and positive flotation of many ores, with high-grade concentrates and good extractions. It proved to be especially desirable for the flotation of carbonates of lead and of copper which had received an artificial coating of sulphide by treatment with a solution of a soluble sulphide.

I also received a sample of crude pyroligneous acid from sage brush prepared by Dr. W. C. Ebaugh, of Salt Lake City. It did not do very good work in flotation, but it is possible that by concentration of this product to obtain the residual tar a good flotation oil might result. This has not been done as yet and I am sorry to see that Mr. Clevenger has not done this. It might be that the dissolved tar, added to the settled tar, would make a total yield of flotation oil higher than the 4 per cent. which Mr. Clevenger feels is a safe estimate. The importance of this point will be seen in the following discussion of the possible costs of sage brush oils for flotation.

I have been fortunate in obtaining the approximate costs of collecting sage brush, as well as the costs of operation in the ordinary wood-distillation plants. I was given the former by E. H. Snyder, who with a number of associates cleared some land of sage brush and made potash from the

ashes of the burned brush. The latter were given me by R. C. Palmer, formerly of the Forest Products Laboratory of the Department of Agriculture.

The clearing of land of its sage brush is ordinarily accomplished by means of a tractor pulling a special frame made up of railroad rails. This breaks off the bushes so that they can be collected in hay ricks and hauled to a central point. For merely uprooting the plants and burning them on the spot the cost is about $4 per acre, but to recover them and take them to a central point ready for burning or distillation, costs about $6 per acre.

Mr. Snyder further informs me that with the average 4-ft. stand of brush in southeastern Nevada the yield per acre is about 7 tons of brush. I am informed that there are considerable stretches of country around Bear Lake in Idaho and Utah where the brush is nearly 10 ft. high. Hence considerably higher yields might be possible. However, we can take as a safe figure $1 per ton for cutting and collecting the brush to a central point.

The cost of destructive distillation of hard wood is in the neighborhood of $8 per cord (4,000 lb.) of wood, or about $4 per ton—possibly $5 per ton of wood. While the sage brush might be more bulky than the wood, and hence cause higher costs, this factor can not be estimated at the present time and it is probably best to count on a distillation cost of $5 per ton.

That would make the total cost of treatment of each ton of sage brush about $6, and the yield, according to Clevenger, is 4 per cent. or 80 lb. of tar. This would mean about 10 gal. produced at a cost of $6, or $0.60 per gallon, or 7.5 c. per pound. This is rather high but is comparable with the present price of pine oil, of which sage brush oil seems to be the full equivalent, if not the superior. With most ores, less than 0.5 lb. of sage brush oil should be needed per ton of ore.

If Clevenger's figure of 4 per cent. yield of oil could be raised to 6 per cent. by addition of the dissolved tar from the pyroligneous acid, the cost of the oil would be about 43 c. per gallon. Further, if greater yields of brush from each acre were obtainable, the costs would be cut slightly, although this point is not of so much importance, as the principal costs are due to the distillation. It is also possible that in some cases owners of the land would be glad to pay $1 to $2 to have their land cleared of brush, where it would usually cost them about $4. Only skillful management and the choice of the proper regions for operations would probably bring in this latter source of income.

There is one other source of income from the products of the sage plants, namely, the potash. Mr. Synder informs me that he found the potash in carefully burned brush ashes to amount to 15 to 20 per cent. of the total weight. Most of this was soluble in water but the remainder

was "acid soluble $K_2O$." The ash amounts to 7 to 10 per cent. of the weight of the plants. Hence the $K_2O$ content of the brush can be estimated as from 1 to 2 per cent., or 20 to 40 lb. per ton of brush. In normal times potash is worth about 3 c. per pound. This would make the value of the potash in the brush about $0.60 to $1.20 for every ton. The cost of leaching and crystallizing the potash from the ashes is unknown, but might possibly be $3 on every ton of ash or 20 to 30 c. referred to every ton of brush. It is not impossible that the burning of the sage brush charcoal under the boilers of the distillation plant, followed by leaching of the ashes would yield potash of sufficient value to allow a profit, and hence allow of production of sage brush oils for flotation at a lower net cost than the above rather pessimistic figure. One difficulty at the potash end of the work is that it tends to be carried away in the combustion gases and a Cottrell precipitator might have to be used in order to insure recovery of all the potash.

The above is merely a suggestion to show what might be expected if one were to attempt the commercial production of sage brush oil. Rather high costs were assumed, but it is certain that other sources of expense than those enumerated will make such a pessimistic figure desirable, for the sake of safety.

About 10,000 gal. of steam-distilled pine oil are being used every month in the United States for flotation purposes and more would be used if the price were lower. The last quotation that I heard for pine oil was 67 c. per gallon, f.o.b. New York. If sage brush oil could be produced for 40 c. per gallon it is probable that the market would jump to at least 25,000 gal., or roughly 1,000 gal. per day. With this would be produced 3,000 to 4,000 lb. of potash, an amount which is much less than 1 per cent. of the total consumption in the United States. Hence the sale of potash from such a source should have no bad effect on the market. On the other hand, if the oil could be produced for 25 c. per gallon, there is no reason why its use should not amount to 10 times the consumption of the oil at a 40-c. rate.

It is to be hoped that these figures are attractive enough to cause experimentation on a reasonably large scale, to determine whether operations can be profitable. Only by such a method can it be estimated whether we have a new source of flotative agents and a new source of potash.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Conservation of Phosphate Rock in the United States*

BY W. C. PHALEN,† PH. D., WASHINGTON, D. C.

(New York Meeting, February, 1917)

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* Published with the permission of the Director of the U. S. Geological Survey.
† Mineral Technologist, U. S. Bureau of Mines.

## INTRODUCTION

Nobody will dispute the fact that the conservation in every legitimate manner of our valuable high-grade phosphate-rock deposits is a present-day problem of importance.

The table and curve, given herewith, show that during the past 10 years the exportation of high-grade rock has averaged very close to half the total output of the entire country. The bulk of this exportation is from Florida, for obvious reasons. It is plain that the deposits in this State, more particularly, are being wastefully depleted under a system of selecting the cream of the product for exportation to Europe, leaving the comparatively low-grade rock, running from 65 to 70 per cent., and under, in bone phosphate of lime, for our own fertilizer manufacturers to work up when all the best rock is gone.

In this paper the writer has endeavored to make a point of describing in some detail the important methods of production and conservation in Tennessee. These ought to prove of educational value to our American agriculturists and fertilizer manufacturers, and should result in a demand for the highest-grade rock, both for direct application to the soil and for use in making acid phosphate. It is certainly evident that the European

manufacturer is alive to the situation and is demanding the highest-grade rock from the Pacific Islands and Florida. The cost of transporting in this country low-grade rock and the acid phosphate resulting from it, is another factor which should appeal to the self-interest, if to no higher motive, of the American producer, and user as well. This factor, as well as the important one of keeping our high-grade rock at home, is the fundamental reason for a change in our policy with reference to our high-grade phosphate rock.

## PRODUCTION AND EXPORTATION OF PHOSPHATE ROCK

Since the beginning of phosphate-rock mining in the United States, there has been a total output of 48,457,906 tons, more than half of which has been produced in the past 10 years. During this 10-year period there has been an exportation of nearly 11,000,000 tons, or about 43 per cent. of the marketed production in the same period.[1] The exported material does not represent average grades, but the highest-grade rock, running 77 per cent. and more in phosphate of lime (bone phosphate) and 3 per cent. and less in iron and alumina.

*Production and Exports of Phosphate Rock with Ratio between Them,* 1905–1914

| Year | Production, Long Tons | Exports, Long Tons | Percentage |
|---|---|---|---|
| 1905 | 1,947,190 | 934,940 | 48.0 |
| 1906 | 2,080,957 | 904,214 | 43.4 |
| 1907 | 2,265,343 | 1,018,212 | 45.0 |
| 1908 | 2,386,138 | 1,188,411 | 49.8 |
| 1909 | 2,338,264 | 1,020,556 | 43.6 |
| 1910 | 2,654,988 | 1,083,037 | 40.8 |
| 1911 | 3,053,279 | 1,246,577 | 40.8 |
| 1912 | 2,973,332 | 1,206,520 | 40.6 |
| 1913 | 3,111,221 | 1,366,508 | 44.0 |
| 1914 | 2,734,043 | 964,114 | 35.0 |
| Total.......... | 25,544,755 | 10,933,089 | 43.0 |

## METHODS OF CONSERVATION

### Introductory Note

There appear to be many differences of opinion among soil chemists and agriculturists as to the form in which phosphorus shall be applied

[1] Figures for 1915 are not included in these computations, for the reason that the phosphate-rock industry was in an abnormal condition during that year and also during the latter part of 1914. The production of phosphate rock during 1915 was 1,835,667 long tons, and the exports 253,549 tons, or 13.8 per cent. of the production.

to the soil. One group argues for the use of superphosphate, especially where quick returns are desired, and another for the use of ground rock or

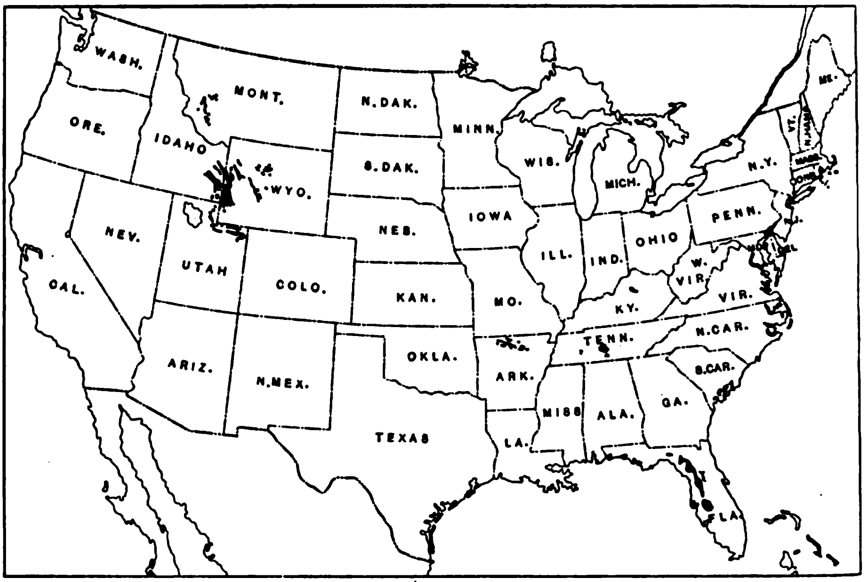


FIG. 1.—MAP SHOWING THE LOCATION OF THE PHOSPHATE-ROCK DEPOSITS OF THE UNITED STATES.

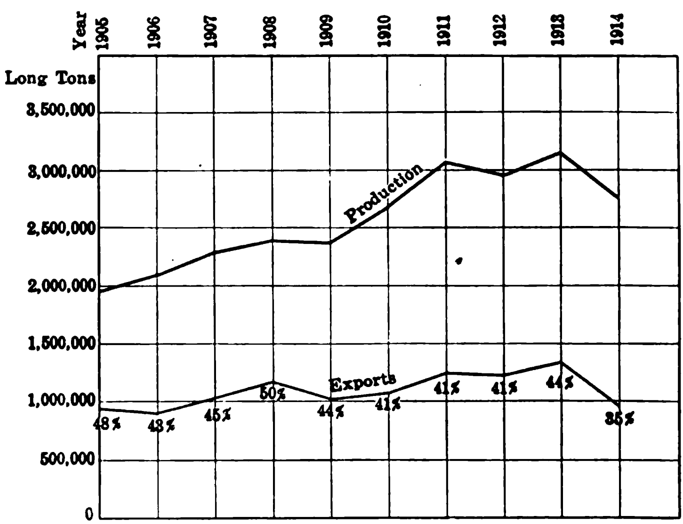


FIG. 2.—CURVE SHOWING PRODUCTION AND EXPORTS OF PHOSPHATE ROCK, 1905–1914. FIGURES ON EXPORT CURVE INDICATE PERCENTAGE OF PRODUCTION LEAVING THE COUNTRY.

"floats," especially where permanent results are the object. Without considering the merits of either side of this question, it is certain that the conservation of our phosphate resources must be of interest to both.

The conservation of phosphate rock will be considered in the following pages, under the heads of (1) mechanical, and (2) chemical methods of conservation, and (3) the use of substitutes, either natural or manufactured, as a source of phosphate of lime or phosphorus.

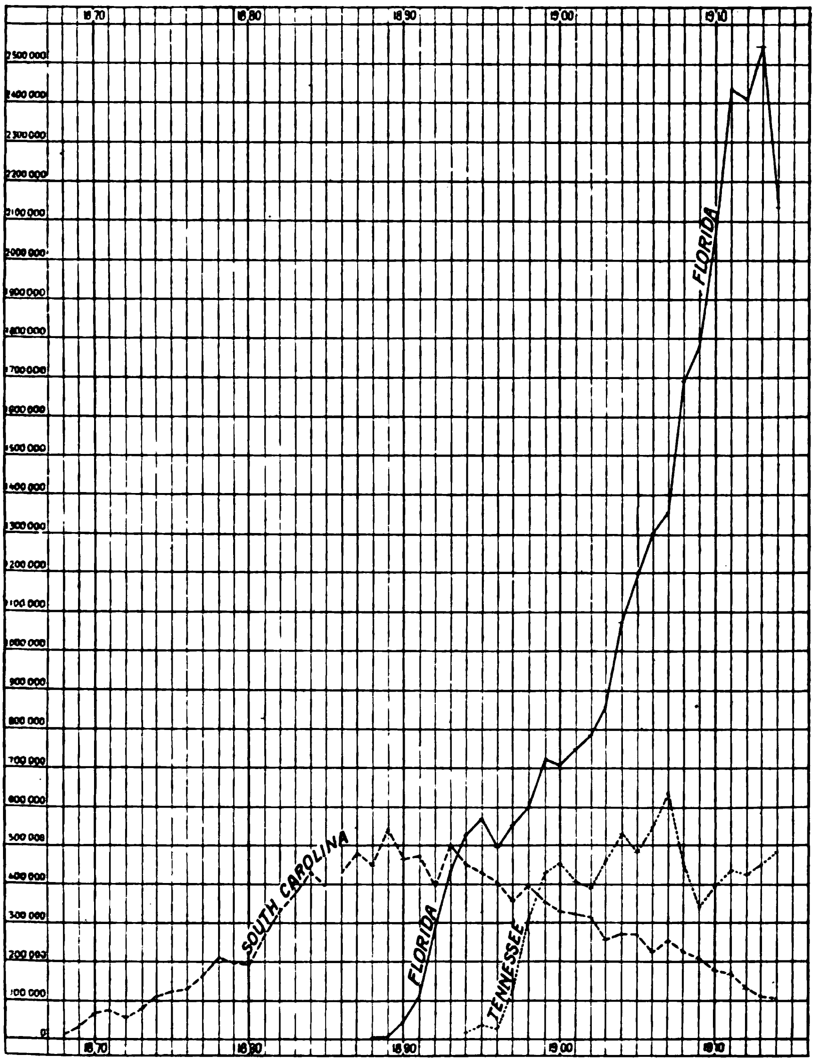


FIG. 3.—DIAGRAM SHOWING MARKETED PRODUCTION OF PHOSPHATE ROCK IN SOUTH CAROLINA, FLORIDA AND TENNESSEE IN LONG TONS FOR YEARS INDICATED AT TOP AND BOTTOM.

## MECHANICAL METHODS OF CONSERVATION

It is well known that in the early days of phosphate mining in the more important phosphate fields of the United States, there was a large

waste of good material. In many places this waste is still going on. In certain places the material once thrown aside is in such condition that it may be reworked and is actually being reworked. In other places, it is lost beyond all hope of recovery in so far as can be told at the present time. The devising of methods to prevent or reduce such losses and yet maintain the grades set by commercial standards is one of the problems that has faced and is now facing the phosphate-rock miner; and in the following pages will be given descriptions of the methods by which the problem is being met in Tennessee, which may serve as a type of what is going on in the other important phosphate-producing States.

In speaking of the early wasteful methods employed in Tennessee, W. H. Waggaman of the Bureau of Soils[2] says: "For years the richest of brown rock in the Mount Pleasant region was worked by hand, and only when these deposits were considered to be nearly exhausted did the operators seem to realize the crudity, wastefulness, and inefficiency of the methods they were using. Even now a few small firms and farmers are employing the pioneer method of shaking out all rock not held by the tines of a potato fork and drying the larger pieces in the sun or on ricks of wood."

The same writer, in speaking of the waste connected with the mining of hard rock in the Florida phosphate field,[3] says: "In order to meet the present demand for a high-grade product, a vast amount of phosphatic material is thrown aside. The marketed product is probably not more than 15 per cent. of the total material mined. The remainder, consisting of sand, clay, and the phosphates of lime, iron, and aluminum, is washed out upon a waste pile. This discarded material varies in its content of phosphoric acid, but seldom if ever contains less than 10 per cent. The actual amount of phosphoric acid discarded, therefore, is almost twice as great as the quantity saved. This enormous amount of low-grade phosphate will no doubt eventually be used."

In speaking of the waste connected with the mining of pebble phosphate in Florida, the same writer makes the following observations:[4]

"The percentage of phosphoric acid washed away in preparing pebble phosphate for the market is fully as great as that wasted in mining hard-rock phosphate. In the vicinity of the large phosphate washers many acres are covered to a considerable depth with this detritus. The author made no determination of the exact composition of this finely divided material, but it resembles the wash from the hard-rock mines. . . . An analysis, however, was made of the material having a diameter greater than 2½ in. discarded before the material containing the phosphate goes through the washing process. The composition of a sample thrown

[2] *U. S. Bureau of Soils, Bulletin* No. 81, p. 9 (1912).
[3] *U. S. Bureau of Soils, Bulletin* No. 76, p. 14 (1911).
[4] *Loc. cit.*, pp. 21–22.

from the picking table at one of the plants was also determined." These two analyses are given in Table 1.

TABLE 1.—*Chemical Analyses of Discarded Material Connected with Preparation of Florida Pebble Phosphate*

| Description | Analysis | | | | |
|---|---|---|---|---|---|
| | $SiO_2$, Per Cent. | $Al_2O_3$, Per Cent. | $Fe_2O_3$, Per Cent. | $P_2O_5$, Per Cent. | $Ca_3(PO_4)_2$, Per Cent. |
| Clay balls, 2½ in. in diameter, containing phosphate pebbles from discard.......... | 44.10 | 5.24 | 1.78 | 17.60 | 38.54 |
| Material from picking table.. | 50.58 | 5.18 | 12.96 | 7.20 | 17.68 |

In the Mount Pleasant phosphate field at the present time, and probably in other parts of the Tennessee brown-rock phosphate areas, changes are being made that will result in leaving little or no wasted phosphate rock in the ground. Some phosphate is going into the waste ponds, but, without doubt, the time will come when all this material will be reworked, and even now some companies are working or are planning to work these old tailings. The modern mining and milling methods of the last decade are revolutionizing the industry and incidentally conserving this valuable fertilizer material. They are in striking contrast with the crude and wasteful methods formerly employed in the brown-rock field. Though the large operators are using up-to-date methods, even now some of the small operators are employing the old-fashioned hand methods which in the past resulted in the loss of much valuable rock.

The object, of course, in preparing phosphate rock for market is to remove as much of the clay, chert, and limestone as possible from it. Theoretically, it is possible to remove all these impurities, but this is not practicable, especially in the case of the clay. There is no sharp division between the finest phosphate sand and the clay, and it would obviously be wasteful to carry the process of obtaining the fine sand up to or beyond the point where the cost would offset the value of phosphate obtained. This is one of the practical considerations connected with the modern conservation of phosphate rock which perhaps has not always been given just and deserved consideration.

The point beyond which it is not practicable to carry the preparatory treatment is not fixed, and standards vary from time to time, and probably at a given time, among individuals and corporations. Thus, in the phosphate-mining industry as practiced in Tennessee in the early 90's, rock was discarded which has a high value today, and the former apparent lapses from the highest standards have in the course of time proven to be not lapses at all, but simply conditions imposed by the trade and the

times. In other words, the phosphate once discarded is now being utilized. The open-cut or surface method of mining brown rock, as practiced in the Tennessee field, with which the writer is more especially familiar, is peculiar in this respect, and the generalizations made do not cover any other classes of mining, and certainly will not apply to underground mining in general.

### *Grades of Brown Phosphate Rock*

Most of the rock from the Mount Pleasant field is shipped in three grades, namely, those containing 72, 75, and 78 per cent. of calcium phosphate. Five per cent. of iron oxide and alumina is the maximum allowed, and this is usually referred to as "I and A" in the trade and in commercial analyses. At one time only 78 per cent. rock was shipped from the Mount Pleasant field, and rock of this grade is still known as export rock. The guaranteed content in phosphate of lime, "bone phosphate," or BPL, as it is commonly referred to in the trade, next fell to 75 per cent., and at the present time many of the companies are finding it difficult to ship this grade exclusively, and the life of the 75 per cent. rock is limited. Every per cent. of iron oxide and alumina less than the 5 per cent. limit is regarded as equivalent to an additional 2 per cent. of calcium phosphate, for it is considered that in the subsequent treatment of the phosphate in the manufacture of fertilizers the harmful effect of 1 per cent. of iron oxide and alumina offsets the good effect of 2 per cent. of calcium phosphate. If there is more than 5 per cent. of iron oxide and alumina, the superphosphate becomes gummy and farmers find it difficult to drill it into the land.

### *Preparation of Phosphate Rock for Market, with Phases of Conservation Involved*

There are many stages to be considered under the heading of preparation of phosphate rock for market, but they may all be subdivided into three major operations as follows: (1) Removal of overburden, (2) mining, and (3) milling, in which is included drying. The present methods of utilizing and thus protecting from loss, or conserving, the brown phosphate-rock supplies, especially in the Mount Pleasant field, naturally are included under the above headings, and, therefore, will be described in connection with them so far as this can be done.

### *Removal of Overburden*

The overburden of the brown rock in the Mount Pleasant field varies from 1 ft. up. Usually it is less than 20 ft., but a thickness of 30 ft. is known, although that is excessive in places where mining is now in prog-

ress. The methods of removal of overburden are diverse. Under exceptional conditions, the old-time crude and expensive hand methods have to be resorted to, but in most places, and especially where virgin ground is being opened, operations are conducted in the most up-to-date fashion. Where the overburden is not very thick or hard, it may simply be plowed up and removed with scrapers, or it may be loosened with dynamite and then removed with scrapers. A favorite method of getting rid of the overburden, used especially in ground that is being reworked, is to first "hog" or undercut it, pry it off with bars, and then scrape or carry it away.

The drag-line excavator and the steam shovel are types of up-to-date machinery used in removing overburden in this field. The hydraulic method is also used. Both the overburden and the rock itself are removed by this last-named method, which is simplicity itself in action and which requires a minimum of labor in operation, usually one man to handle the hydraulic gun and two to keep the sluices clear. As practiced at the plant of the Blue Grass Phosphate Co., in the southern part of the Mount Pleasant field, the rock is mined out in small areas and the overburden from one area is washed into the mined-out cavity next to it. The resultant topography is level. The land is thus conserved for farming purposes for future generations and, indeed, greatly improved, for the phosphate sand and rock, formerly below the subsoil, is thoroughly incorporated in the soil and the fertilizing value of the phosphate thus rendered available in time.

### *Methods of Mining*

Much of the mining in the Mount Pleasant field has to be done by hand on account of the method of occurrence of the brown rock. The steam shovel has not proved successful because it cannot discriminate as to grade of rock mined, with the result that much clay and flint get into the product, and have to be subsequently removed. The cantilever adjunct to mining, which is employed at the plant of the Hoover and Mason Phosphate Co., is unique, there being only one in this field. The hydraulic method of mining is used at two plants and has many advantages, as pointed out under the preceding topic. These mechanical methods of mining and removing overburden, which have cheapened operating costs, have played the major part in conserving Tennessee brown rock.

*Reworking Deposits.*—The brown phosphate rock occurs in blanket form. From the base of this blanket, so-called deep "cutters" project like a network of roots, into the underlying limestone, the irregular projections of the latter, upward and into the phosphate rock, being known as "horses." In the early days of mining, all the phosphate rock occurring between the limestone "horses" was left, owing to the difficulty in mining it. Moreover, nearly everything that went through the tines of

a phosphate fork or a 2-in. screen was discarded. The latter material, for this reason, has come to be known as "screenings" or "throwbacks," and at certain plants this is now being worked. The screenings or throwbacks can be easily distinguished from the normally occurring plate or lump rock by its mixed-up or heterogeneous appearance, the lump rock being scattered irregularly throughout the mass. From the fact that the throwbacks represent material that has been worked over and from which the larger lump rock has been removed, it follows that the proportion of muck, sand, and clay is larger than in unworked territory.

On the Ruhm Phosphate Co.'s property, in the northeastern part of the Mount Pleasant field, the worked-over phosphate deposits are located on one of the terraces that surround the town.[5] This location at the top of a hill is such that the ore can be handled easily by gravity, and the hydraulic method of mining is therefore employed. The rock appears practically at the surface in quantity, and in addition to the muck left from early operations it contains much small lump rock, discarded in the days of early mining. The limestone horses often get in the way and have to be blasted out, but this is not difficult, owing to the loose or platy character of the phosphatic limestone associated with the brown-rock deposits. In addition to the hydraulic method, which can be employed only in certain favorable locations, hand mining is also employed. Where hand mining is practiced, the ore is usually screened on the spot where the miner is at work. The fine material passes through the screen and is saved and washed; and the coarse rock is hauled away and dried by burning on ricks of wood in the open, thus saving rehandling in the mill. The lump rock, as mined, usually contains from 20 to 21 per cent. of moisture, and drying it in this way reduces the moisture to 1 per cent. or less.

At the Century plant of the Federal Chemical Co., on the Williamsport Pike, northwest of Columbia, screenings are now being worked on an extensive scale.

The accompanying analyses show the content in phosphoric acid and phosphate of lime, of material left in earlier mining operations but now being worked at the plants mentioned above. It is certain that these analyses do not exactly represent the standard in phosphate content of a great deal of the material which is being reworked in this field. It is doubtful whether any hand sample can adequately represent the normal content in calcium phosphate of this material, owing to the difficulty of properly apportioning the lump rock and the muck. It is more than likely, however, that very low-grade material can and will be profitably worked, provided some of the cheaper methods of operating can be brought to bear upon it. There is also given in Table 2, an analysis of a sample of phosphate muck. This material is found in larger proportion

[5] The field observations were made in the Fall of 1914.

in the throwbacks than in the normally occurring rock, for the reason that in the original mining operations only the lump rock was saved. The writer has been told that the recovery of the phosphate in the muck handled is about 50 per cent. on the wet basis and about 40 per cent. on the dry, assuming that there is from 20 to 21 per cent. of moisture present. Unless the muck runs well up in calcium phosphate, it hardly pays to handle it.

TABLE 2.—*Analyses of Brown and Muck Phosphate from Reworked Deposits, Maury Co., Tennessee*

(W. C. Wheeler, Analyst)

| | $P_2O_5$, Per Cent. | $Ca_3(PO_4)_2$, Per Cent. |
|---|---|---|
| No. 2 | 21.16 | 46.24 |
| No. 27 | 31.88 | 69.66 |
| No. 37 | 20.13 | 43.98 |
| No. 38 | 27.31 | 59.69 |

No. 2: Phosphate muck from 2-ft. layer; Hoover & Mason Phosphate Co.

No. 27: Throwbacks or material reworked 5 ft. thick; Federal Chemical Co., Century plant.

No. 37: Material reworked. Average thickness, 4 ft. where sample was collected; Ruhm Phosphate Mining Co.

No. 38: Material reworked. Average thickness, 4 ft. where sample was collected; Ruhm Phosphate Mining Co.

*Working "Cutters."*—The phosphate rock in the "cutters," or deep places between the limestone horses, was left unmined in the early days of mining, as has been pointed out. The development of the cutters, which probably took place along original joint planes, varies greatly within the restricted Mount Pleasant field. In some places, notably south of Mount Pleasant, on the Hoover and Mason property, they are of large size. Some were observed 30 to 35 ft. wide and as much as 20 to 25 ft. deep, averaging probably 18 to 20 ft. They vary greatly in length. In these abnormally wide and deep cutters, it is not uncommon to have small limestone horses. A short distance away from the Hoover and Mason Co.'s property, some of the cutters are so narrow that the phosphate rock in them can be removed only with difficulty.

Hand methods of mining have to be employed almost exclusively to remove the phosphate rock from these cutters, owing to the peculiar method of its occurrence (Fig. 4). Hydraulic methods are also employed, as is the case in the Hickman County field where the depth of the cutters is also great. Owing to the depth of the cutters the work has to be done in benches of convenient height for the miners. The ore is picked out and shoveled from bench to bench, and finally into wagons in which it is hauled

to the mills. Mining the deep cutters is usually carried on in fair weather, or when the roads are good. In working over virgin ground at the present

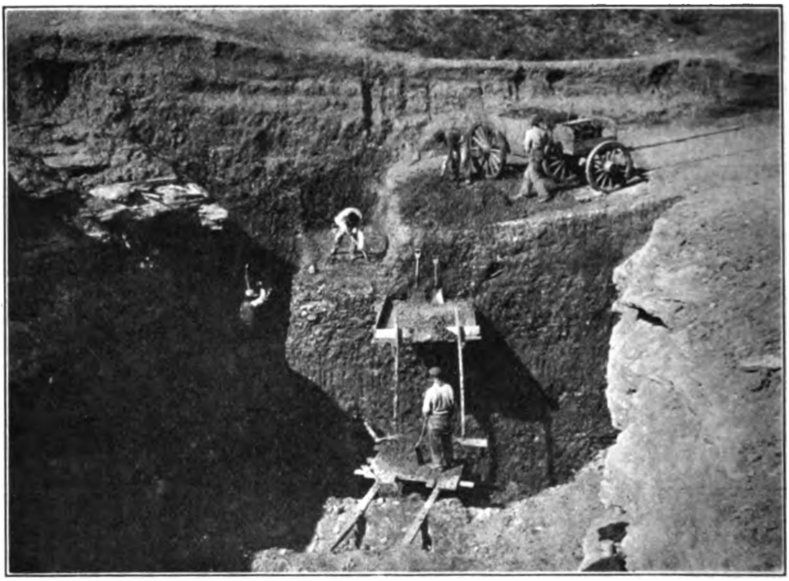

Fig. 4.—Method of Removing Phosphate Rock From a Deep "Cutter." Hoover and Mason Phosphate Co., Mt. Pleasant, Tenn.

Fig. 5.—Old Phosphate Workings Showing Phosphatic Limestone, at the Plant of the Blue Grass Phosphate Co., South of Mt. Pleasant, Tenn.

time, the rock in the cutters is readily and cheaply obtained by the cantilever method in use at the Hoover and Mason plant.

Territory formerly worked over is again being worked at the Arrow mine of the Charleston, S. C., Mining and Manufacturing Co. Most of this work is in rather shallow cutters. The material is picked out and screened either on the tines of a phosphate fork or on a small movable 1-in. mesh screen. The coarse rock is dried or burned on ricks of wood; the muck is washed at the company's mill. The old cutters containing phosphate rock are located by hand prospecting with a long, sharp, steel rod.

Fig. 6.—Outcropping Limestone Horses and Associated Phosphate Rock, Tennessee-Arkansas Phosphate Co., Scotts Mill, Maury County, Tenn.

## *Washing and Drying*

The washing processes whereby the mined rock is freed from clay, chert, and limestone are elaborate, and the mills in which the work is done are, for the most part, large and modern. These modern washing plants, which have done so much to make the mining of low-grade rock in this field profitable, and which, therefore, are playing such an important rôle in the conservation of phosphate rock in Tennessee, have practically all been installed during the last decade. The principles of the washing processes are the same in the different plants, but the details of manipulation differ. The phosphate rock, as mined, is brought to the washer either in wagons or by tram. Where hydraulic mining is practiced, it goes to the plant through a flume. The material mixed with water is delivered into a hopper at the top of the mill and the subsequent opera-

tions, for the most part, are conducted by gravity. From the hopper the rock passes through a toothed revolving crusher and then into log washers. From the washers, it passes to a cylindrical or conical screen with circular perforations. The coarse, or lump rock, which fails to pass through the screen, passes on to a picking belt where limestone and chert fragments and clay balls are removed. The material then goes to the wet-storage sheds or piles to be dried later. The fine material may go through a settler or clarifier provided with riffles, or through several settling tanks in succession, in which the sand settles out. The clay and sand not caught in the process go to the waste ponds. The above descriptions briefly outline the fundamentals of the washing process as carried on at most of the plants, but, of course, as has been mentioned, details are widely divergent.

The clay and the phosphate sand which pass to the waste ponds are of great interest in the problem of conservation. When the material reaches the waste pond, the coarse sand settles out first, and, naturally, nearest the end of the waste pipe or flume. This material is the highest in calcium phosphate. It is planned to work material of this character at one of the plants near Mount Pleasant, and already at another the old tailing dumps are being worked. At the latter plant much attention has been paid to the process of separating the clay and phosphate sand. There is a washer at this particular plant which differs from any other in the field, and is most thorough in its action. The clay resulting from the action of this washer was observed in the waste pond. It had been in suspension for a long period of time, and material taken and rubbed between the fingers appeared almost of impalpable fineness. Some of the phosphate sand from this washing process is so fine in texture that it sifts through the meshes of the sacks in which it is shipped.

The analyses of material from ponds abandoned years ago, but which are either being worked or which it is planned to work, are given in Table 3. It has been suggested that the material in these waste ponds might be used in its present form on Tennessee farms, but this has been found impracticable, as it will not bear the cost of transportation. The high phosphate content in certain of the samples collected is noteworthy.

Drying is accomplished in two very different ways, which are representative of the old and new methods employed in the Tennessee brown-phosphate field. At nearly all the large plants, modern rotating cylindrical driers, similar to rotary cement kilns, are in use, but the rock is fed both at the hot and cold ends. It would seem that the latter method would be the more efficient. There is generally some special cause when the old-fashioned method of drying on wood ricks is employed, and where it is in use it usually saves extra handling or haulage. Drying generally reduces the moisture present from 20 to 21 per cent. to 1 or 2 per cent.

TABLE 3.—*Analyses of Waste Material from Phosphate Washers in the Mount Pleasant, Tenn., Phosphate Field**

(W. C. Wheeler, Analyst)

| | $P_2O_5$, Per Cent. | $Ca_3(PO_4)_2$, Per Cent. |
|---|---|---|
| No. 13 | 31.79 | 69.47 |
| No. 20 | 27.20 | 59.44 |
| No. 25 | 29.91 | 65.34 |
| No. 32 | 20.38 | 44.53 |
| No. 36 | 14.36 | 31.37 |
| No. 42 | 30.63 | 66.92 |
| No. 47 | 20.71 | 45.26 |
| No. 50 | 28.59 | 62.47 |
| No. 54 | 26.50 | 57.90 |

No. 13: Sample of material from waste pond; International Agricultural Corporation, Frierson plant.

No. 20: Sample of material formerly discarded but now being reworked; Federal Chemical Co., Tennessee plant.

No. 25: Sample of material from waste pond; Tenn. Ark. Mining Co..

No. 32: Sample of material from waste pond, near outlet of waste pipe. It is planned to rework this; Federal Chemical Co., Century plant.

No. 36: Sample of material from waste pond; Ruhm Phosphate Mining Co.

No. 42: Sample of material from waste pond; International Agricultural Corporation, Jackson plant.

No. 47: Sample of material from waste pond; Blue Grass Phosphate Co.

No. 50: Sample of material from waste pond; Hoover and Mason Phosphate Co.

No. 54: Sample of material from waste pond; Charleston, S. C., Mining and Manufacturing Co.

### *Conservation of Fines*

In drying phosphate rock, much material in finely divided form has been lost by being carried out through the flue, owing to the powerful drafts employed, especially in the modern types of driers. At many of the plants, steps have been taken to save this material. This is accomplished by means of bends in the flue or by hoods or baffles. The following is an analysis of fine material caught in the chimney at the Century plant of the Federal Chemical Co. (W. C. Wheeler, Analyst):

No. 35.......... $P_2O_5$, 30.29 per cent.; $Ca_3(PO_4)_2$, 66.19 per cent.

## CHEMICAL METHODS OF CONSERVING PHOSPHATE ROCK[6]

### *Their Application in the West*

There is associated with all large important phosphate-rock deposits, considerable rock which is not up to the present commercial requirements

* Some of this material is now being worked and some of it will soon be worked.

[6] J. A. Barr: Tennessee Phosphate Practice, *Trans.*, vol. 50, pp. 917–933 (1915).

W. H. Waggaman: *U. S. Bureau of Soils, Bulletin* No. 76, pp. 15 and 16 (1911); also *U. S. Department of Agriculture, Bulletin* No. 144, pp. 25 and 26 (1914).

in content of calcium phosphate. There is also being produced, in connection with the preparation of commercial phosphate rock for market, a great deal of low-grade material. To bring these classes of material up to commercial grade, that is, to a grade containing 70 per cent. or more calcium phosphate, various chemical methods have been used. The time will undoubtedly come when these chemical methods will find much more extended application than at present, and when this time arrives it will result in conserving a great deal of phosphate rock now consigned to the waste ponds and dumps. Such methods are of more than ordinary interest in connection with the Western field, owing to the long distance that phosphate rock now has to be transported before reaching a market. The immense quantities of sulphuric acid potentially available in the immediate vicinity of the Western phosphate field, and which should become available in increasing quantity as time goes on, is another important element in the situation. Indeed, the chemical method of concentrating phosphate, and thus enabling it to be transported long distances, may well be worked out in connection with the high-grade rock that this field is able to produce, and it may also be the means of conserving the enormous amount of low-grade phosphate rock in the Florida, Tennessee, South Carolina, and other Eastern fields.

### *Chemistry of Process*

Phosphate rock is marketed now as such, and in the form of acid phosphate, including in the latter term ordinary super and double-acid phosphate, the latter containing two to three times as much soluble phosphoric acid as ordinary superphosphate.

Before the discovery of the extensive high-grade deposits of phosphate rock in this country and abroad, the manufacture of the concentrated grades of soluble phosphate was in fairly common practice. The large supplies of high-grade phosphate rock have rendered this unnecessary, though in France, Germany, and possibly other European countries, and in South Carolina, this practice is reported to be still in use.

The basic reaction involved in the preparation of soluble acid phosphate takes place when ordinary rock phosphate $Ca_3(PO_4)_2$ is treated with sulphuric acid. In simple form, the reaction that takes place may be represented thus:

(1) $Ca_3(PO_4)_2 + 3H_2SO_4 = 2H_3PO_4 + 3CaSO_4$.
(2) $4H_3PO_4 + Ca_3(PO_4)_2 = 3CaH_4(PO_4)_2$

Reduced to one equation, this is as follows:

$$Ca_3(PO_4)_2 + 2H_2SO_4 = CaH_4(PO_4)_2 + 2CaSO_4.$$

In the presence of water, which has been omitted from the above equations in order to simplify them, the calcium sulphate would be changed

into gypsum by abstracting water from the mass. The last reaction is the one desired by the manufacturers.

To utilize low-grade rock and tailings, and to make concentrated phosphatic fertilizers, the phosphoric acid produced by the first reaction is evaporated in pans until it contains about 45 per cent. phosphoric anhydride. It is then treated with a fresh supply of phosphate rock, when the following reaction ensues:

$$Ca_3(PO_4)_2 + 4H_3PO_4 = 3CaH_4(PO_4)_2.$$

It will be observed, therefore, that ordinary superphosphate is largely a mixture of soluble calcium phosphate and gypsum, while the double-acid phosphate contains little or no calcium sulphate, or dehydrater, and thus has to be artificially dried. Either the phosphoric acid itself, the double phosphate, or such compounds as ammonium phosphate, might be shipped from our Western field, since they are highly concentrated products.

## SUBSTITUTES FOR PHOSPHATE ROCK

The use of substitutes for ordinary phosphate rock has been in the past of great importance, but, since the discovery of large deposits of high-grade phosphate rock, the price of phosphatic fertilizers has greatly decreased, as a result of which the more highly priced guanos and similar materials have been driven out of the market. The use of substitutes, however, still continues. There is still much low- and medium-grade material which could, if necessary, actually take the place of phosphate rock as a source of phosphorus and which may possibly be used at some future time as commercial conditions change.

In the following pages, the different substitutes for phosphate rock that have suggested themselves are named and briefly described. They may be classified in two groups: (1) The natural and (2) the artificial substitutes for phosphate rock. Under the natural substitutes come (*a*) phosphatic limestone; (*b*) other phosphate-bearing minerals, such as apatite, nelsonite, wavellite, and others; (*c*) guano; (*d*) marl; (*e*) excrement, both human and animal; (*f*) bones. Within the class of artificial substitutes may be included (*a*) the basic slags and (*b*) manufactured compounds, like ammonium phosphate; the double-acid phosphate, phosphoric acid, and ordinary superphosphate from the low-grade phosphate rock discarded at phosphate mines.

### The Natural Substitutes for Phosphate Rock

#### *Phosphatic Limestone*

Directly below the phosphate-rock horizon in the Mount Pleasant and other parts of the Tennessee field, occurs the phosphatic limestone from which the brown rock itself has been derived. There must be an

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enormous tonnage of this phosphatic limestone scattered throughout the phosphate-rock areas of middle Tennessee. A long period of time must elapse before any attention will be given to this comparatively low-grade material as a source of phosphate, but it would be hazardous to say that this will never be done. Of course, in the mined-over area, much of the richer limestone has been covered so deeply that it will be difficult and expensive to get at. Analyses of this limestone, some of which was in a leached and some in a partially leached condition, occurring in horses between cutters, show calcium phosphate ranging from 4 to more than 42 per cent. The carbonate and the phosphate of lime mixture in this material has considerable value as fertilizer when applied directly to the land in finely pulverized form, and, although it is difficult to predict how and when this material will be utilized, it seems fairly certain that it will prove of value at some future time.

Under present conditions of mining, the phosphate is dug from around the limestone horses and boulders, and the pits are then either abandoned or filled with material from the overburden. The time has not arrived for the utilization of such material, but, when it does, it will be removed, broken up, crushed, and spread on the land, either with or without treatment with acid, or, as suggested by Waggaman,[7] the phosphatic limestone may be burned in a kiln and then slaked with steam or hot water and the rock thus disintegrated. In experiments made by heating the phosphatic limestones, after their phosphate content had been determined, it was found that the percentage of the latter was increased in quantity, the increases ranging from 1.40 to 8.62 per cent.[8]

### *Apatite, Nelsonite, and Other Minerals Containing Phosphate*

The mineral apatite is among the most definite in composition, if not the most definite, of the crystalline phosphate-bearing minerals. It is widely distributed and occurs in rocks of various kinds, but most commonly in those of the metamorphic and crystalline types, such as crystalline limestone, dolomite, gneiss, the mica schists, etc. The two common varieties are fluor-apatite, $3Ca_3(PO_4)_2.CaF_2$, and chlor-apatite, $3Ca_3(PO_4)_2.CaCl_2$, with intermediate compounds containing both chlorine and fluorine. The normal varieties contain in the case of fluorine 42.3 per cent. phosphoric acid ($P_2O_5$), and the chlor-apatite 41 per cent. of phosphoric acid ($P_2O_5$). The fluor-apatite is much the more common variety and here belongs the apatite found in St. Lawrence County, N. Y., Canada,[9] Spain, and the Alps. The Norway apatite is of the chlor-

[7] *U. S. Bureau of Soils, Bulletin* No. 81, p. 13 (1912).

[8] *Loc. cit.*

[9] R. W. Ells: *Geological Survey of Canada, Mineral Resources of Canada, Bulletin on Apatite* (1904).

For a complete list of minerals containing at least 1 per cent. of phosphoric acid, see W. B. Phillips: *Trans.*, vol. 21, pp. 188–196 (1892–93).

ine-bearing type. The apatite associated with the magnetite at Mineville, northern New York, on the property of Witherbee, Sherman & Co., is worthy of special mention. Attempts have been made, according to report, to separate the apatite on a commercial scale, but they have not proved successful.

*Apatite in Virginia.*—Rock of igneous origin, rich in titanium and phosphorus, occurs in the eastern foothills of the Blue Ridge in Virginia, near Roseland, about 7 miles northwest of Arrington on the Southern Railroad, and 24 miles northeast of Lynchburg. The titanium occurs in the form of ilmenite, and the phosphorus in the form of apatite, these being the dominant minerals. The rock occurs in dikelike forms of varying size and irregular shape, and to it the name nelsonite has been applied.[10]

Many varieties of rock have been included under this name, for example, ilmenite nelsonite, magnetite nelsonite, biotite nelsonite, hornblende nelsonite, and rutile nelsonite, from the dominant mineral that may be present, and gabbro nelsonite, from the rock-type gabbro, which some of the facies of nelsonite resemble. Ilmenite nelsonite, to which the name was first applied, is the normal and most abundantly occurring variety. The apatite present is the fluor-apatite, chlorine being present only in traces. The content of the nelsonite in apatite ranges from 0.3 to 30 per cent.

Experiments having as their object the commercial utilization of nelsonite have been carried on at the Bureau of Soils. Both the minerals ilmenite and apatite contain elements which seriously affect each other so far as commercial applicability is concerned. The problem, then, first, is to separate them. Two mechanical methods have been tried, depending (1) on differences in specific gravity, (2) on the magnetic properties of the ilmenite. Neither of these methods was found entirely satisfactory. The chemical method was then resorted to. In the experiments performed by W. H. Fry, of the Bureau of Soils, it has been found that ilmenite may be almost completely freed from apatite by means of sulphuric acid with a minimum of waste and without involving great expense. Moreover, all the products obtained can be utilized commercially.

Preliminary experiments showed that ilmenite is entirely unattacked by dilute sulphuric acid, while apatite is acted on quite energetically by this reagent. Subsequently, it was shown that the apatite remaining in ilmenite after mechanical separation can nearly all be extracted by means of dilute sulphuric acid without appreciably affecting the ilmenite. The

[10] T. L. Watson: *Mineral Resources of Virginia*, p. 300 (1907); Geology of the Titanium and Apatite Deposits of Virginia, *Virginia Geological Survey, Bulletin* 3A, pp. 100 *et seq.* (1913); T. L. Watson: and Stephen Taber: Virginia Rutile Deposits, *U. S. Geological Survey, Bulletin* 430, p. 206 (1910).

details connected with the experiments are outlined in an article by Waggaman.[11]

Thus the possibility is shown of obtaining phosphate fertilizer material from this occurrence.

The great objection to apatite as a source of phosphate is the expense of mining and preparing the rock for treatment with acid. In the case of the apatite containing fluorine, hydrofluoric acid gas and possibly other poisonous gases are given off when the mineral is acted upon by sulphuric acid. Thus, the treatment of this mineral with acid may be attended with danger, unless care is exercised in the manipulation, or unless there is enough of silica or silicates present to react with the liberated hydrofluoric acid. The cheap and accessible sources of phosphate have caused a practical stoppage of the mining of apatite for fertilizer purposes, though at one time a considerable amount of the mineral was so used.[12]

### *Guano*

Guano consists chiefly of the excrement of birds and bats, and has been found in considerable quantity in some places. It has been extensively used in the manufacture of acid phosphate. There are two types: (1) The unleached deposits found in caves and sheltered places, which contain phosphoric acid in readily available form, and also nitrogen, a fertilizer of great importance and high price. (2) The leached deposits, which contain no nitrogen and in which the phosphoric acid content, though high, is generally insoluble. Accessible and valuable deposits of guano are now rather scarce, and only those having the best transportation facilities are being worked.

### *Greensand (Marls)*

Phosphate is found to a small extent in certain greensands of the eastern and southeastern States, notably in New Jersey, Kentucky, Tennessee, South Carolina, Florida, Alabama, and without doubt in many other coastal-plain States of the East.

*New Jersey.*—About 30 years ago, the greensands of certain portions of New Jersey were in great demand. On the first geologic map of that State, the location of the beds containing them was shown, and in some of the earlier reports the deposits were described and numerous analyses given, as well as instructions for their use. In recent years, however, greensand has been supplanted to a large extent by the more highly concentrated artificial fertilizers and is no longer dug extensively.

[11] W. H. Waggaman: A Possible Commercial Utilization of Nelsonite, *Journal of Industrial and Engineering Chemistry*, vol. 5, No. 9, pp. 730–732 (1913).

[12] Ells: *Loc. cit.*

The analyses in Tables 4 and 5 show the composition of the different grades of greensand as dug and applied to the soil. The glauconite in them is of nearly uniform composition, but mixed with it are carbonate, sulphate, and phosphate of lime, quartz sand, sulphide and phosphate of iron, shells, etc. The differences in the kind and quantity of these substances cause wide differences in the appearance of the greensand containing them, as well as in its composition and properties.

Table 4 gives the phosphoric-acid content, in percentages, of typical specimens of New Jersey greensand:

TABLE 4.—*Phosphoric Acid in Typical Greensand (Marl) of New Jersey*

| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
|---|---|---|---|---|---|---|---|---|---|---|
| Phosphoric acid...... | 1.14 | 1.33 | 1.02 | 2.24 | 2.69 | 2.56 | 3.58 | 3.87 | 2.58 | 2.30 |

New Jersey greensand[13] has been of incalculable value to the region in which it is found. It has raised this region from the lowest stage of agricultural exhaustion to a high state of improvement. Found in places where no capital and but little labor were needed to get it, the poorest people have been able to avail themselves of its benefits. Lands which in the old style of cultivation had to lie fallow, by the use of marl produce heavy crops of clover and grow rich while resting. Land which had been worn out, and left in commons, is now, by the use of this fertilizer, yielding large crops of the finest quality. Everywhere in the marl district, may be seen farms which in former years would not support a family but which are now making their owners rich through their productiveness. Bare sands, by the application of marl, are made to grow clover and then crops of corn, potatoes, and wheat. "Pine barrens," by the use of marl, are made into fruitful land. The price of land in the greensand-marl belt of New Jersey was considerably below that in the northern part of the State 40 years ago; now the price is higher.

*Kentucky*.—The greensand (marls) of the Leitchfield, Kentucky, region have been described by N. S. Shaler[14] and have been analyzed by Robert Peter.[15] A sample from near Leitchfield, Grayson County, was sent to the United States Geological Survey by M. H. Crump, of Bowling Green, Ky. It was analyzed in the Survey laboratory by Chase Palmer and found to contain 3.73 per cent. potash ($K_2O$) and 0.13 per cent. phosphoric acid ($P_2O_5$).

Greensands containing approximately these quantities of potash and phosphoric acid are found in large quantities over a considerable area in west central Kentucky, and in the reports cited above their use was sug-

[13] *Annual Report of the State Geologist of New Jersey*, 1886, p. 154.

[14] *Kentucky Geological, Survey Report, new ser.*, vol. 3, pp. 46–47 (1877).

[15] *Kentucky Geological Survey, Chemical Analyses*, vol. *A*, pp. 250–254 (1884).

gested to rejuvenate the lands of the State, worn out as a result of the excessive cultivation of tobacco and other crops.

*Tennessee.*—The greensands of Tennessee were mentioned by G. Troost as early as 1835. They are found in Hardin, McNairy, and Henderson Counties. The analyses by Troost shown in Table 5 are of the greensands of McNairy County:

TABLE 5.—*Analyses of Greensand, McNairy County, Tenn.**

| | 1 | 2 | 3 |
|---|---|---|---|
| Silica ($SiO_2$) | 48.00 | 45.30 | 51.70 |
| Alumina ($Al_2O_3$) | 7.00 | 6.20 | 6.50 |
| Ferrous oxide ($FeO$) | 20.70 | 18.00 | 21.20 |
| Potassa ($K_2O$) | 10.10 | 10.40 | 11.30 |
| Carbonate of line ($CaCO_3$) | 5.70 | 10.80 | 2.00 |
| Water ($H_2O$) | 8.00 | 8.50 | 7.30 |
| Loss | 0.50 | 0.80 | 0.00 |
| Total | 100.00 | 100.00 | 100.00 |

Phosphoric acid, which is doubtless present in the marls, does not appear to have been separated.

*Alabama.*—Materials of several kinds, containing phosphoric acid, have been found in Alabama, in both the Cretaceous and Tertiary formations. In several places in the lower part of Marengo County, notably near Dayton and Nixonville, there occur tolerably compact beds of shell-casts, containing from 20 to 25 per cent. phosphoric acid. According to E. A. Smith,[16] these beds are the most promising sources of phosphate in the State. Such high-grade phosphate beds should have future value as fertilizers.

### *Excrement*

A large part of the phosphorus sent from the country in the form of cattle and grains for consumption in the cities, finds its way ultimately, via the modern sewage system, to the ocean. There it is lost so far as its fertilizer value to the land is concerned. Whitson[17] estimates that the loss in the cities due to human excreta alone is equivalent to 2 or 3 lb. of phosphoric oxide per acre for the entire cropped region of the United States. Supposing this loss to be 2 lb., 1/1,000 ton, this amounts, for

* J. M. Safford: *Tennessee Geological Survey*, p. 515. Nashville, 1869.

[16] E. A. Smith: The Phosphates and Marls of Alabama, *Trans.*, vol. 25, p. 811 (1895).

[17] A. R. Whitson and C. W. Stoddart: The Conservation of Phosphates on Wisconsin Farms, *University of Wisconsin Agricultural Experiment Station Bulletin*, No. 174, pp. 1–8 (April, 1909).

400,000,000 acres of cropped land, to 400,000 tons of phosphoric oxide—equivalent to 1,200,000 tons of phosphate rock.[18] Surely here is a problem for the municipal and chemical engineer of the future.

### *Bones*

Before phosphate rock was found in quantity in the United States and in other parts of the world, bones were the main source of phosphoric acid for fertilizing purposes. There is still a domestic production of this form of phosphate fertilizer. The bones were steamed, charred or burned, and applied to the land after such treatment, or they were made into superphosphate in the way ordinary bone phosphate is now converted, by treatment with sulphuric acid. Ground bone, or bone meal, is a valuable fertilizer without any treatment whatever, due to its nitrogen and phosphorus. The phosphoric acid contained in bones has considerable value as a source of phosphorus for chemical purposes other than fertilizer.

## Artificial Substitutes for Phosphate Rock

### *Basic Slag*[19]

Basic slag is not soluble to any extent in water, but it has been shown that the phosphate contained in it is readily assimilated by growing crops and splendid results have followed from its use as a fertilizer.

It is produced in the manufacture of steel by the basic Bessemer and basic open-hearth processes. In these the phosphorus is separated from the iron by the addition of lime or limestone, which, having a strong chemical affinity for the phosphorus, forms with it basic phosphoric compounds which enter the slag. In slags produced by these processes, the phosphoric acid may run from 10 to more than 25 per cent.

The iron ores used in this country, with the exception of some mined in Alabama, are low in phosphorus and consequently the basic Bessemer process is not applicable to them. The slags produced from such low-phosphorus ores are consequently low in this fertilizing element, but abroad, and especially in Germany, where the iron ores are high in phosphorus, the basic Bessemer process is widely used and as a result the slag, rich in phosphate, is widely applied as a fertilizer. There are indications that the importance of these high-phosphatic slags is coming to be realized in this country.

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[18] C. R. Van Hise: *Conservation of Natural Resources in the United States*, p. 325. The MacMillan Co., New York (1910).

[19] E. C. Eckel: Utilization of Iron and Steel Slags, *U. S. Geological Survey, Bulletin No.* 213, pp. 225–227 (1903).

W. H. Waggaman: Utilization of Acid and Basic Slags in the Manufacture of Fertilizers, *U. S. Bureau of Soils, Bulletin No.* 95 (1913).

In rock phosphate the phosphoric acid is combined with lime as tri-calcium phosphate with the symbol $Ca_3(PO_4)_2$. In slags, however, the combination is that of tetra-calcium phosphate ($4CaO.P_2O_5$ or $Ca_4P_2O_9$), according to certain investigators. Other experimenters have found that different compounds have formed with variations in the composition of the molten bath and that toward the end of the operation, when much silica was present, a compound having the formula $P_2O_5.SiO_2.5CaO$ was formed. It has also been shown that in slags having the same phosphate content, those containing silica were the most soluble and hence the most valuable for fertilizer purposes.

## PHOSPHATE ROCK RESERVES

### FLORIDA[20]

#### *Hard Rock*

Several maps of the Florida phosphate fields have been published, but they are of a general nature and attempt to show only the approximate location of boundaries. Even this is difficult to do with any degree of accuracy without careful prospecting. In estimating the available hard rock, some idea of the total area in the State within which such deposits may occur is essential. The estimates made agree fairly closely, and indicate that the workable beds of hard phosphate rock in Florida occur throughout an area of several hundred square miles. The area actually underlain by workable deposits of hard rock is, however, but a small fraction of that within which the deposits have been mapped. There are, moreover, no existing data from which one may calculate the area underlain by workable deposits with a degree of accuracy that would have any practical value whatever.

Sections of square miles could be taken within which the deposits have been most completely mined out and tonnage estimates made from them, but figures thus obtained could not be used in other areas as a standard, since over many square miles there are no deposits at all. The deposits in the sections which have been most completely mined out, moreover, are usually the most accessible. With due allowances, it is conservatively estimated that there is as much hard rock available in Florida as has already been removed, that is, approximately 10,000,000 tons, and with an annual output of 500,000 tons the hard-rock phosphate deposits may be expected to last at least 20 years longer.

#### *Land Pebble*

The land-pebble beds are more regular in their occurrence than the hard-rock deposits, but close estimates cannot be made except by actual

[20] The writer is greatly indebted to Dr. E. H. Sellards, State Geologist of Florida, for valuable suggestions in preparing this note on the Florida phosphate-rock reserves.

prospecting; and this will be done only gradually by those who are interested in or are engaged in mining. The land-pebble phosphate belt is approximately 30 miles long by 5 to 10 miles wide. On the basis of a conservative estimate of acreage and of tonnage per acre, the writer has calculated a total of 190,000,000 tons of land pebble.

The output of land pebble per year in Florida is, in round numbers, 2,000,000 tons. The estimate of available land pebble, which is considered extremely conservative, leads to the conclusion that this type of phosphate rock in Florida will last several generations, and for present purposes it may be considered practically inexhaustible. The refinements in methods of mining land pebble are gradually reducing the quantity of small pebbles that go to the waste dump, and this factor will tend to prolong the life of these deposits beyond that calculated from the figures given above.

In making up the estimates for Florida, river pebble has not been included, owing to the difficulties connected with estimating its quantity. This factor also adds to the conservatism of the figures given for this State.

### *Waste Material*

The phosphoric acid in the Florida deposits in the form of soft phosphate, so-called, together with large quantities of aluminum and iron phosphate, go to the dumps in the preparation of the hard and pebble rock for market. The loss calculated in terms of phosphate of lime is considerable and it may possibly equal the actual quantity saved and marketed. It has been calculated by W. H. Waggaman[21] that the marketed material is probably not more than 15 per cent. of the total material mined, and that in the discarded material is an average of at least 10 per cent. phosphoric acid. The total quantity of Florida phosphate rock marketed up to and including 1914 is approximately 27,500,000 long tons. On these bases, the low-grade material in the waste heaps is approximately equivalent to 27,000,000 to 30,000,000 tons of high-grade material.

## Tennessee

### *Introductory Note*

There are three varieties of phosphate rock in Tennessee, the white, the blue, and the brown. The very irregular character of the white phosphate-rock deposits, and the fact that they have been but meagerly prospected, make it impossible to estimate the available supplies even approximately. The omission of estimates of white rock may be considered to contribute to the conservatism of the total figure for the State.

[21] W. H. Waggaman: *U. S. Bureau of Soils, Bulletin No.* 76, p. 14 (1911).

### *Blue Rock*

The blue bedded phosphate-rock deposits are found chiefly in Perry, Hickman, Lewis, and Maury Counties. In estimating them, only rock 24 in. and over in thickness will be considered. The area underlain by rock of this thickness is large, but the area in which the rock is thinner is much greater. This estimate includes the rock under heavy cover, but this is not a bar to its availability, since blue rock is worked wholly underground.

As stated in another part of this paper, blue phosphate rock, even in those localities where thickest, is extremely variable and may locally be absent altogether. As an offset to this factor, however, a very large area has been omitted in the calculations in which the rock has been considered to be less than 24 in. in thickness, but which future prospecting may show to be thicker. It may reasonably be expected that outlying areas in which the rock is locally workable will be found in the future. It must also be considered that rock only a foot thick, but of high-grade, may be considered of future workability. It is also possible that all of the blue rock estimated may not be up to present commercial requirements. As time goes on, however, the standard in this respect will tend to fall, and ore may then be treated chemically to raise the grade for shipment to distant points.

Taking all the above factors into consideration, it is estimated that there are approximately 84,000,000 tons of available blue rock phosphate in Tennessee.

### *Brown Rock*

The brown phosphate rock included in the estimate given below occurs chiefly at the Bigby horizon. It is found chiefly in Maury, Giles, Hickman, Lewis, and Sumner Counties, Tennessee. The Mount Pleasant, Maury County, district contains the most extensive deposits, and the major operations on brown rock phosphate in the State are located there. The quantity of brown phosphate rock in the Mount Pleasant district, which may be taken as a type, ranges from 600 to 1,000 tons per acre-foot; and 850 tons is considered a fair average. A considerable portion of the areas where are located important deposits of brown rock has been worked over in Tennessee, especially in the Mount Pleasant field. This fact, and the additional fact that the entire acreage can not be underlain by phosphate rock, would tend to reduce the estimate; but, on the other hand, it is very reasonable to suppose that as detailed prospecting takes place new and important areas, though small, will be found. Two such areas have recently been brought to the writer's attention. Taking these factors into consideration, it is estimated that an available tonnage of approximately 4,000,000 tons of brown rock

remains in middle Tennessee, or approximately about as much as has already been marketed.[22]

In addition to the virgin rock remaining unmined in Tennessee, a great deal of phosphate rock will be recovered from waste ponds and from the dumps from old workings. Such work is in progress, as described earlier in this paper. The recovery of this material will tend to greatly prolong the life of the brown phosphate-rock field beyond the period indicated by a theoretical calculation based on the estimate of reserves made above and the yearly marketed output of brown rock in Tennessee.

## South Carolina

Several estimates have been made of the quantity of phosphate rock available in South Carolina for future use. In the very early days of the industry, the data for such estimates were lacking, but increased knowledge gained from working the deposits has lessened this difficulty to some extent. Shepard, in 1880, estimated the total available supply to be less than 5,000,000 tons, but from 1881 to 1914, inclusive, the quantity of rock removed was 11,600,000 tons out of a total of 13,000,000 tons for the State since the industry began in 1867.

P. E. Chazal[23] in 1904 estimated the quantity of rock remaining in the land deposits at between 9,000,000 and 11,000,000 tons. Since 1904, there have been removed from the South Carolina deposits more than 2,000,000 tons, leaving still available between 7,000,000 and 9,000,000 tons of phosphate rock. These figures do not include the river rock. The higher figure, namely, 9,000,000 tons, may therefore be taken as a conservative estimate of the available supply in South Carolina.

The annual production of South Carolina phosphate rock is approximately 100,000 tons. At the present rate of production, the phosphate rock supply of South Carolina would last approximately 90 years. In view of the fact that the production of this State has been steadily decreasing for many years, and the possibility that other deposits may be found, it seems safe to conclude that the South Carolina deposits can produce rock of somewhat low grade for many years to come and should become an important source of supply when the chemical concentrating methods come into general practice.

It is interesting in this connection to note that W. H. Waggaman[24]

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[22] James A. Barr: (*Bulletin* No. 110, p. 244, Feb., 1916) states: "In the event of low grades of phosphate becoming commercial products, 20,000,000 tons would become available within a 50-mile radius of Mount Pleasant alone, and perhaps 100,000,000 tons of rock, especially if phosphatic limestone is taken into account."

[23] P. E. Chazal: *A Sketch of the South Carolina Phosphate Industry*, pp. 17 and 18 (1904).

[24] W. H. Waggaman: *Journal of Industrial and Engineering Chemistry*, vol. 6, No. 6, p. 464 (June, 1914).

has estimated a somewhat higher available supply of phosphate rock of high grade in South Carolina, namely, 10,000,000 tons. F. B. Van Horn's[25] estimate made in 1909 places the available South Carolina phosphate rock at 3,000,000 tons, but that writer qualifies the estimate by observing that careful and deep prospecting may increase these figures.

## Kentucky

The rock occurring in this field is of the brown-rock type, similar to the brown phosphate-rock deposits occurring in middle Tennessee. The areas underlain by high-grade deposits are scattered, but the most important areas occur to the south and northwest of Midway, Woodford County. The writer has explored this region with the drill, though not closely enough to give close figures. A very conservative estimate of the phosphate rock available in this region is 1,000,000 tons.

## Arkansas

Though there is only a small production of phosphate rock in Arkansas at the present time, it must not be inferred that the deposits in this State will not prove of future value. In many of the analyses published by Branner and Newsom,[26] the content of the iron oxide and alumina runs high, and although this factor tends to unfit the Arkansas rock for the manufacture of superphosphates under present conditions, the time without doubt will come when these deposits will be extensively exploited for direct application to the soil. Chemical treatment should also render a very large part of the Arkansas phosphate available for future use. The tonnages per acre calculated for small tracts in which conditions are well known, show a large quantity of phosphate rock.[27] Calculations of tonnages indicate as much as 11,600 tons in one locality where the thickness of the rock was approximately 4½ ft. In another place, the acre tonnage was 23,200 where the average thickness was considered to be 8 ft. The latter figure is reckoned on the basis of 2,900 tons per acre, which, in turn, is based on a weight of 150 lb. per cubic foot of rock. Waggaman[28] estimates 20,000,000 tons of high-grade rock in Arkansas.

## The Western States

The available tonnage estimates for the Western field, which are only partial, are chiefly for Idaho, but include figures for small parts of Utah,

[25] F. B. Van Horn: Phosphate Deposits of the United States, *U. S. Geological Survey, Bulletin No.* 394, p. 164 (1909).

[26] *Arkansas Agricultural Experiment Station Bulletin, No.* 74, pp. 116–119 (1902).

[27] *Loc. cit.*, pp. 71–80.

[28] W. H. Waggaman: *Journal of Industrial and Engineering Chemistry*, vol. 6, No. 6, p. 464 (June, 1914).

Wyoming, and Montana. The estimates prepared, which are considered conservative, are all for high-grade rock, 65 per cent. or more tricalcium phosphate, and refer chiefly to the main bed which ordinarily lies near the base of the phosphate shales and in the Idaho field is usually 5 or 6 ft. thick. Thus they do not include the thinner beds of high-grade rock nor the great body of low-grade material. The main bed itself is included only for those parts of the field in which, according to the present practice of the Geological Survey, the rock lies at depths considered workable. The estimates are based on the assumption that the phosphate rock at depth, and remote from the outcrop, maintains the generally uniform qualities displayed at the surface in so many parts of the field. Some tendency toward enrichment by weathering has indeed been noted, but observations thus far obtainable suggest no marked decrease in richness within the body of the material.

Under the above conditions the areas examined in the field work of 1909 to 1913, inclusive, in Utah, Wyoming, and Idaho, are estimated to contain at least 5,290,296,900 long tons of high-grade phosphate rock. In addition, the Elliston field in Montana is estimated to contain 86,000,000 short tons,[29] or approximately 76,785,700 long tons, a total available estimate of 5,367,082,600 long tons. Since 1913, additional areas in some of these regions have been examined in detail and the presence of still more of the high-grade rock has been determined. There yet remains a considerable area of withdrawn land that has not been examined, in which it is probable that high-grade phosphate will be found. When the results of these examinations and the results of more detailed work in regions now known only by reconnaissance are added to the figures given, it is probable that the estimates of high-grade rock in the Western field will be considerably increased.

## Total Tonnage Available

In Table 6 is given the estimated tonnage of phosphate rock available in the United States at the present time.

Table 6.—*Phosphate Rock Available in the United States*

| Eastern States | Long Tons | Western States | Long Tons |
|---|---|---|---|
| Florida | 227,000,000 | Montana, Idaho, Utah, and Wyoming | 5,367,082,600 |
| Tennessee | 88,000,000 | | |
| South Carolina | 9,000,000 | | |
| Kentucky | 1,000,000 | | |
| Arkansas | 20,000,000 | | |
| | 345,000,000 | Total | 5,712,082,600 |

[29] R. W. Stone and C. A. Bonine: The Elliston Phosphate Field, Montana: *U. S. Geological Survey, Bulletin No.* 580, pp. 382–383 (1915).

The United States is now producing for domestic use and export about 3,000,000 tons annually.[30] More than 99 per cent. of this comes from the Eastern States, and in 1914 nearly 80 per cent. came from Florida. On this basis, Eastern phosphates should last fully 100 years, taking into account material of good grade.

## FOREIGN PHOSPHATE DEPOSITS

Important deposits of phosphate rock are located outside the United States. Perhaps the best known of these foreign deposits are those in the South Sea Islands, Christmas Island in the Indian Ocean, the African deposits, and those on Curaçao in the Dutch West Indies. In Africa, the principal deposits are located in Egypt, Tunis, and Algeria. In the South Pacific, Angaur of the Pellew group of islands, Nawoda (Pleasant), Panapa (Ocean), and Makatea (Aurora) islands, all contain important deposits. A deposit has recently been reported in northern Chile.

### African Deposits

#### *Egypt*[31]

The development of the extensive deposits of phosphate near the Red Sea has, during the past 2 years, assumed important proportions. The mines are worked by a British concern and are connected by rail with the Red Sea, where the rock is loaded on steamers for export. The rock contains 65 per cent. or more of tricalcic phosphate.

A company managed by Italians and founded in 1912, has obtained extensive concessions about 12 miles inland from Kosseir and also at Sebaia, on the eastern bank of the Nile, between Keneh and Assouan. The former concession is being connected with the port of Kosseir by a light railway, which should shortly be completed. The rock from the latter mines will be transported down the Nile by a ropeway and thence to Alexandria by boat for shipment.

The total output of phosphate in Egypt for the years 1908 and 1912 was as follows: 1908, 700 tons; 1909, 1,000 tons; 1910, 2,397 tons; 1911, 11,925 tons; and 1912, 69,985 tons. According to the *Financial Adviser's* report, the output during 1913 exceeded that for 1912 by about 33,000 tons.

Although other beds of phosphate are found in various districts in Egypt on both sides of the Nile Valley, the Red Sea area is responsible for almost the whole output. The rapid development of the business of the Egyptian Phosphate Co., the British concern, and the impending com-

[30] Normal conditions are referred to.

[31] U. S. Consul-General Olney Arnold: *U. S. Daily Consular and Trade Reports*, p. 991 (Aug. 20, 1914).

mencement of active operations by the other company, will lead to a considerable increase in production.

Practically all the raw phosphate produced is shipped from Egypt, principally to Japan. That country in 1912 took about 49,000 tons out of a total export of 52,000 tons, and in 1913, 59,000 tons out of a total of 64,000. According to the customs' returns, the average value of the raw phosphate shipped during 1913 at Port Safaga was about £1 ($4.87) per ton.

A foreign correspondent of the Survey gives the following interesting information with reference to the extent and continuity of the North African deposits:

They correspond geologically to the deposits of phosphate rock in Tunis and Algeria, and there can be little doubt that the phosphate-bearing formation extends from Algeria to the Red Sea, a distance of about 2,000 miles. The intervening territory, comprised in Tripoli and the Lybian Desert, has not been explored. There is also evidence that the same phosphate-bearing formation extends beyond the Red Sea and through the Arabian Desert into Persia, a farther distance of at least 1,000 miles, and there is a possibility that it extends much farther.

### *Algeria and Tunis*[32]

The deposits of phosphate rock in Algeria are continuations of the deposits in Tunis. The two important mining districts in Algeria are located near the towns of Setif and Tebessa, in the eastern part of the State. They have been developed during the last 15 years, the production having increased from 1,057 short tons in 1899 to 550,000 in 1912. The percentage of lime phosphate in the rock exported from the Setif district ranges from 58 to 63 per cent., and that in the rock from the Tebessa district, from 58 to 68 per cent.

*Tebessa District.*—The deposits of Kouif, the most important exploited in Algeria, are located near Tebessa, close to the Tunisian frontier. Of the five beds, three are workable, the thickness ranging from 3 to 9 ft., 3 to 4½ ft., and 1⅗ to 3 ft., in the different beds. The average percentage of phosphate in the rock varies, being, respectively, 58.64 per cent., 68.50 cent., and 48 per cent. in the three beds. Where the overburden does not exceed 24 ft. in depth, open-cut mining is practiced; where the overburden is deeper, the rock is mined by tunneling. Because of the basin, or saucer shape of the deposits, the tunnels are inclined.

Important deposits of phosphate rock have been found at Dy Nord and Djebel-Onck, about 62 miles south of Tebessa, which are believed to contain 300,000,000 to 400,000,000 tons of rock. These deposits have not been thoroughly explored, and estimates are therefore not entirely dependable.

[32] *U. S. Daily Consular and Trade Reports*, pp. 1240–1242 (Aug. 30, 1913).

*Setif District.*—In 1906, La Compagnie des Phosphates de Paris leased for 20 years the deposits in the commune of Bordj-Rhir. Its shipments are made by the way of a 12-mile cable to El Anasser, on the railway from Setif to Algeria; thence they go to the part of Bougie, from which the exports amounted to 64,986 short tons in 1910, 64,824 tons in 1911, and 62,702 tons in 1912.

La Compagnie Algérienne des Phosphates is exploiting two deposits in the commune of Tocqueville. The beds are from 1 to 6 ft. thick. The phosphate is transported by a 90-mile narrow-gage branch line to the railway station of Texter-Tocqueville. In 1900, 17,807 short tons, and in 1912, 230,864 short tons of rock were mined. At present 350 workmen are employed.

Another French company, with a capital of $1,000,000, is exploiting the Mzaita mine in the communes of Maadia. A broad-gage railroad has been built from the mine to the station of Ain-Tassers, and an electric plant of 150 hp. has been installed for the stamp mill. Only 7,560 tons of rock were extracted in 1912, but it is expected that eventually 300,000 tons of rock will be mined annually. The estimated supply of rock is 16,500,000 tons.

### Southern Pacific Ocean (South Sea) Islands

The South Sea islands contain the richest, if not the most extensive, phosphate deposits in the world. Japan, Australia, Hawaii, and our own Pacific coast under normal conditions, take very nearly half the production of the British, French, and German South Sea companies engaged in mining the phosphate rock. The deposits in the South Seas are notably high in calcium phosphate, and contain from 85 to 90 per cent. of this ingredient. They are also notable in their very low content of iron oxide and alumina, which ranges usually below 1 per cent.

#### *Angaur (Pellew Group)*

It is understood that the Japanese Government is now in control of the Pellew group of islands in the South Pacific, the islands having passed from German to Japanese control since the outbreak of hostilities in Europe. The principal deposits of phosphate rock are located on Angaur Island, and these are reported to be now worked to supply the Japanese fertilizer manufacturers. The deposits on the island are reported to contain in round numbers from 2,000,000 to 3,000,000 tons of high-grade phosphate rock.

#### *Ocean Island*

Ocean Island is estimated to contain not less than 50,000,000 tons of the highest grade of phosphate rock. Some shipments from this island

are reported to run from 87 to 89 per cent. calcium phosphate and less than 1 per cent. of iron oxide and alumina.

### *Makatea (near Tahiti)*

On Makatea, near Tahiti, it is estimated that about 10,000,000 tons of phosphate rock of high grade are available.

## Other Foreign Deposits

### *Christmas Island*

Christmas Island, near the west end of Java in the Indian Ocean, produces as high-grade phosphate rock as is found in the South Sea islands. Great secrecy is maintained as to the quantity of phosphate existing on the island. It has been assumed that there is an available tonnage of high-grade rock amounting to 8,000,000 tons, but this estimate is not considered authoritative.

### *Curaçao (Dutch West Indies)*

After a suspension lasting approximately 20 years, the reorganized Curaçao Phosphate Mining Co. began operations in June, 1913, at its Santa Barbara mines, shipped its first cargo in October, and is pushing business now. The phosphate goes to Germany and England. About 200 men are employed, and in one way or another a considerable amount of money is left in the island because of the industry.[33]

Curaçao produces the same high grade of phosphate rock as do the South Sea Islands. It is reported that there came from this island in 1914 about 100,000 tons of phosphate rock averaging 85 to 90 per cent. calcium phosphate. The mining of such high-grade deposits, which will tend to increase as time goes on, owing to the opening of the Panama Canal, may to a certain extent influence the export trade of Florida, but should at the same time tend to conserve these deposits for future use.

### *Chile*[34]

A large, rich deposit of phosphate has been discovered in the valley of the Huasco River, about 300 miles north of Valparaiso. Government engineers are preparing a report thereon, and it is considered of much importance, since the use of phosphate on the farms of Chile is increasing rapidly, with good results. In 1905, only 3,726 metric tons were con-

[33] Consul Elias H. Cheney: *U. S. Daily Consular and Trade Reports,* p. 359 (April 20, 1914).

[34] *U. S. Daily Consular and Trade Reports,* p. 1560 (June 15, 1914).

sumed in Chile, against 20,000 metric tons for 1912. The Government railways give a reduction of 30 per cent. on transportation charges for fertilizers.

## Available Foreign Reserves

From the recent available data, it is evident that the foreign reserves of phosphate are very large, but apparently they are not so large as those within the United States. It must be remembered, however, that the north African deposits, which are thought to extend eastward across Arabia, into Persia, have not been explored sufficiently to know even approximately what their real magnitude is. The Algerian deposits, those considered in some detail in this report and in a former report by the writer,[35] are apparently low-grade, but the apparent tonnage runs up into the hundreds of millions. The high-grade rock of the South Sea Islands is estimated approximately at 70,000,000 tons.

[35] W. C. Phalen: Production of Phosphate Rock in 1913, *U. S. Geological Survey, Mineral Resources of the United States* 1913, Pt. II, pp. 273–289 (1914).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Geology and Ore Deposits of Mohave County, Arizona*

BY FRANK C. SCHRADER,† WASHINGTON, D. C.

(New York Meeting, February, 1917)

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* Published with the permission of the Director of the U. S. Geological Survey. Originally presented at a meeting of the Southern California Local Section, October, 1915.

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## Introduction

The present sketch is submitted by request in the hope that it may serve as a basis for geologic discussion of the mining camps in Mohave County, which is experiencing a marked revival of activities.

The region, commonly known as the Mohave district and Kingman district, lies in western Arizona in the southern part of Mohave County, bordering California and Nevada on the west (Fig. 1). Kingman, the principal town, is situated near the center of the area on the Atchison, Topeka & Sante Fe Transcontinental Railway.

This region is composed of naked desert ranges of mountains and broad detritus-filled valleys, the southern extension of the characteristic topography of the Great Basin. In altitude it varies from 500 ft. in the southwest to 8,300 ft. on Hualpai Peak southeast of Kingman.

The mountains trend north-northwest. They rise about 3,000 ft. above the valleys, are generally rugged and were formed mainly by erosion. They are composed in the main of a complex of pre-Cambrian granitoid rocks which underlies the area as a whole. Like the valleys, they average about 10 miles in width. Beginning on the east, they are the Grand Wash Cliffs, the Cerbat Range, the Black Mountains or River Range, and the Eldorado Range.

The upper or dominantly cliff half of the Grand Wash Cliffs, marking the edge of the Colorado Plateau, is composed of nearly horizontal sedimentary Paleozoic strata of the Grand Canyon section, and the lower half of the underlying pre-Cambrian complex.

The Cerbat Mountains situated in the central part of the area, and the Black Mountains situated between Detrital-Sacramento Valley on the east and Mohave Valley, the great trough of the Colorado River, on the west, are locally flanked or overlain by Tertiary volcanics (Fig. 2). The latter consist of five groups of mountains of which the most important is the Black Mesa group on the south.

The Eldorado Range, rising from the great trough of the Colorado on the west and containing Searchlight, Eldorado Canyon, and other camps, is topographically and geologically similar to the Black Mountains.

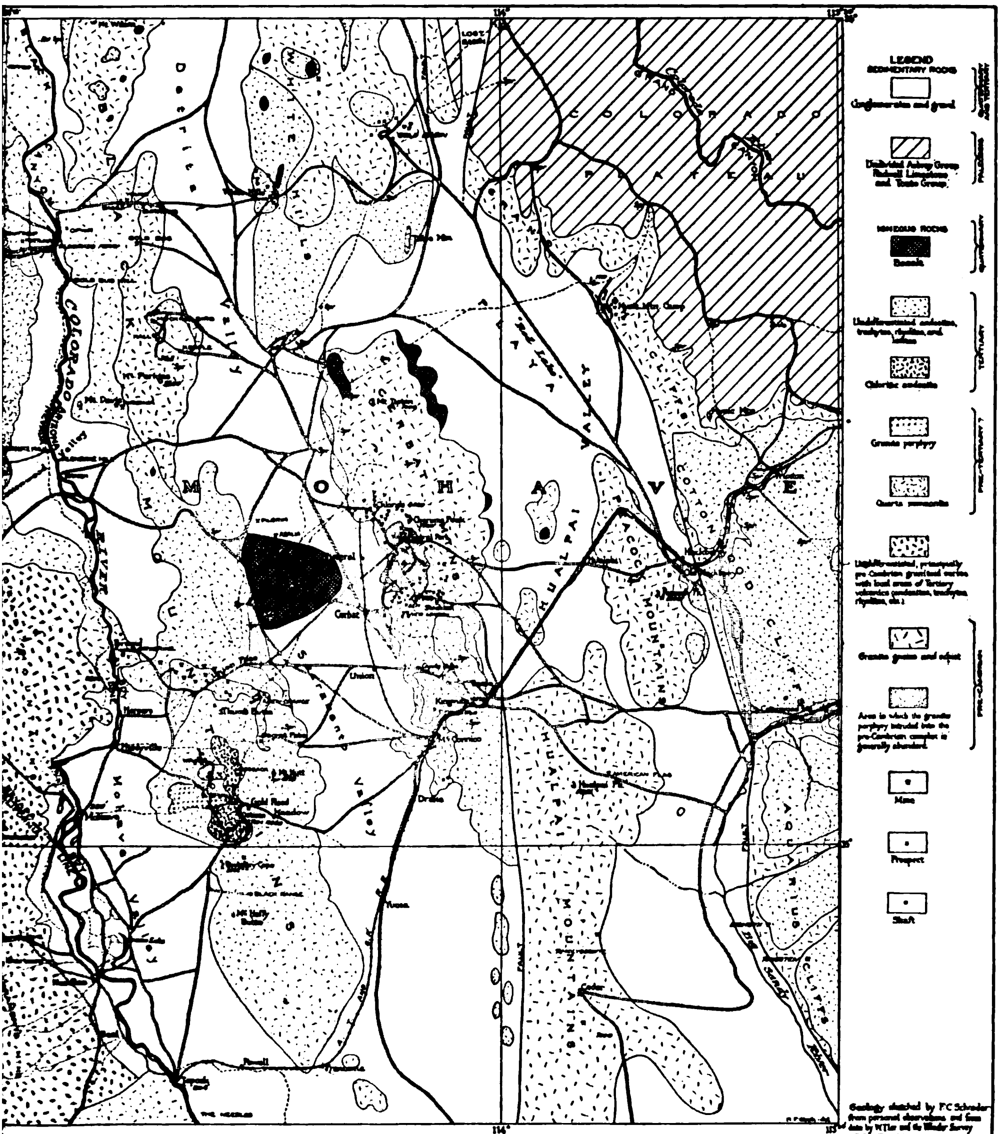
## Geology of the District[1]

The rock groups beginning with the oldest are the pre-Cambrian complex, Paleozoic sediments, pre-Tertiary intrusives, Tertiary volcanics, and Tertiary (?) and Quaternary sediments (Fig. 1). The first and third of the divisions named are the most important.

The pre-Cambrian complex consists of gray gneissoid granites, coarse,

[1] A fuller description of the rocks appears in *Bulletin* No 397, *U. S. Geological Survey* (1909).

porphyritic granitoids and their related schists, all of igneous origin and with varying degrees of metamorphism. It is traversed by a schistose

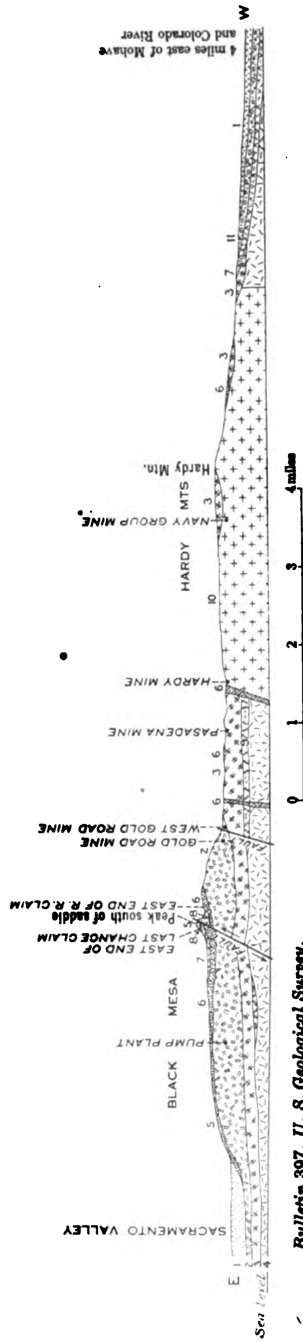



GEOLOGIC RECONNAISSANCE MAP OF THE MOHAVE COUNTY MINING REGION, ARIZONA

Fig. 1.

structure trending N. 30° E., and contains nearly all the mineral deposits of the Cerbat Range and Grand Wash Cliffs.

The Tertiary (?) and Quaternary sediments consist of several placer




*Bulletin 397, U. S. Geological Survey.*

FIG. 2.—GENERALIZED SECTION ACROSS BLACK MOUNTAINS. 1, Sands and gravels; 2, undifferentiated volcanic rocks; 3, green chloritic andesite; 4, gneissoid granite; 5, basalt; 6, rhyolite; 7, young andesite; 8, rhyolite tuffs; 9, andesite and andesite tuffs; 10, granite porphyry and micropegmatite; 11, conglomerate.

gold-bearing detrital formations or "wash", locally 2,000 ft. in thickness, partially filling the intermontane valleys.

Locally intruding the pre-Cambrian rocks are pre-Tertiary igneous masses and dikes thought to be of late Jurassic or early Cretaceous age. They occur chiefly in the Cerbat Mountains and are connected with the genesis of the deposits. The most important are granite porphyry, a light gray medium-grained rock, and lamprophyric rocks, the latter occurring mainly as dark, complementary, narrow dikes accompanying the acidic intrusives.

The Tertiary volcanics consist mainly of andesites, trachytes, rhyolites, and latites, lying in broad superimposed sheets, flows and beds locally aggregating 3,000 ft. in thickness (Fig. 2). They are best developed in the Black Mountains, particularly in the southern part (Fig. 3). They contain most of the mineral deposits of the range and played an important part in their genesis.]

## ORE DEPOSITS OF THE DISTRICT

### *General Description*

The discovery of mineral and the beginning of mining in the Mohave area date from the finding of ore at the Moss mine, 4 miles northwest of Gold Road in the early sixties. From 1904 to 1914[2] the production was nearly \$16,000,000, of which \$11,500,000 is in gold, nearly all derived from the Tom Reed and Gold Road mines. Besides gold and silver, zinc, lead, copper, tungsten, molybdenum, and bismuth are produced. The distribution of the districts or camps, about 30 in number, is shown in Fig. 4.

[2] *Mineral Resources, U. S. Geological Survey,* 1904–1914.

The deposits are contained in two distinct groups of fissure veins. The first group consists of the veins of the Cerbat Range which occur chiefly in the pre-Cambrian rocks and are genetically connected with the Mesozoic intrusives, especially granite porphyry and lamprophyric rocks. They are quartz fissure veins in which the quartz carries principally silver but also gold and ores of the other aforenamed metals. They were deposited in depth by hot waters. Their deep-seated character and close association with the major geologic structures indicate continuity in depth. They seem likely to continue productive long after the gold deposits now attracting so much attention in the volcanic rocks of the Black Mountains shall have become exhausted. Oxidation extends to depths of about 300 ft. At present the sulphide ores are principally utilized, though the rich secondary oxidized silver ores furnished most of the early-day production.

The second group comprises the veins of the Black Mountains which occur chiefly in the Tertiary volcanic rocks and whose filling besides quartz includes calcite, adularia, and fluorite. They are deeply oxidized. The valuable constituent is almost wholly free gold.

### *The Cerbat Mountains Group*

*General Description.*—The deposits of the Cerbat Mountains are mostly located at from 9 to 20 miles north of Kingman. Their production for the year 1915, according to the Chloride Mining Bureau, is $3,000,000. They occur in two sets of well-defined fissure veins, with steep dip-forming conjugate systems, one striking about N. 20° W., parallel with the dominant jointing, and the other N. 60° W. perpendicular to the schistosity of the rocks. Many of the veins have a length of nearly a mile. The structure is irregularly massive. Among the primary-ore minerals, the most important are pyrite, chalcopyrite, arsenopyrite, galena, and sphalerite; more rarely, molybdenite, gold-silver telluride and stibnite. The decrease in galena and increase in pyrite noted in the lower levels, suggests a gradual change in the primary filling. Silver and lead predominate in the Chloride, Mineral Park, and Stockton Hill districts; gold, zinc, and silver in the Cerbat district. The primary ore is leaner in gold and silver than the oxidized ore, and many mines which near the surface were silver mines, with increase in depth carried more lead, and at still greater depths have become cupriferous. The so-called "copper belt" of the area extends from Mineral Park northwestward toward Chloride for a distance of several miles. It contains the Pinkham and Midnight copper mines, and in the Mineral Park end of the belt a recently discovered "copper porphyry" deposit which is attracting attention.

The water level is found at a maximum depth of about 400 ft. In general, the ores above the water level are oxidized, but in many places

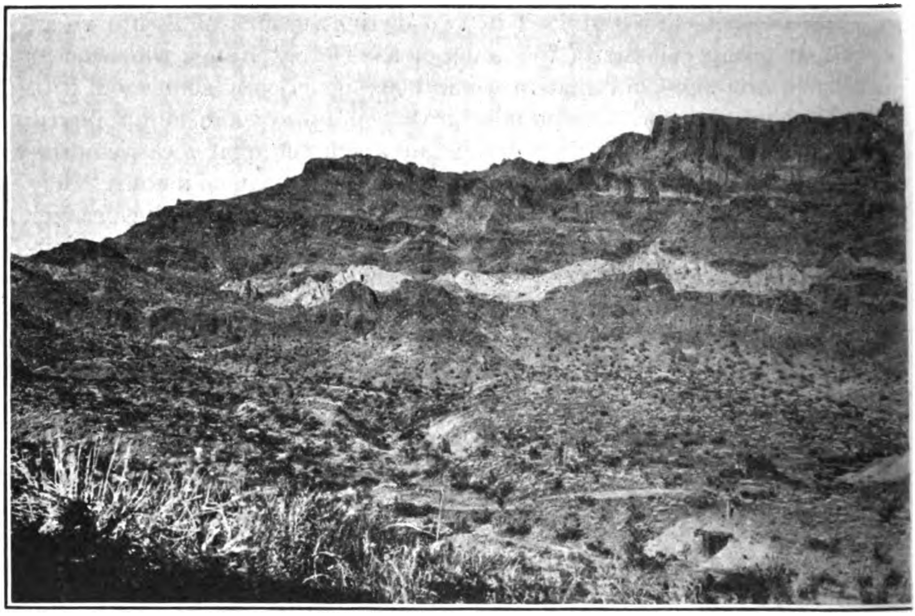
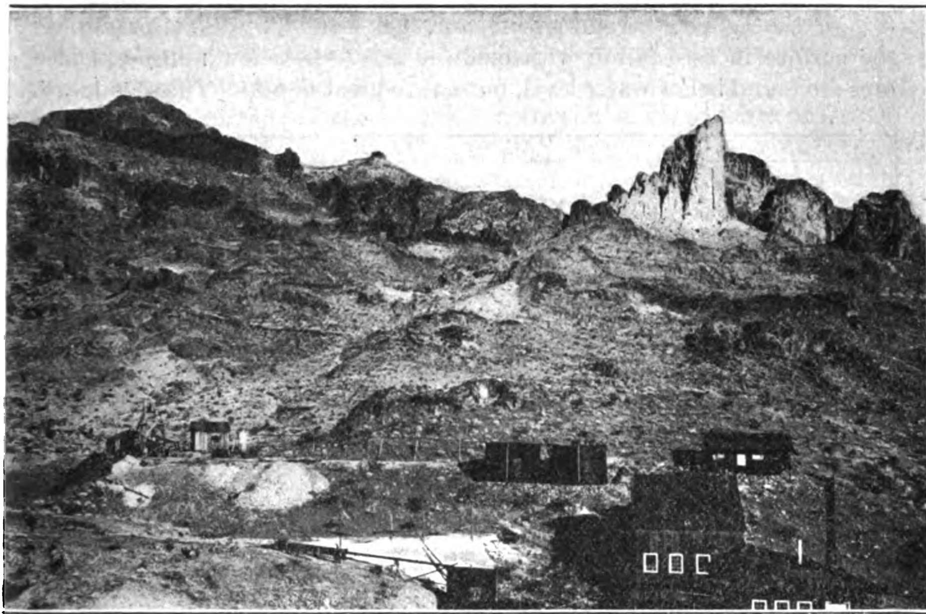


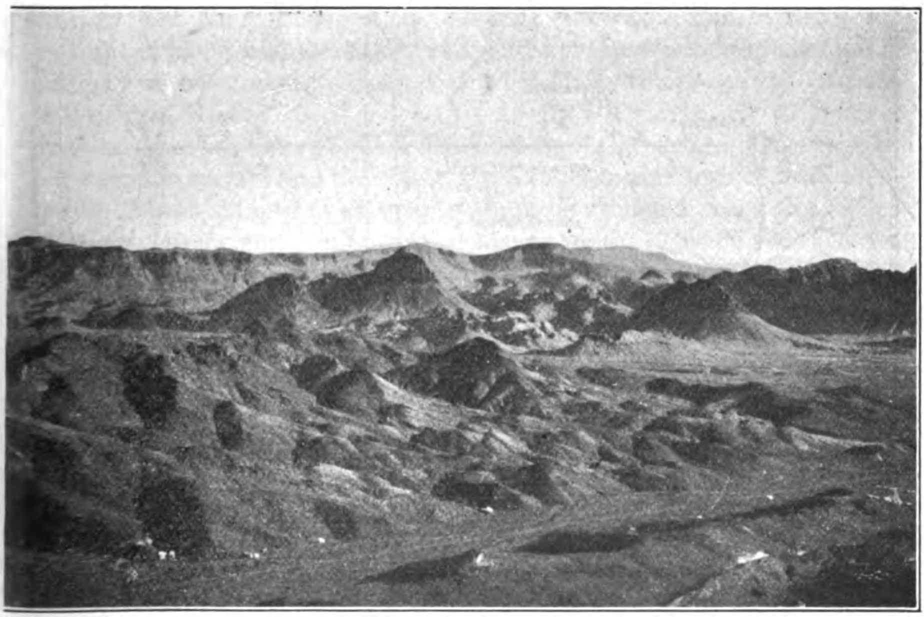
FIG. 3A.—TOM REED MINE AND VICINITY, LOOKING NORTHEAST (IN 1907). DARK

*Bulletin 397, U. S. Geological Survey.*

FIG. 3B.—FOOTHILLS OF GREEN CHLORITIC ANDESITE, LOOKING EAST FROM LELAND RANGE, 4 MILES DIS-

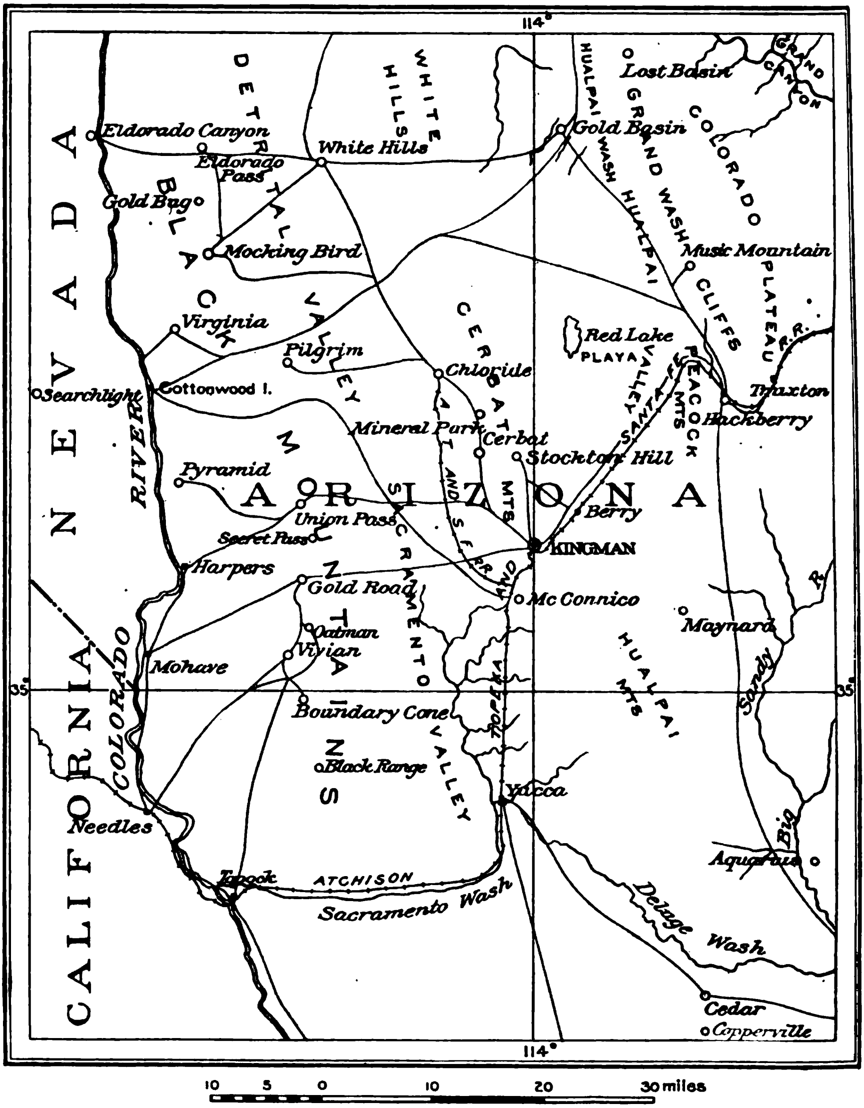


Volcanic Rocks of Range in Background, Intruded by Light-colored Rhyolite.



Mine. Mines Near Vivian, and Green Chloritic Andesite in Foreground; Main
tant, in Background.

(*e.g.*, in the Tennessee vein) galena, and also, locally, pyrite, appears near the surface in association with oxidized ores. In a few mines oxidized ores are found below water level, but not to great depth. The secondary,




*Bulletin 397, U. S. Geological Survey.*

FIG. 4.—MAP SHOWING MINING CAMPS IN THE MOHAVE COUNTY MINING REGION, ARIZONA.

or oxidized ores, consist chiefly of native silver, horn silver, and cerusite. Ruby silver and argentite are also present with oxidized ore, but do not occupy any well-defined zone between the oxidized and primary ores.

Many of the oreshoots coincide with intersections or forkings of veins. Good examples were noted in the Pinkham, Elkhart, Rainbow, Pay Roll, and Tennessee mines.

*Tennessee Mine.*—In the Chloride district, a dozen or more mines are opened to depths of 200 to 1,000 ft. or more, and expose large quantities of good gold-silver and other ores. Among them the Tennessee mine, situated a mile east of Chloride and owned by the United States Smelting, Refining & Mining Co., has long been one of the greatest lead-zinc producing properties of the State. It is credited with a present monthly production of $150,000. It has good orebodies on the 400, 800, 900, 1,000, 1,200, and 1,400-ft. levels, the last-named being the present limit of development. During a considerable portion of the time in recent years it has shipped about 200 tons of ore daily, mostly to Needles. The present daily output is said to be about 300 tons, mostly from the 1,170, 700 and 500-ft. levels.

The mine is on the Tennessee vein, which is regarded as a part of the great lead-bearing "lode" on which the Schuylkill and Elkhart mines to the north are situated. The vein is 12 ft. or more in width, and is locally banded. It dips about 68°E. in pre-Cambrian gneiss, with granite and schist near by and a pegmatite footwall reported in the lower levels. The orebodies which occur as lenses in the vein average about 5 ft. in width. The ore consists mainly of galena and blende, but carries a fair amount of silver and some gold and copper. At present the zinc ore is shipped to the company's smelter at Bartlesville, Okla., and the lead ore to Midvale, Utah.

The mine has been productive almost from the surface down. From between the surface and the 400-ft. level, thousands of tons of rich galena ore have been shipped. Here the main oreshoot had a horizontal extent of about 250 ft., and in places was 15 ft. in width. On the 400-ft. level, an orebody 21 ft. in width with 5 in. of pure galena was mined for the distance of about 40 ft. From the fourth to the fifth level there was a predominance of blende, but from the fifth to the sixth level galena increased to the proportion found in the upper part of the mine.

The 500-ft. level contained good ore for a distance of 800 ft., and the raise from it showed 12½ ft. of almost pure galena. On the 600-ft. level, the vein contained about 10 ft. of good ore. Besides the aforedescribed deposits, large bodies of good zinc ore, some 12 ft. in width, on the 200-ft. and 500-ft. levels, have been left standing in the mine. According to recent reports there has just been opened up on the 1170- and 1400-ft. levels fine bodies of ore averaging about 25 per cent. each in lead and zinc. The body on the 1,170-ft. level has an average width of 8 ft. and a known horizontal extent of 250 ft.

*Midnight Mine.*—The Midnight mine, located about 2 miles south of Chloride, and adjoining the Pinkham mine, is said to have been recently

purchased by Salt Lake parties for $250,000. It has produced considerable high-grade copper ore, which contained also important amounts of silver and gold. It is opened to the depth of 300 ft. and is said to have 25,000 tons of pay ore blocked out in the workings, including workable bodies of relatively pure zinc ore. On the 200-ft. level, where the lode is 40 ft. in width, the average zinc content is 15 per cent.

*Mineral Park.*—The copper porphyry deposit recently discovered near Mineral Park and owned by the Copperfield Copper Porphyry Co., occurs in "porphyry" which seems to be the intrusive granite porphyry aforedescribed, the abundant source of mineralization in this part of the field. The deposit is said to have a width of 1,000 ft. and a length of ½ mile. It contains seams and small bodies of chalcocite and native copper disseminated through the porphyry, which, throughout the greater portion of a 160-ft. crosscut tunnel, carries from 3 to 30 per cent. of copper, with a width of 6 ft., averaging 25 per cent. The deposit is reported to contain by estimate 100,000 tons of 5 per cent. ore. Ore removed in doing development work is reported being shipped to the Humboldt smelter.

*Golconda Mine.*—The deposits of the Golconda mine operated by the Union Basin Mining Co., in the Cerbat district, occur chiefly in the Golconda vein in the pre-Cambrian complex and seem to be associated with the Mesozoic intrusives. They have produced from essentially surface workings several hundred tons of rich ore containing chiefly gold, silver, and lead with some copper and zinc. The drift on the 300-ft. level is said to have been driven 200 ft. on a 4-ft. oreshoot that averaged about 50 per cent. of zinc, and more recently the mine is reported to be daily shipping to Bartlesville about 100 tons of high-grade zinc ore on which net returns of 9 c. per pound of zinc is realized. Some ore averaging about $12 to the ton is also treated in a 30-ton oil-flotation plant at the mine. The present monthly production is said to be about $250,000.

The mine is reported to have commercial ore on the 1,100-ft. level and a large amount of good ore in all other levels. The present production is derived mainly from the 800-ft. level. On the 900-ft. level, the oreshoot has a known extent of 850 ft. and a large tonnage of high-grade ore is being stoped. From this level a crosscut is being extended to the Tubb vein which parallels the Golconda vein 120 ft. distant on the west and has produced considerable lead-silver ore. On the 700-ft. level, the stopes are working in a 12-ft. shoot of excellent milling ore.

During the year 1915, the mine is reported to have paid two dividends of $85,000 each, which is about 20 per cent. on the issued capital, and having proved the continuity of the oreshoot in depth the company is now erecting a 200-ton oil-flotation plant for treatment of zinc ores. There is said to be $400,000 worth of zinc in the tailings on the dump and in the old stopes in the mine. The mill will be operated by electric power supplied by the Desert Power and Water Co. from its oil-burning

plant at Kingman for about $12 per horsepower per month. The power line is also being extended to Chloride and the Tennessee mine. The introduction of electric power into the Cerbat Mountain districts seems likely to result in production from many mines now dormant but which, like the Tennessee and Golconda, are known to contain workable deposits. The prospect of cheaper power in the near future is said to be good.

*Other Ore-bearing Districts.*—Deep development is also being done by the Middle Golconda Co. on the adjoining Big Bethel and Silver claims, which are believed to contain the north extension of the veins of the Golconda mine. Here the main vein is 50-ft. wide and contains much good-grade zinc ore.

A few miles to the east, the Arizona-Butte Mines Co. is building a very complete mill to treat zinc-lead-gold-silver ores of the Banner and other Stockton Hill mines.

South of Kingman in the Yucca, Cedar Valley, and Aquarius Cliffs districts, respectively, plants are in operation producing concentrates of tungsten, molybdenum and bismuth ores.

*Lost Basin District.*—In concluding remarks on the eastern part of the region, attention is here called to certain copper deposits known for a decade or more in the Lost Basin district on the northeast. The occurrence of these deposits on the trend of the great northwest-southeast mineralization belt of Arizona which contains Bisbee, Ray, Globe, Prescott, Jerome, and other important districts, and under similar geologic conditions as productive deposits in most of those districts, seems to render them worthy of mention at this time.

With a width of about 9 miles, the Lost Basin district extends from the Colorado River at the mouth of the Grand Canyon southward in the Grand Wash Cliffs for the distance of about 20 miles, with Hualpai Wash roughly forming the western boundary. The topography is mostly rugged. The country rock consists of the pre-Cambrian granitic complex, which here is considerably schistose, and the overlying Paleozoic sediments of the Grand Canyon section (Aubrey group, Redwall limestone and Tonto group).

Important gold- and silver-bearing veins occurring mostly south of the middle part of the belt in the pre-Cambrian rocks have long been worked from time to time, and some are now producing. They strike about north. Their ores are fine in texture and are excellent cyaniding ores.

The copper deposits extend from the middle of the belt eastward nearly to the summit of the Grand Wash Cliffs and edge of the Colorado Plateau. They consist of copper-bearing quartz veins and lodes which trend northwest-southeast, some being exposed by erosion through a vertical range of several hundred feet or more. They occur chiefly in the granitic rocks, but some of them, notably on the east, are in the

Paleozoic limestone and other sediments. The croppings are prominent and some are extensive. They consist principally of masses of brown and blackish iron, copper and manganese-stained quartz containing malachite and azurite. Some of the ore is reported to assay from 17 to 20 per cent. in copper and to contain also gold and silver.

### *The Black Mountains Group*

The deposits of the Black Mountains are mostly on the western slope of the range. They occur in well-defined fissure veins, but differ in most respects very markedly from those of the Cerbat Range. They are found chiefly in the Tertiary volcanic rocks, and belong to the great group of deposits found in this class of rocks throughout the West.

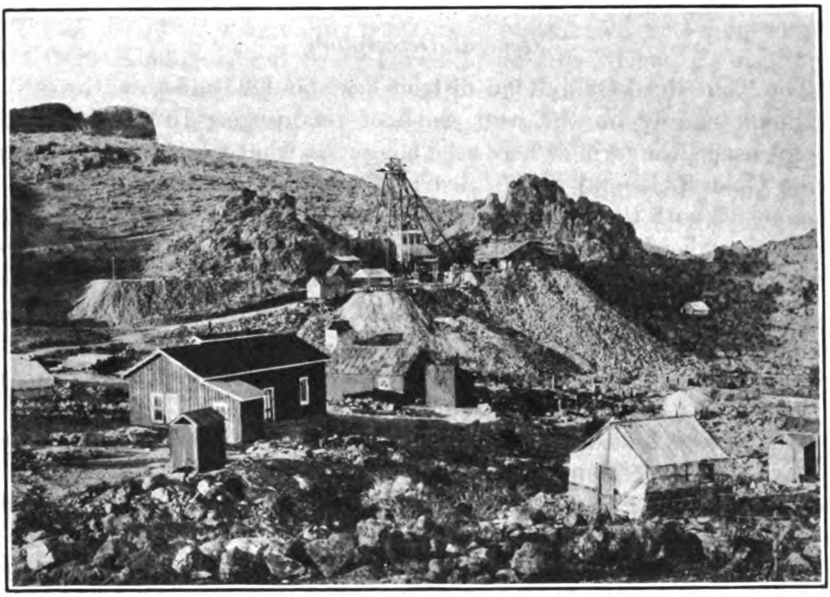
Until recently the most favorable ore horizon was regarded as in the green chloritic andesite and the undifferentiated volcanics, with profitable though subordinate deposits occurring also in the upper rhyolitic series. Recent developments, however, seem to indicate that, in the Tom Reed-Gold Road district at least, the main ore zone probably extends to a deeper horizon, in the so-called older andesite or still lower rocks.

The veins in general trend northwest-southeast with steep northeast dip. They are fairly regular, but the walls are usually rough, broken and frequently full of stringers branching off from the vein. There is a general absence of fluccan or gouge. The gangue primarily was mainly calcite and dolomitic carbonates, but these minerals have largely been replaced by quartz and adularia, a variety of orthoclase free from sodium, semi-translucent and which is so intimately intercrystallized with the quartz that it is not recognizable to the eye. The gangue contains also many inclusions of brecciated altered country rock.

A striking feature of the gangue in many places, particularly in the Tom Reed-Gold Road district, is its characteristically laminated or platy, bladed and cellular structure, pseudomorphic after calcite, barite or other spar in which many contiguous or connecting plates are variously arranged. This material is aptly termed by the miner "fish-scale quartz," from the adjacent plates partly overlapping one another. The plates range from minute up to an inch in diameter and from the thickness of paper to 1/10 in. in thickness. Much of the quartz intimately associated with the better-grade ore is of greenish or yellowish-green color and waxy luster, which has led to inquiry concerning the source of the color. The cause of the color is not definitely known, nor easy to determine. From preliminary tests it seems to be mainly silicates of iron, manganese, and perhaps other minerals, chlorite, actinolite, rhodonite, dioptase, etc., which may be an important source of the black iron and manganese oxides common as stain, small bodies and pockets in the croppings and more oxidized ores. It is noticeable that the greenish quartz

occurs more frequently in a crustified or banded form than does the uncolored gangue, which method of forming more readily favors the entering of various salts and minerals into its composition. In the Miller mine on the Hardy vein, 2 miles west of Gold Road, the greenish color of the quartz seems to be due largely to fluorite which is present in considerable quantity in the vein, much of it being replaced by quartz.

The deposits seem to have been formed near the surface by thermal solutions which circulated through the lavas at the close of igneous activity. They seem to belong to the late Tertiary epoch of metallization. They are oxidized to depths of 600 to 700 ft. and, as a rule, contain little or



*Bulletin 397, U. S. Geological Survey.*

FIG. 5.—GOLD ROAD MINE, MAIN SHAFT LOOKING SOUTH.

Silicified lode and vein wall croppings on both sides. Edges of heavy flows of volcanic rocks in left background.

no sulphides. Gold is almost exclusively the valuable constituent, usually no base metal being present. The gold as a rule is free, but occurs in very minute particles and is best recovered by the cyanide process. Gold telluride is reported from a few mines.

There is no gossan nor iron hat in the outcrops of the veins. In general, the veins weather in relief only where the filling consists chiefly of quartz or a mass of cemented silicified rock. There the croppings form prominent reefs. Likewise, the vein walls are frequently strongly silicified and hardened with the result that they too weather in forms rising to heights of 20 ft. or more above the surface and extending for considerable distances as seen at the Gold Road mine, Fig. 5. This hardened wall

rock, or so-called "ledge matter," is sometimes netted by stringers of quartz branching off from the vein. It denotes arresting or damming back of copious mineral-bearing solutions that circulated at the locality, and generally indicates workable deposits in the adjacent underlying portion of the vein as described later under the Gold Road mine. Many of the deposits, as exemplified by the Tyro, the Gold Road, and other veins, carry relatively unimportant values near the surface.

Of the ten or more districts in the range the most important is the Tom Reed-Gold Road district.

## Tom Reed-Gold Road District

### *General Description*

The Tom Reed-Gold Road district lies about 25 miles southwest of Kingman, mainly on the west slope of the range. In keeping with present usage, the term as here used comprises what was formerly known as the Gold Road and Vivian districts, and the area is approximately co-extensive with the southern part of the San Francisco district of early days. The district has a north-south length of about 10 miles and a width of 6 miles. The principal camps and centers of activity are Oatman, the settlement of the Tom Reed and neighboring mines, situated in the west slope of the range 27 miles from Kingman, and Gold Road, 2 miles north of Oatman. For more than a year Oatman continued to be the center of attraction in the Southwest.

Mineral was first discovered in the district, as aforedescribed for the Mohave County region, in the early sixties at the Moss mine 4 miles northwest of Gold Road. The mine soon produced $240,000, in gold from rich surface ore.

Production in the district has continued more or less steady since the discovery of the Gold Road mine in 1902. Recently discoveries in the Tom Reed mine and vicinity have been attracting attention to the district, with the result that the value of the plants and machinery now installed at the various mines is said to aggregate nearly $2,000,000. Some 50 odd plants are in operation. The greater portion of them have been installed since the first of the year 1915, during which time nearly 200 companies have been organized to operate in the district, of which 150 are fully equipped and most of the others are receiving machinery. Thirty or more properties hitherto dormant have become active, and the population, which is gathered from all the mining camps in the West, has increased from 600 to more than 7,000 and is gradually increasing. Oatman, which is said to recall Goldfields' boom, is described as a well-equipped, substantial town, a cleanly, orderly, model camp, where living is such as one expects to find only in large towns. It is electrically lighted, has three news-

papers, schools, churches, a stock exchange, and well-stocked stores and business houses of all kinds. It is rapidly becoming the outfitting center for a large territory containing many new districts for a distance of nearly 100 miles north and south along the range. Wildcatting is checked by the laws of Arizona.

Owing to the demand for building space, a new camp, Old Trails, has grown up in the broad wash south of Oatman and there are a dozen other surrounding town sites and additions. Mail, passenger, and express service is by automobile chiefly from Kingman. A similar service exists between Oatman and Needles, 20 miles distant on the southwest, in which the Colorado is crossed by boat. Freight is delivered in the district at a cost of $11.50 per ton by motor truck, road locomotive, and mountain tractor from Kingman, and from Topock, formerly Mellen, located 25 miles distant at the east end of the railroad bridge crossing the Colorado. Early construction of a branch railroad from Topock to Oatman, which is quite feasible, is reported under consideration.

Good water for domestic use is pumped from neighboring springs found mostly in the porous rhyolite tuff or water rock. Seemingly, ample water for milling purposes is being found in deep mining. It also is palatable, being suitable for domestic and all other purposes. Should an additional water supply be required, it can be pumped from the Colorado River 14 miles distant, preferably from wells sunk in the gravel beds on the bank of the river. For this purpose, it is said, a company is being organized.

*Development*

With a capitalization of more than $53,000,000, operations are being actively prosecuted by 125 separately organized mining corporations and the activity seems to be warranted by substantial results of nearly all deep development. More than half of the companies now in the field are sinking or prepared for deep mining. Ten have good milling ore opened up and four have large producing mines of proven merit. Mining operations are steadily increasing in volume and area; more than 2,200 miners are actually employed in the district and more than $25,000 per day is being expended for wages and equipment.

The approved method of prospecting is sinking to depths of 300 to 500 ft. and then crosscutting and drifting. Practically no surface work is carried on. Gas-engine hoists, compressors, and Jackhamer drills are the usual equipment. Usually also much lateral development must be done before pay ore in large quantities is found and the mine proved. The automobile, a prominent feature in the present activity, has taken the place of the burro in prospecting.

The cost of mining and milling is about $6 per ton, of which $1.25 is for power. The power at the larger mines is electric power supplied

by the oil-burning plant at Kingman. At the Gold Road mine, treating 200 tons of ore daily, the best record obtained for mining and milling is reported to be slightly less than $3 per ton. At the Tom Reed mine, however, where 20 stamps are used, the cost is about $6. There is said to be no profit in treating $5 ore in the district on a small scale. Both the Gold Road and Tom Reed mines treat their ore by the cyanide process, and have installed the counter-current decantation system.

From what has been said of the Tyro and Gold Road veins, and from the large number of other widely distributed profitable orebodies being found at depth and the cost of mining and milling, this is not a camp for the small operator but seems rather to offer encouraging possibilities for capital to engage in deep mining. The district has received the approval of many eminent mining engineers, a number of whom have become investors there and are now directors in some of the larger companies.

### *Topography*

The district lies mainly in the Black Mesa Mountains, which, with an average elevation of 4,000 ft., extend from Gold Road 20 miles southward to the end of the range east of Needles. Their rugged forms are due chiefly to deep dissection of a huge volcanic plateau known as Black Mesa.

The district ranges in elevation from 2,000 ft. on the west and about 3,000 ft. on the east to 4,500 ft. at the top of the range. The range portion, which is about 4 miles in width, is marked by deep canyons, steep slopes, and peaks. In a horizontal distance of about 1½ miles, the surface declines from the elevation of 4,500 ft. at the crest to 2,500 ft. on Silver Creek just below Gold Road. The edges of the harder lava beds present steplike cliffs (Fig. 3 *A*).

The principal outliers are the Hardy Mountains, a group of hills situated about 3 miles west of Gold Road. They are about 3 miles in diameter and rise about 600 ft. above the surrounding country. Two miles to the north is a smaller group, the Moss Hills, while Leland Mountain at Vivian represents similar features on the southwest.

### *Geology*

The Tertiary volcanic rocks prevail, particularly in the eastern or range portion of the district. They practically constitute the range, dip gently eastward toward its axis and are in places covered by younger rhyolite, andesite and basalt. In the southern part the green chloritic andesite is dominant, while on the west occur also local areas of the pre-Cambrian gneiss, younger granite porphyry and micropegmatite, greenstone agglomerate, and overlying sheets of supposed Tertiary conglomerate and younger gravel and lava flows. Locally intervening between the pre-Cambrian and the overlying volcanics are occasional remnantal patches of tilted and metamorphosed Paleozoic limestone and shale be-

longing to the Grand Canyon Section. These sedimentary rocks are not as yet known to have any bearing on the deposits or mining other than to indicate to the miner where encountered the general lower limits of the volcanics.

Recent mine developments have disclosed the geology of the ore-bearing volcanics to be more complicated and seemingly of more importance to the district from a gold-producing standpoint than was at first supposed.

In the vicinity of Vivian, and extending from there toward Oatman, occurs the older or basal andesite, which is light gray, calcitic, 300 ft. in thickness, and rests mainly on the pre-Cambrian complex and Paleozoic sediments. The older andesite, however, is not known to be of wide extent in the district, a fact seemingly overlooked by Bancroft and others. It is seemingly absent from Secret Pass where the next higher rock, the green chloritic andesite rests directly upon the pre-Cambrian granite, and from the Hardy Mountains where the green chloritic andesite similarly rests upon the Mesozoic granite porphyry or micropegmatite.[3] It is not known to be present at the Gold Road mine, and according to Sperr[4] the rock underlying the green chloritic andesite in the deep workings of the Tom Reed mine does not correspond to the older andesite described at Vivian. The older andesite is unconformably succeeded by another series of flows, the green chloritic andesite which contains an important part of the mineral deposits in the Tom Reed-Gold Road district (Figs. 1 and 3*B*). The flows aggregate a known thickness of 800 ft. The rock consists mainly of a greenish, fine-grained groundmass containing abundant whitish feldspar phenocrysts. It is very chloritic and calcitic. It is intruded by black latite and younger lavas.

The intrusive character of the green chloritic andesite or rocks grouped with it is well shown at the head of the wash, just west of the Leland mine, where dikes from 2 to 20 ft. in width, given off from the main mass, extend ⅛ mile or more westward into the older andesite. A black, fresh-looking specimen of it collected by the writer from the Leland mine proved by microscopic study and chemical analyses to be latite, and it contains chlorite in abundance throughout.[5]

The intrusive nature of the green chloritic andesite and the association of ore deposits with its intrusive phases in various parts of the district are also abundantly corroborated by later work of Sperr, Probert, Bancroft, and other engineers. Probert[6] believes it to be both intrusive and

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[3] *Bulletin No.* 397, *U. S. Geological Survey*, p. 35, and Fig. 2 (1909).

[4] J. D. Sperr: The Tom Reed-Gold Road Mining District, Arizona, *Engineering and Mining Journal*, vol. 101, No. 1, pp. 1–5 (Jan. 1, 1916).

[5] *Bulletin No.* 397, *U. S. Geological Survey*, pp. 36–37 (1909).

[6] Frank H. Probert: Oatman, Arizona—A Prohibition Camp, *Mining and Scientific Press*, vol. 112, No. 1, pp. 17–20 (Jan. 1, 1916).

extrusive, that dikes and sills of it occur in the older andesite and that mineralization is dependent upon this association.

Bancroft[7] writes that in the vicinity of the mines which he examined in localities rather widely scattered in the district, he found evidence of the intrusive nature of this formation, and that the orebodies are largely formed within the intrusive.

More recently, according to Smith,[8] the bottom as well as the collar of the Tom Reed shaft at 1,075 ft. in depth was in the green chloritic andesite which in the bottom of the shaft was ore-bearing, and he suggests that the rock may here be intrusive. The supposition of the rock being here intrusive, probably as a neck, would help to account for the unusual thickness of the formation at this point, which seems to be local, since elsewhere in the Tom Reed mine and in the neighboring United Eastern, Pioneer and other properties the workings, according to Schader,[9] passed through the green chloritic andesite and into the older underlying andesite at shallower depths and have workable ore in the lower rock.

Therefore, according to the observations of six or more investigators, the green chloritic andesite (formation) contains rocks which vary considerably mineralogically from the normal andesite, rocks with which the ore deposits in general seem to be associated and which are known to be intrusive into the older andesite. The most important of these rocks seems, to the present writer, to be the dark latite occurring at the Leland mine and elsewhere. It seems to intrude not only the older andesite but also the green chloritic andesite as sheets, necks and dikes, and to be intimately connected genetically with the ore deposits. More recently, Sperr,[10] whose observations in the district have been extensive, regards all the commercial ore as occurring in the andesites intimately associated with latites. The intrusive nature of the rocks associated with the ore deposits obviously favors continuity of the deposits in depth.

The deposition of the green chloritic andesite was followed by atperiod of great fissuring and faulting accompanied and followed by erupion of the next higher group, the undifferentiated volcanic rocks 2,000 ft. in thickness, containing the Gold Road and other important veins, and by intrusions of younger rocks, especially latite and rhyolite in the form of dikes, necks, and rounded plug or stocklike masses, and seemingly the formation of many of the larger fissure veins. The undifferentiated volcanics are succeeded by a series of younger light-colored tuffaceous

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[7] Howland Bancroft: Geology of Gold Road District, *Mining and Scientific Press*, vol. 3, No. 1, p. 21 (July, 3, 1915).

[8] Howard D. Smith: The Oatman District, Arizona, *Mining and Scientific Press*, vol. 3, No. 5, p. 172–175 (July 31, 1915).

[9] Carl F. Schader: Personal letter, Feb. 6, 1915.

[10] J. D. Sperr: "Conversational Geology" at Oatman, *Engineering and Mining Journal*, vol. 101, No. 26, p. 1119 (June 24, 1916).

rhyolites locally 1,000 ft. in thickness and known as the "water rock," which is succeeded by dark reddish andesite which in turn is followed by black olivine basalt, the youngest of the effusive rocks, which remains as a capping over a large part of the Black Mesa Mountains.

With the extensive development recently done in the district, the rocks merit detailed study with reference to their sequence and bearing on the genesis of mineralization. Such a diagnosis seems certain to prove of great economic value in preventing useless expenditure of money in some directions and leading to profitable development in others.

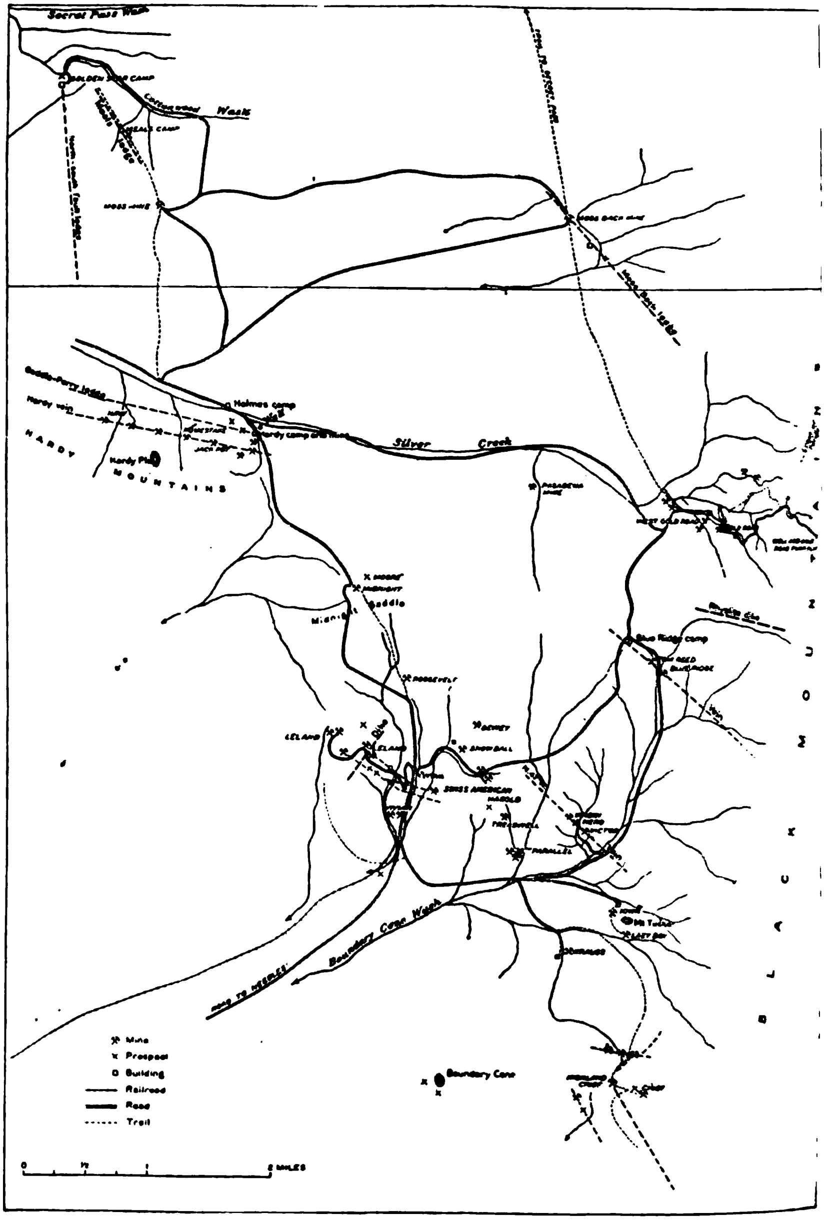
### *Ore Deposits*

*General Description.*—The deposits, which are numerous, are chiefly gold-bearing fissure veins or lodes of the character already described for the Black Mountains. The veins vary from 5 to 70 ft. in width and from a few hundred feet to several miles in length. In general they are strong and persistent. They strike northwest with steep dip to the northeast. They are almost devoid of metallic sulphides, the gold being free. They occur chiefly in the lower part of the undifferentiated volcanic series, the green chloritic andesite, the granite porphyry and micropegmatite, other underlying rocks and also along certain contacts, where latite and rhyolite are generally the intrusives. Some of the deposits are very rich, but the large bodies of low-grade ore constitute the main resource. Ore having a metallic content of $10 or less is considered low-grade.

The older andesite, from the ill behavior and feathering out of certain vein deposits on entering it from the green chloritic andesite, was originally regarded by the writer as unfavorable for mineral, or essentially barren, particularly in the Vivian district. Owing to its tufaceous brecciated and fragmental nature it is almost devoid of lava-cooling shrinkage cracks and fissures, which elsewhere form favorable repositories for ore deposition. According to Palmer[11] "the occurrence of any oreshoots in the earlier (older) andesite is yet to be demonstrated."

Also E. W. Brooks limits the area of commercial mineralization in this part of the field to the green chloritic or "younger andesite." Later developments, however, it is gratifying to note, in the Oatman and Vivian camps, report workable ore deposits in the older andesite also. It is hoped that with development similar reports may be received from several mines near Vivian which, though well-equipped for operations nearly a decade ago, have remained inactive. That the writer has never doubted that major veins probably occur in and below this formation is evidenced by the following statement: "The veins cut through the great mass of Tertiary volcanic rocks which characterize the range and un-

[11] Leroy A. Palmer: The Oatman District, Arizona, *Mining and Scientific Press*, vol. 113, No. 6, p. 195 (Aug. 5, 1916).




*Bulletin* 397, *U. S. Geological Survey.*

FIG. 6.—MINES AND VEINS IN THE TOM REED-GOLD ROAD DISTRICT.

doubtedly continue in depth into the underlying pre-Cambrian granitic rocks.[12]

According to Palmer,[13] "some ore of value has recently been found in the pre-Cambrian."

Since the deposits are confined to the vein filling and do not as a rule form metasomatic replacements in the wall rock, as at Cripple Creek and other camps, the selective preference which any bounding wall rock by reason of its more favorable physical or chemical properties for replacement may exert in favor of ore deposition seems to be practically nil. Accordingly, there is no apparent reason, other conditions being equal, why the deposits should not be equally developed in any one of several formations through which the fissure vein with like strength may extend.

The deposits consist of two types—those in which the gangue is chiefly quartz and adularia and those in which it is chiefly calcite. The source of the quartz and adularia is referred to the siliceous magmas and that of the calcite to basic or andesitic magmas with possible contributions derived from underlying limestones. The former carry the best values, occur mostly in the undifferentiated volcanic rocks and in granite porphyry and have a general northwest-southeast trend. The latter seem to occur mainly in the green chloritic andesite and trend more nearly north-south. Among the most important of the former type are the Gold Road and Tom Reed veins; among the latter, the Pasadena, Mossback and Meals veins (Fig. 6). In some cases the veins are associated with boldly cropping silicified dikes of which the deposits in certain instances may be in part replacement.

According to Platts,[14] the most productive veins, such as those in the Tom Reed, United Eastern, and Big Jim mines, are in a complicated series of fissures, part of which strike about N. 45° W., and others N. 60° W., producing with each other a conjugated system with numerous intersections near which many large orebodies are found.

*Zones.*—Surficially, the veins seem to mostly fall into four main zones[15] which, named in order from north to south, are the Gold Road, Tom Reed, Vivian, and Black Range zones. The Tom Reed zone is the best developed and contains the most interesting discoveries.

There seem also to be two or more horizons or vertical ore "zones." The largest and richest orebodies seem in general to lie in a "zone" of enriched oxides between the 300-ft. and 500-ft. levels. Below this zone

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[12] *Bulletin No.* 397, *U. S. Geological Survey,* p. 48 (1909).

[13] Leroy A. Palmer: *Op. cit.*, p. 195.

[14] J. B. Platts: Geology of Oatman, *Mining and Scientific Press*, vol. 112, No. 23, p. 814 (June 3, 1916).

[15] Leroy A. Palmer: The Oatman District, Arizona, *Engineering and Mining Journal,* vol. 101, No. 21, p. 895 (May 20, 1916).

the ores decrease in value, but continue to be of workable grade beyond the deepest point yet penetrated by any working. The richness of this zone as suggested by Smith[16] is probably due to secondary enrichment, by contributions leached from shallower depths, in support of which the presence of vugs and manganese oxide in the upper part of the veins is cited. This view is also seemingly corroborated by the tendency of the zone to parallel the contour of the surface. For instance, its occurrence at about the same depth in the Gold Road mine as in the Oatman camp, though at correspondingly greater elevations and higher geologic horizons. The gold was probably precipitated in large part along with the manganese oxide.

If the thickness of 600 or 800 ft. assigned to the green chloritic andesite be correct, this ore zone in the Oatman camp or, more generally speaking, in the triangular area of several square miles comprised between the Tom Reed, Pioneer, and Pasadena mines, should lie mainly in this formation.

There seems to be also present, notably in the Oatman camp and vicinity, a shallow or surface ore zone of leached oxides to which pay ores found at or near the surface are generally confined. It extends to depths of about 150 ft., between which and the zone of enriched oxides, or 300-ft. level, lies a 150-ft. intermediate zone of leached or relatively barren ground, although the valuable oreshoots, according to Sperr,[17] almost without exception come at least within 100 ft. of the surface.

These two zones have probably suffered about the same amount of leaching, the upper zone certainly not less than the intermediate or barren zone. The upper zone seemingly owes its greater ore content to the more siliceous, and consequently resistant, character of the ore which accordingly better withstands the process of leaching.

Differing from the view of enrichment by leaching and redeposition in the main zone is that of Platts[18] which holds that the ore is essentially a primary deposit formed by heat ascending solutions, that from the nature of the gangue it is evident that acid solutions could not exist, and that, except for the oxidation of the pyrite, there is no evidence of the action of surface water on the ore.

It seems quite possible, as suggested by one writer, that the groundwater table in the district may be in part dependent upon the neighboring Colorado River. If this view be correct, physiographic study will probably be able to correlate certain horizon features of the vertical section as leaching, etc., with relatively prolonged pauses in the historical downcutting of the river. It does not, however, seem safe to assume that the water table at Oatman coincides with the level of the Colorado River,

[16] *Op. cit.*, p. 173.

[17] J. D. Sperr: "Conversational Geology" at Oatman, Ariz., *Engineering and Mining Journal*, vol. 101, No. 26, p. 1119 (June 24, 1916).

[18] J. B. Platts: *Op. cit.*

which is 2,000 ft. lower than Oatman, and that therefore the ores if they persist downward will continue to be oxidized and of the same milling character to that depth as advocated by Palmer.[19] Owing to the greater elevation of the gathering zone on the east, which probably extends to the Hualpai Mountains, or longitude of Kingman, the ground-water table is not a level surface but gradually rises from the Colorado River, eastward, and at Oatman it probably stands several hundred feet or more above the level of the river.

*Structure of Oreshoots.*—The ore occurs chiefly as a series of more or less tabular or lenticular oreshoots and pay streaks dipping and plunging variously within the vein, with which they exhibit a greater or less degree of parallelism. The shoots vary from 1 ft. wide to the width of the vein. They usually carry gold for their full width. They range up to nearly 1,000 ft. in length and depth, and there is a general similarity or repetition of the shoots in the same vein. They seem to have been formed by thermo-aqueous processes that followed igneous activity. In general, the quartz and values favor the hanging wall, which of the two walls is generally the best defined, and contains stringers branching off obliquely from the vein, while the spar or calcite favors the footwall. The gold is mostly associated with the quartz-adularia gangue and not rarely where sulphides have existed, it, according to Platts,[20] occurs in hematite (which is pseudomorphic after pyrite) in the quartz.

According to Palmer,[21] the first indications of the vein encountered in sinking are small stringers of quartz and calcite scattered through the andesite, usually accompanied by slight pyritization in the vein-wall andesite which yields a little free gold in the pan, while in the oreshoots the vein matter shows pronounced hematite and manganese stains. It is said that the problem in development is not so much the finding of veins as the discovery of oreshoots in the veins, that nothing sufficiently tangible has yet been found to use as the basis for a theory to guide the operator in the search for ore.

Though no rigid rule can be laid down to guide the operator in search for ore, nevertheless, from the apparently well-established facts that the metallic values have been largely imported by the replacement quartz-adularia solutions and that more gold is found where the replacement of calcite is most nearly complete, in formulating plans of exploration much benefit in most cases should be derived from a correlative study of the criteria indicating the probable courses followed by these solutions, namely, quartzose vein croppings, silicified wall rock, the quartz pseudomorphic structures, etc., which have been described. It was the quartz

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[19] L. A. Palmer: The Oatman District, Arizona, *Mining and Scientific Press*, vol. 113, No. 6, p. 196 (Aug. 5, 1916).

[20] J. B. Platts: *Op. cit.*

[21] L. A. Palmer: *Idem.*, vol. 101, No. 21, p. 896.

adularia or siliceous waxy-appearing character of the deposits seen in the Tom Reed mine and the recognition of their marked similarity to the then-producing deposits of the Gold Road mine that apparently led to the resumption of operations in the Tom Reed mine following the writer's visit in 1907.

### *Gold Road Mine*

Some of the principal mines in which the deposits have been worked are the Gold Road, Tom Reed, Leland, Pioneer, Victor-Virgin, Midnight, and United Eastern. Their general distribution is shown in Fig. 6. The most important producers are the Gold Road and Tom Reed mines.

*General Description.*—The Gold Road mine, owned by the United States Smelting, Refining and Mining Co., is situated at Gold Road, on the western rugged slope of the range about 1 mile below the crest, at an elevation of about 2,900 ft., mainly on the western part of the Gold Road vein. From its croppings, the Gold Road vein has long been known, but the discovery of values which resulted in the opening of the mine was made about 1902. The mine soon began to be worked systematically and paid dividends of 5 per cent. on the capitalization of \$500,000. Several years ago it was acquired by the present owner for \$1,500,000.

The mine is opened to a depth of 1,000 ft. or more, mainly by shafts and drifts, and ore has been mined in quantities for a distance of 4,000 ft. on the vein.

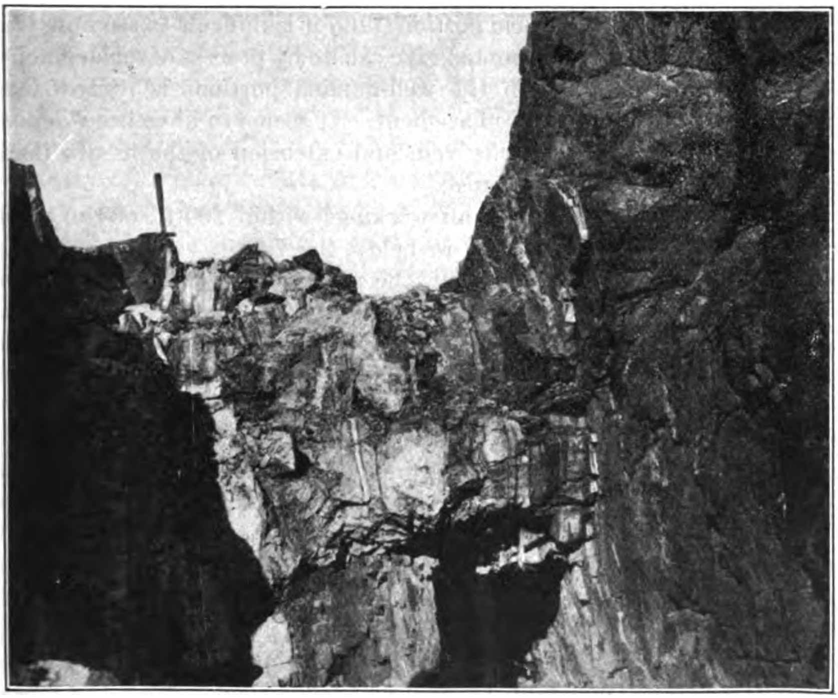
Most of the work done up to 1907 was comprised in the ground extending about 200 ft. from the main or Gold Road shaft (then known as the Gold Road mine), in either direction on the vein, and to the depth of 500 ft. Since then, however, extensive developments have been made, especially to the east on the Billy Bryan and adjoining ground.

The gross production of the property is estimated at more than \$6,000,000. To the end of 1907 the production was about \$2,250,000, most of which was made during 1905 and 1906. Since 1909 the output has averaged about \$800,000 annually, which figure in some years was much exceeded. The present monthly production is nearly \$80,000 in gold bullion.

*Geology.*—The country surrounding the mine is occupied by the undifferentiated volcanic rocks consisting mainly of heavy sheets of volcanic flows (Fig. 5). The series comprises andesite, trachyte, latite, rhyolite, and dacite, which aggregate nearly 2,000 ft. in thickness and are difficult to separate. The series extends from a point about 1,000 ft. west of the main shaft eastward nearly to the crest of the range. On the west it gives way to the underlying green chloritic andesite, which in the West Gold Road mine, situated near the contact, has been penetrated to a depth of 455 ft. On the east it is overlain by the upper rhyolite. The

contact on the west is probably a fault contact, as some of the green chloritic andesite occurs in the main shaft workings. The dominant dynamic structure is a pronounced close sheeting, which strikes N. 40° W., with vertical dip. The series is intruded by dikes of rhyolite and the younger dark andesite. Much of the rhyolite carries gold values. A dike of it, 40 ft. in width, located about 600 ft. south of the mine, is said to average about $5 in gold to the ton, the gold occurring chiefly in contained stringers of quartz and calcite.

*Ore Deposits.*—The Gold Road vein extends from a point about 700 ft. west of the main shaft southeastward through the Gold Road, Rail-



*Bulletin 397, U. S. Geological Survey.*

FIG. 7.—GOLD ROAD VEIN AT EAST OF LINE ROAD CLAIM, LOOKING NORTHWEST.

road, Billy Bryan and Last Chance claims to beyond the crest of the range, a distance of nearly 1¼ miles. It dips about 80° NE. approximately parallel with the close sheeting in the country rock. It varies in width from 22 ft. on the west to less than 1 ft. on the east. Between the bottom of the mine and the crest of the mountains it has a known vertical range of about 2,300 ft., and its croppings have a vertical range of about 1,300 ft. They consist essentially of iron and manganese-stained quartz, silicified rock, and calcite. In places they form conspicuous reefs or knobs rising 20 or 30 ft. above the surface, as at the main shaft or Gold

Road mine (Fig. 5). Here the associated hard silicified prominent wall-rock croppings on both sides of the vein have a lateral extent of 20 to 60 ft. The best ore usually underlies these prominent croppings, which seem to represent pool-like courses along which mineralizing solutions deposited more freely than elsewhere. Where the croppings weaken or break down, the underlying portion of the vein generally becomes lean or barren, though the fissure and its walls and filling may continue unchanged.

The vein consists essentially of quartz and adularia, with some calcite and brecciated altered rock, and is locally more or less crustified (Fig. 7). Since the more siliceous portions frequently exhibit a perfection of crustification not found in the calcic portions, and it is difficult to see how this structure could be derived from massive calcite by process of replacement, the quartz-adularia filling in the well-banded portions is regarded as largely primary rather than replacement. It seems to have been deposited in reopenings of the calcite vein and extension of the fissure itself eastward into the axis of the range.

In 1907, the vein in nearly all workings within 200 ft. of the main shaft and from the 300-ft. level to below the 700-ft. level was mostly good milling ore from wall to wall, the amount of waste in mining as shown by the dump (Fig. 5) being very small. Elsewhere, however, as seen on the Billy Bryan and other ground, portions of the vein, sometimes for a considerable extent, are relatively barren.

The vein is strongest on the west, where, as developed in the main shaft workings, it is uniformly about 10 ft. in width. It is usually in sharp contact with well-defined firm walls of the country rock, consisting of andesite, trachyte, latite, and rhyolite. It is, in general, "frozen" to the walls and is locally enriched where stringers extend from it at acute angles into the hanging wall. As a rule, hard and rough walls indicate good ore, and conversely, soft and smooth walls generally correspond with lean ore. The ore consists chiefly of fine-grained, light-greenish or waxy quartz. Much of it is platy or hackly with a peculiar chalcedonic or drusy appearance. Much of it is pseudomorphic after calcite and many of the pseudomorphic plates are thickly studded with minute quartz crystals of a still younger growth. The gold, as seen microscopically, and in places by the naked eye, is very finely disseminated, principally in the quartz-adularia portion of the gangue. On the 600-ft. level, however, the vein contained more spar or calcite than on any other level, and here the oreshoot is reported to have had an uninterrupted extent of 1,100 ft. Some sulphide ore containing pyrite, the first encountered in the mine, was found here, and it is reported to have contained higher values than the overlying oxidized ore, which is probably due to arrest in the downward progress of leached concentrates. On the 700-ft. level, the country wall rock, consisting of green chloritic andesite, was more or

less pyritic and seemed to indicate that the mine was entering the sulphide zone. The mine has not produced much ore from depths shallower than the 300-ft. level. The lowest-grade ore taken out was obtained between the surface and the 300-ft. level, where it fell to $5 a ton. The ordinary mill-run ore, it is said, rarely falls below $8 or $9 and ranges from that up to $22 a ton. It averages about $10 to the ton, but $100 ore and upward is occasionally encountered on the hanging wall, where nearly all the high-grade ore occurs.

On the Billy Bryan claim, nearly ½ mile from the main Gold Road shaft, the vein and the associated rocks show essentially the same characteristics and relations as in the Gold Road mine. The variations and phases of the rocks are well displayed in the large dump at the mouth of the drift. The principal rock corresponds to trachyte. It is in general considerably altered. The croppings of the vein are prominent and wall-like. The vein varies from 1 to 20 ft. in width and dips, in general, steeply to the southwest instead of normally to the northeast. It consists mostly of spar or calcite, which contains bands of greenish quartz. Beyond the southeast limits of the Billy Bryan claim, and, in fact, to the end of the Railroad claim, quartz prevails and carries good values. The quartz is greenish with glassy luster and locally is very brittle, closely crustified, wavy, and crinkled, its structure resembling the fine flow structure of certain lavas. Weed,[22] having access to later development, described the vein filling as "the usual mixture of calcite flakes and waxy quartz varying to a dense waxy yellow quartz in curly or crinkled bands and shreds sometimes showing a faint blackish stain."

In the eastern part of the Line Road claim, and in a considerable portion of the Billy Bryan and Railroad claims, the vein has produced very rich shipping ore from surface workings, and it is under these workings, notably on the Billy Bryan and Railroad claims, that the Gold Road Co. in 1908 encountered good ore in depth.

In its extension eastward through the Last Chance claim, and across the crest of the range, the vein contains some good-looking croppings and openings, but it is split by horses at several points and locally narrows to unworkable dimensions. In August, 1915, the mine was reported to have good-sized bodies of $20 ore on the 800-ft. level, and in November it is said to have encountered on the 900- and 1,000-ft. levels a body of higher-grade ore than any hitherto found on the lower levels.

Recently it is reported that the vein as stoped averages about 12 ft. in width. Three oreshoots are being stoped on the 500-ft. level and at greater depths. These are respectively 1,200 ft., 70 ft., and 700 ft. in length, being separated by small intervals of barren ground.

[22] Walter Harvey Weed: The Kingman District of Arizona, *Mining World*, vol. 32, No. 23, pp. 1113–1114 (June 4, 1910).

### *Tom Reed Mine*

*General Description.*—In the last year or two, owing to new discoveries, interest in the district has centered mainly in the Tom Reed mine and neighboring properties, wherefore this part of the district is known as the Tom Reed district or camp, Oatman district or camp, or simply Oatman. Here the geology and ore deposit are similar to those of the Gold Road camp, except that the deposits occur largely in lower geologic horizons in the green chloritic andesite and still older rocks (Fig. 3*A*). Some of the veins are "blind," being covered by later flows of rhyolite and younger rocks whose contact is traceable by the softened character of the contiguous weathered portion of the underlying rock.

The Tom Reed (formerly Blue Ridge) mine is situated at Oatman about 2 miles south of the Gold Road mine and about 200 ft. below it. It lies on open ground in Blue Ridge wash, near the base of the central part of the range, at an elevation of about 2,700 ft. (Fig. 3*A*). It is one of the well-known mines of the country and contains well-defined ore-shoots, which for nearly a decade have been worked with great profit.

It was discovered about 1900 and was soon after owned by a party composed of Ely Hilty and others. About 1901, the Gold Road Co. sunk two shafts on the property, the Ben Harrison and the Tom Reed shafts, each to a depth of about 100 ft. with good results. About 1904 the mine was purchased by the Blue Ridge Gold Mines Co., which installed a mill and operated the mine and mill for about a year and a half, milling on an average about 30 tons of $7 ore a day. In 1906 the Blue Ridge Co. was succeeded by the present owner, the Tom Reed Gold Mines Co., which resumed operations in 1907, and in 1908 the mine was reported to be working and making gold bullion shipments regularly, since when it has been a steady producer. A little later a 12-ft. wide body of $12 ore is said to have been encountered on the 300-ft. level.

The property comprises a group of 11 or more claims, adjoining one another in part end on and extending along the vein for a distance of about 3 miles. The mine is located near the middle of the group.

*Ore Deposits.*—The country rock is mainly the green chloritic andesite. The vein, the Tom Reed (formerly Blue Ridge) vein, strikes about N. 50° W. and dips about 70° NE. It nearly parallels the Gold Road vein on the north and the Victor-Virgin vein on the southwest, to both of which it is geologically and mineralogically similar. On the Tom Reed property, it has an extent of about 3 miles, Fig. 6, and it is said to be continuous with the Pasadena vein on the northwest, in which event it has a length of about 4½ miles, being probably the longest vein in the district. At the Tom Reed mine it ranges up to about 40 ft. or more in width with the fissure walls usually ill defined. It outcrops boldly for the length of nearly two claims. The croppings consist principally

of the usual dark iron and manganese-stained quartz, silicified rock, and calcite.

The vein is mainly of the quartz-adularia-calcite type. Of the early-day ore, a considerable portion is reported to have run $25 in gold to the ton for the first 30 ft. in depth and about $12 from that point down.

In the Ben Harrison shaft and its workings, the vein has a width of 16 to 22 ft. On the 100-ft. level the vein consisted chiefly of 16 ft. of crushed quartz and rock with neither wall well defined; but toward the hanging-wall side there was 6 ft. of good-looking, more or less porous, clear quartz ore.

On the 150-ft. level the vein consisted mainly of crushed rock, but the footwall side of the drift was in quartz; the hanging wall contained vugs 6 in. to 1 ft. in diameter, containing blackish, porous, oxidized quartz ore.

*Development.*—The mine is opened to a depth of 1,400 ft. It has more than 30,000 ft. of underground work, with the longest drift extending more than 4,000 ft. out from the shaft and in ore nearly all the way. The production to June, 1916, was nearly $6,000,000. That for the year ending March 31, 1916, was $661,870.68, the average value of the ore being $22.12 to the ton and the extraction 98.6 per cent. Dividends paid during the year amounted to nearly $164,000, or 18 per cent. on the par value of the outstanding stock. By estimate 11,000 tons of ore were blocked out in the stopes at the end of the year. The mine has paid more than $2,500,000 in dividends. The net realization on the mine by June, 1913, was $3,019,569.75. By June 24, 1914, the 44th dividend had been declared. Later the mine was reported to be paying for the last several years monthly dividends of from 6 to 7 per cent. on the par value of the stock, and the ore then blocked out, by estimate $2,000,000 worth, was said to be sufficient to continue their payments for several years to come. By 1907 the production considerably exceeded $120,000, and that since 1910 is more than $5,000,000. The annual production for the last few years is reported to average about $1,200,000 in bullion, besides which a large tonnage of ore is accumulating at the mine, especially of $10 ore in the workings.

Up to June, 1913, much of the ore produced had been drawn from an orebody between the third and fifth levels, where about an equal amount remained, and this same orebody had been proven to a further depth of 200 ft. below the fifth level. The seventh-level drift, at this time 233 ft. in length, was all in ore of about the same average value as that on the fifth level. Recently, in the Black Eagle section of the mine, the crosscut on the 400-ft. level is said to have passed through 35 ft. of good ore, 30 ft. of which averages $25 to the ton, and the ore tonnage on the 600-ft. level is very large.

Later, good orebodies were reported on the 500, 600, 700, 900, 1,075,

1,200 and 1,400-ft. levels. The ore on the 1,200-ft. level is said to be similar in character and grade to that on the upper levels. Progress work on the 1,400-ft. level it is said has revealed a vein width of 12 ft. with a large orebody having a known extent of 300 ft., averaging $12 to the ton and containing some high-grade ore.

Explorations on the 1,075-ft. level for 225 ft. west of the shaft have shown the oreshoot throughout that distance to have a width of about 18 ft. and to range from $22.50 to $40 to the ton. On and below this level the vein is reported to be disturbed by a fault, but at the 1,175-ft. level it has fully recovered its former size and values.

*Character of Ore.*—The ore is similar to that of the Gold Road mine. It consists of a mixture of flaky calcite, waxy quartz, adularia, brecciated altered rock, and pinkish argillaceous material which is frequently of high grade. According to Weed,[23] a dense quartz whose color and luster closely resembles that of beeswax constitutes the richest ore. The ore is not hard and most of the gold is free, especially in the ore from the lower levels, but it requires fine grinding to free the finely disseminated gold and expose it to the action of the cyanide. The gold is seldom visible even in rich ore.

In milling, the total sliming system of cyanidation is employed, followed by treatment of the ore in Pachuca vats and Dorr thickeners. "Dorr thickeners," according to Smith, "appear to be particularly suited to conditions in this district where little silver is present, weak solutions are used, and the slime settles quickly." The ore amalgamates about 50 per cent. of its value on the plates and a high extraction is obtained at reasonable cost by cyanidation. In 1910, the average extraction was $42.46 to the ton. In 1912 it was $23.22. The amount of ore treated in 1911 was 39,447 tons; in 1913, 948,110 tons, with an average extraction of $24.09, and a recovery of 97.05 per cent. The average value of the ore mined in the fiscal year 1914 to 1915 is $21 to the ton. The gold is generally pure, the proportion of silver present being very small. The mill treats about 4,000 tons of ore a month.

### *Other Mines*

Among the new properties which are attracting most attention is the United Eastern which adjoins the Tom Reed mine on the northwest and is often referred to as the "Bonanza." It is but a year and a half old, and is reported to have in sight, according to the estimates of conservative mining engineers, $11,000,000 worth of $26 ore. The mine contains more than 2,000 ft. of drift and has good orebodies on all levels between the depths of 300 and 700 ft. The vein is reported to be 43 ft. wide on the 555-ft. and 665-ft. levels, and the entire width is pay milling ore. Of this

[23] *Op. cit.*

width, 30 ft. proven for the distance of 650 ft. averages $40 to the ton and carries considerable free gold. Since August, 1915, daily shipments of $30 ore removed during development are being made to the Gold Road mill. A new 200-ton cyaniding plant, in which gyratory crushers and ball mills instead of stamps will be used, is being installed at the mine. The equipment will be adequate for sinking to the depth of 2,000 ft.

Many other new properties like the United Eastern are being opened up and in many of them good ore is being found at depths of from 200 to 500 ft. A score or more are worked by incorporated mining companies.

Extensive developments are being undertaken at the Pioneer (formerly German-American) mine by the Oatman Pioneer Mines Co. which by coöperation and use of its efficient machinery and 400-ft. level will immediately facilitate the exploration and working of adjoining properties. The Pioneer is on one of the three main lode outcrops of the district, and is said to have $1,000,000 worth of commercial ore in sight between the 400-ft. level and the surface. It is working on two veins of the 400-ft. level, of which the northeast, or Pioneer vein, has an 8-ft. oreshoot having a known extent of 600 ft. and averaging $16 to the ton, and on the southwest, or Snowball vein, the crosscut has penetrated a width of 16 ft. of good-grade ore with the outer wall not yet reached. On the 200-ft. level, a 2-ft. shoot of high-grade ore is being worked.

The Boundary Cone mine adjoining the Pioneer on the east has good ore on the 750-ft. level, and is said to contain a 5-ft. shoot of ore that averages $100 to the ton on the 200-ft. level. On the 500-ft. level, where the oreshoot has been proved for a distance of 90 ft., it has a width of 12 ft. and averages about $20 to the ton. The mine is credited with a considerable production of high-grade ore, some of which contained crystallized gold.[24]

In the Big Jim mine, a mile northwest of Oatman, the vein on the 400-ft. level is said to be 51 ft. in width, of which 46 ft. averages about $8 to the ton and 8 ft., on the hanging-wall side, $15 to the ton with some pay streaks which are very rich. On the 485-ft. level, the same oreshoot has been opened for the extent of 200 ft. and is good milling ore, most of which averages more than $12 to the ton. Here the ore is said to be more oxidized and silicified than on the upper levels. The vein parallels the projected course of the Tom Reed vein, but whether it is the northwestward extension of the Tom Reed vein which may here be faulted to the northeast, or a new vein, has not yet been determined. The mine is daily shipping about 30 tons of ore removed in development.

In the Carter mine, ½ mile south of Oatman, the main oreshoot, about 15 ft. in width descending from the 150-ft. level to the 400-ft. level, is said to contain 5 ft. of ore which averages about $30 to the ton, and 8

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[24] W. P. DeWolf: The Tom Reed-Gold Road, *Salt Lake Mining Review*, vol. 17, No. 13, p. 16 (Oct., 1915).

ft. that averages $8 to the ton. This property is said to have shipped, several years ago, some very rich ore taken from between the surface and the 150-ft. level. This same vein is thought to extend through the adjoining Telluride and Lucky Boy properties, where on the 300-ft. level the entire width of 25 ft. is good milling ore.

The Gold Reed mine, a mile south of Oatman, has 4 ft. of $32 ore on the 375-ft. level.

On the Times property, the vein recently opened at 270 ft. in from the portal of the Martin tunnel, reveals 5 ft. of ore, averaging nearly $24 to the ton.

On the north, the Gold Ore mine, ½ mile northeast of Gold Road, is credited with a 9-ft. vein on the 500- and 550-ft. levels containing a 6-ft. shoot of ore which for the distance of 300 ft. averages about $12 to the ton, and a considerable portion of it nearly $100 to the ton. More than 75,000 tons of ore are said to be blocked out above the 550-ft. level. Daily shipments of ore averaging nearly $25 to the ton are made to the Gold Road mill.

The old Moss mine, where the original discovery of mineral in the Mohave County region was made, is being developed by the owners of the Gold Road mine and $60 ore is being mined from the 200-ft. level.

In the Ivanhoe mine, 2 miles northwest of Oatman, the vein, whose footwall is a partially mineralized 75-ft. "quartz porphyry dike cutting andesite and underlying sedimentaries," on the 250-ft. level, has a width of 60 ft., and on the 500-ft. level 6 ft. of milling ore has just been crosscut on the footwall side. Some high-grade ore has been shipped to the Gold Road mill.

Attention is also being attracted to the Secret Pass district, 6 miles north of Gold Road, which with a small mill is making considerable production mostly from high-grade surface ore. Here the occurrence is unusual, the ore, according to Payne,[25] being found chiefly as replacement deposits in the rock walls of a fissure occupied by a dike which the ore seems to postdate. The bullion, which is shipped to the U. S. Mint at San Francisco, is said to average about $15 to the ounce in gold. The population during the last few months of 1915 increased from 100 to 400, with prospecting extending over an area of several square miles.

At about 6 miles south of Oatman, promising deposits are reported in the Black Range zone, where a dozen companies are operating. Concerning the rocks in this part of the field, which have been roughly grouped with the undifferentiated volcanics, little is known as yet.

The deposits occur in a series of veins of which the one on which the principal mines are located is a prominent quartz outcrop known as the Nellie vein. It has been traced for the distance of several thousand feet and opened to the depth of 300 ft. on the Black Range, Nellie and Green

[25] C. Q. Payne: Oral communication.

Quartz properties. Associated with the more pronounced quartz replacement phase of the vein on the Black Range and Nellie ground, are said to be rich streaks of ore that show coarse gold when panned. The Black Range mine is said to have milling-grade ore of irregular occurrence on the 300-ft. level, of which 3 ft. averages about $30 to the ton. Here, also, the country wall rock extending along the vein, for a width of 60 ft. or more, is impregnated with replacement deposits and averages nearly $6 in gold to the ton.

In referring to the future of the Tom Reed-Gold Road district, some men favorably compare it with Goldfield, Cripple Creek or other large camps and hold that it will become one of the greatest gold-producing districts in the United States. Among the more conservative and seemingly reasonable views is that of Palmer,[26] who believes that it will become comparable with the Tintic district, Utah, that it will become a large low-grade camp with several producing mines which will yield dividends for many years to come.

### *Fields Similar to the Tom Reed-Gold Road District*

Recent investigations[27] have shown that the southern end of the Black Mountains containing the Tom Reed-Gold Road district is the easterly one of a number of similar volcanic areas which extend interruptedly westward on either side of the railroad through the distance of nearly 100 miles to the longitude of Barstow, Cal. In these areas—which embrace the Mohave and Chemehuevis Mountains, the ranges west of Von Triger, Clipper Mountain, the Cady Mountains, the Newberry-Ord district, and the well-known Calico Mountain group and others—the geological and mineralogical conditions are very similar to what they are in the Tom Reed-Gold Road district. The areas lie from a few miles to 25 or 30 miles back from the railroad. Their volcanic rocks, which in character, recurrence, and succession, are in general identical with those in the Tom Reed-Gold Road district, range up to 2,000 ft. or more in thickness, are frequently well-mineralized, and contain strong veins. Most of the areas have been but little prospected, but some, as that of the Calico Mountain group, are productive.

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[26] *Op. cit.*, p. 900.

[27] N. H. Darton and others: Guidebook of the Western United States, Part C, The Santa Fe Route, *Bulletin No.* 613, *U. S. Geological Survey*, pp. 142 to 162 and sheets 21, 22, and 23 (1915).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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# The Influence of the Movement in Shales on the Area of Oil Production

BY RICHARD A. CONKLING,* A. B., TULSA, OKLA.

(New York Meeting, February, 1917)

A SHALE layer, buried beneath two or three thousand feet of strata, in some instances, will upon folding become thicker in the synclines and thinner on top of the anticlines.

This can be accounted for, in part, by the stretching on the crests of the fold and the compressing in the troughs; but this will by no means account for all of it, as is shown in the example herein set forth.

It is my firm belief that the rest of the thickness is due to flow. What causes the movement, however, does not concern us here, so long as there is movement, for this article purposes to show its effect upon the producing area of different sands.

It is the author's hope, in this way, to drop a hint or two that may be valuable to the oil geologist, in making estimates of future productions and values of oil properties. An example will be given from property in the famous Cushing Pool in northeastern Oklahoma, where the author has had occasion to make a detailed study before recommending some property to his company, and has then been able to watch the results of this work.

The author never mapped the surface structure in this pool, as this had been done by Frank Buttram when he was with the Oklahoma Geological Survey. We are using, therefore, a section of his map in this article for our surface contours. The main east dip comes farther east than this section shows.

From the 10-ft. contours in Fig. 1, it will be seen that the surface folding is very slight, but in the contour map of the Bartlesville sand (Fig. 2), which was the greatest producing sand, we see much more complex folding. It seems that the folding becomes greater with the depth. Folding may have started in Middle Pennsylvanian times and continued through the Upper Pennsylvanian which is found at the surface. This may, or may not, have been the cause of the increase of folding with depth. However, the fact that mostly concerns us is the interval of shale between the Bartlesville sand and the Tucker sand. The Tucker is the next productive sand below the Bartlesville. Not enough wells have been drilled to the Tucker sand to make a contour map of it, but

* Head Geologist, Roxana Petroleum Co.

the figures beside the wells on Fig. 2 show the interval between the top of the Bartlesville and the top of the Tucker.

The entire area shown in Fig. 2, except the southeast corner, was productive in the Bartlesville sand. On the west edge of Section 10, a syncline will be observed. Water was encountered in this syncline at the bottom of the Bartlesville, but it did not rise high enough to ruin the wells for some time. The producing area of the Tucker sand is much smaller, however, as the surface of the sand dips more steeply

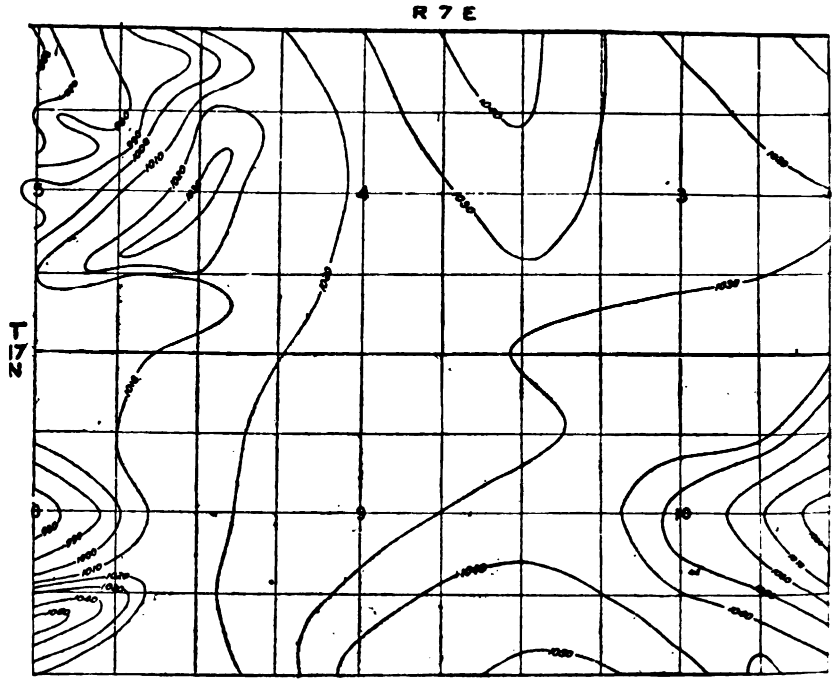


FIG. 1.—SECTION OF CUSHING FIELD. CONTOURS ON PAWHUSKA LIMESTONE.

and goes more rapidly into water. It will be seen from the figures on the north line of Sections 8 and 9, and from the cross-section (Fig. 3) taken along that line, that the Tucker sand dips almost twice as steeply as the Bartlesville sand. The latter dips 70 ft. from Well No. 12, Maley Yarhola, to Well No. 4 of the Gypsy Oil Co., 1/4 mile east, while the former dips 123 ft. in the same distance. The difference is entirely due to the thickening of the shale interval between the two sands.

The producing area of the Tucker sand is, therefore, only a little over 1/4 mile across. On top of this narrow anticline, however, was drilled the biggest producer of light oil known to the writer, namely, the No. 11 Jackson Barnett of the Gypsy Oil Co. of Oklahoma, which came in at 14,000 bbl. per day.

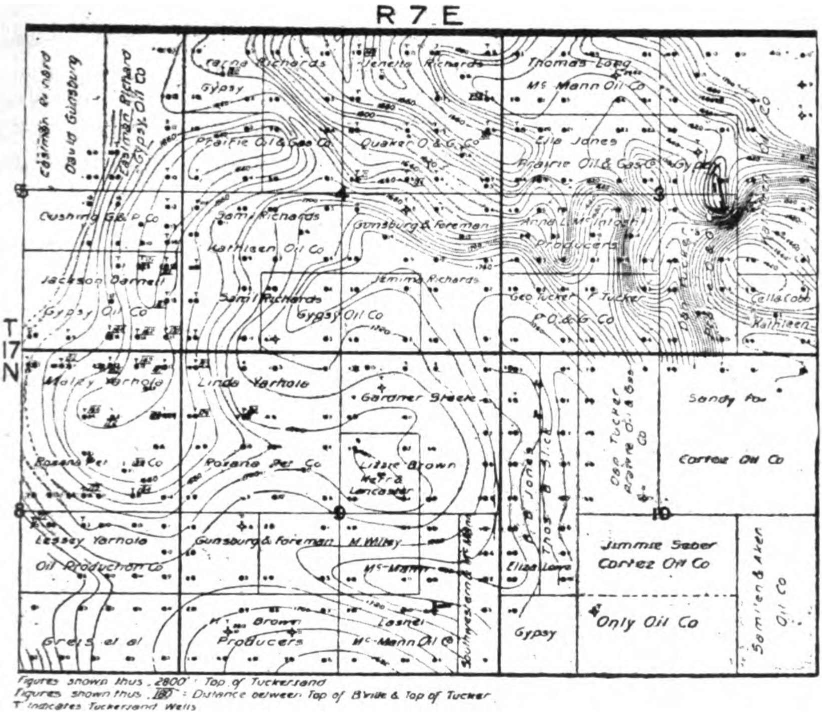


**Fig. 2.—Section of Cushing Field. Contours on Bartlesville Sand.**

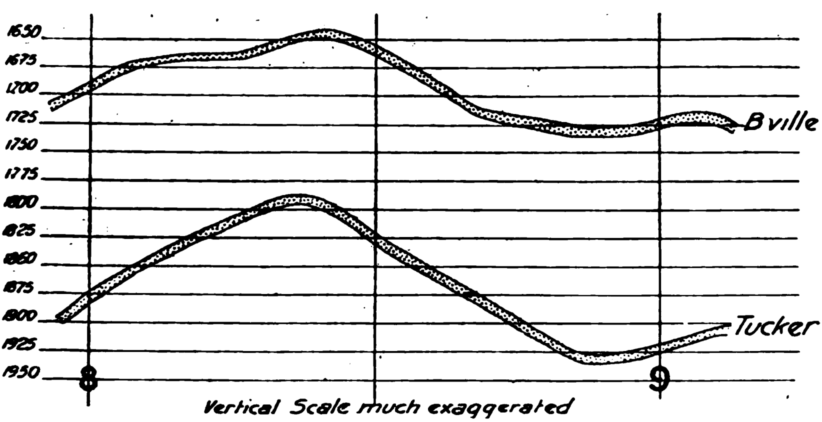


**Fig. 3.—Section Showing Top of Bartlesville and Tucker Sand.**

After a few wells had been drilled to this sand, the exact limit of production was figured out, and our company is drilling accordingly. By taking this thickening into account, millions of dollars were made for our company on purchasing of land.

It will be seen that the same is true of the other domes on the maps. Of course these details cannot be told from the surface, but if one recognizes the conditions that these facts set forth, one need not go far astray when estimating the possible deeper-sand production in a developed field.

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# The Genesis of Asbestos and Asbestiform Minerals*

BY STEPHEN TABER, PH. D.,† COLUMBIA, S. C.

(New York Meeting, February, 1917)

CONTENTS



## Introduction

The term asbestos, as commonly used, includes half a dozen minerals all having a well-developed fibrous structure, but differing in chemical composition and in some of their physical properties. In its strict application the name is limited to the fibrous varieties of the monoclinic amphiboles. Commercially, however, the most important of the asbestiform minerals is chrysotile, a fibrous variety of serpentine. About 95 per cent. of the asbestos used in manufacturing is chrysotile, and it commands a much higher price than any of the other fibrous minerals now on the market.

Although the production of asbestos has increased rapidly in recent years, comparatively little has been published concerning its origin. The present paper is preliminary in its nature, and therefore does not pretend to exhaust the subject. The ideas herein developed are the result of field investigation, laboratory experiments in the growth of fibrous

* Manuscript received by the Secretary on June 26, 1916.

† State Geologist and Professor of Geology at the University of South Carolina.

crystals, and the examination of asbestos specimens from the more important producing districts, as well as a careful study of the work of previous investigators.

## Asbestiform Minerals

### *Chemical and Mineralogical Relations*

The table given below contains the group name and species of the various fibrous minerals to which the term asbestos is commonly applied. The name of each mineral is accompanied by its chemical formula as given in Dana's *System of Mineralogy*.

Asbestiform Minerals

- Amphibole Group
  - Orthorhombic Section
    - Anthophyllite $(Mg,Fe)SiO_3$
  - Monoclinic Section
    - Tremolite $CaMg_3(SiO_3)_4$
    - Actinolite $Ca(Mg,Fe)_3(SiO_3)_4$
    - Crocidolite $NaFe(SiO_3)_2.FeSiO_3$
- Serpentine $H_4Mg_3Si_2O_9$
    - Chrysotile
    - Picrolite

All of the minerals listed in the table have both fibrous and non-fibrous varieties. The fibrous varieties never show the outward form of crystals except as pseudomorphs, but the fact that they are crystalline is proved by their optical properties and other evidences of regular internal molecular structure. Well-developed euhedral crystals of tremolite and actinolite are quite common, especially where these minerals have developed in metamorphic limestones or have grown in cavities. Euhedral crystals of anthophyllite are very rare, they have never been reported for crocidolite, and serpentine has no crystal form of its own, though frequently occurring as a pseudomorph after other minerals.

Tremolite and actinolite usually assume more or less elongated prismatic forms. Often the crystals occur in aggregates of parallel prisms or in groups having a radial arrangement. There is every gradation in structure from the normal prismatic forms through the slender and columnar into the fibrous. The habit of anthophyllite is similar, except that radiating groups are relatively more abundant and the structure is more inclined to be fibrous. Crocidolite is usually fibrous but is sometimes massive or earthy.[1] Serpentine is generally massive in appearance, frequently fibrous, and sometimes foliated. Examined under the micro-

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[1] E. S. Dana: *A System of Mineralogy*, Ed. 6, p. 400 (1914).

scope, even the massive varieties commonly show a more or less finely fibrous structure, though occasionally the serpentine is in the form of minute scales. The amphiboles possess a well-developed prismatic cleavage intersecting at angles of 54° to 56°, while the chrysotile variety of serpentine is said to have a prismatic cleavage of 50°.

In chemical composition these minerals are characterized by the absence of aluminum and by the presence of magnesium in all except crocidolite, which contains instead a notable percentage of sodium. Tremolite and actinolite are distinguished by the presence of calcium as an essential element. The fibrous amphiboles are all normal metasilicates, and serpentine is a hydrous orthosilicate. These minerals are all secondary, being formed from other minerals, and for the most part they are confined to metamorphic rocks, which have had in some instances an igneous and in others a sedimentary origin.

### *Physical Properties*

The commercial value of asbestos and the uses to which the different varieties may be put are dependent upon the physical properties; fineness, length and flexibility of fiber, tensile strength, and heat- and acid-resisting properties.

All of the different varieties of asbestos may be split up into exceedingly fine fibers, and when the finest obtainable fibers, having a diameter of 0.002 mm. or less, are examined under the microscope, they are usually seen to be made up of still smaller fibers. The number and fineness of the smallest fibers distinguishable under the microscope increase with every increase in the power of magnification, and there is no apparent limit to this subdivision.

Bunches of amphibole asbestos, of the slip-fiber type, are sometimes found with a length of 2 or 3 ft., but when these bunches are split up into smaller and smaller bundles, the length of the bundles tends to decrease with their diameter. This indicates that the long bundles probably consist of shorter interlaced and overlapping bundles of fibers. In chrysotile veins the fibers seldom have a length of over 2 or 3 in., and the bulk of the material mined averages less than ½ in. The longer fibers are commonly cut by planes of jointing, sometimes almost invisible, and sometimes marked by the presence of foreign material.

Chrysotile fibers are the most flexible, but fibers of crocidolite are almost if not equally as good in this respect. The other varieties of amphibole asbestos are, on the whole, less flexible, and, while this property varies greatly in each of the different species, the fibers of anthophyllite seem to be somewhat more flexible than similar fibers of tremolite or actinolite. Picrolite is similar to chrysotile except that the fibers are coarser and more brittle. The latter mineral is of no commercial im-

portance. Little is known concerning the tensile strength of the different species of asbestos, but the experiments of Haussman[2] indicate that crocidolite has the greatest tensile strength, and chrysotile probably ranks second.

Chrysotile easily withstands temperatures of 2,000° to 3,000° F., while with some varieties a temperature of 5,000° F., produces no visible effect.[3] At red heat it gives up water and becomes brittle. Anthophyllite under the same conditions remains practically unaltered. Tremolite and actinolite fuse at somewhat lower temperatures, while crocidolite fuses so easily that it is useless for many purposes where asbestos is commonly employed.

Chrysotile is readily attacked by relatively weak acids, being decomposed with the separation of silica; the amphibole varieties, especially tremolite and anthophyllite, are very resistant even when subjected to the action of concentrated acids.

## Types of Asbestos

Three types of asbestos fiber are recognized—cross-fiber, slip-fiber and mass-fiber. Cross-fiber asbestos occurs in veins with the fibers extending transverse to the strike of the vein. Usually the fibers are approximately perpendicular to the inclosing walls, frequently they are more or less oblique and occasionally they are curved or abruptly bent. Slip-fiber is found along fault planes, often accompanied by slickensides, and the direction of the parallel fibers records the direction of displacement. The amount of displacement is usually small. Slip-fiber asbestos is commonly distributed in thin layers that are not as a rule continuous for any considerable distance, but occasionally it is found in masses a foot or more in thickness. All gradations are to be found between the cross-fiber and slip-fiber types. Mass-fiber asbestos occurs in fibrous bundles or groups varying in size and orientation. The fibers may be parallel but are usually divergent and often radiating. In the type occurrence at Sall Mountain, Georgia, mass-fiber asbestos makes up practically the entire rock mass. Gradations between the mass-fiber and slip-fiber types probably exist, although, so far as the writer is aware, none has been described.

Anthophyllite has been reported as occurring in all three ways, and it is the only variety of asbestos known to occur as mass-fiber. The asbestiform varieties of tremolite and actinolite are practically limited to the slip-fiber type although two occurrences of true asbestos in the

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[2] J. F. L. Haussman: *Handbuch der Mineralogie*, p. 734 (1847).

[3] Fritz Cirkel: Chrysotile-Asbestos, Its Occurrence, Exploitation, Milling and Uses, Ed. 2., *Canada Department of Mines, Mines Branch, Report* No. 69, p. 30 (1910).

form of cross-fiber have been described.[4] Crocidolite has been reported only in cross-fiber veins, but there is every reason for believing that it may be found also as slip-fiber. Very little is known as yet concerning the occurrence of crocidolite. Most of the commercially important deposits of chrysotile are of the cross-fiber type, but slip-fiber is also very common.

## Origin of the Fibrous Structure

### *Views of Previous Investigators*

The wonderfully fibrous structure of the asbestos minerals has aroused interest and curiosity ever since they were first discovered, and yet little by way of explanation of this peculiar structure is to be found in geologic literature. Why these minerals sometimes possess an asbestiform structure and sometimes do not is a question on which few geologists have advanced an opinion.

Merrill[5] expresses the belief that the fibrous structure in the case of anthophyllite "and of the true asbestos as well, is due, in many instances at least, to a process of shearing—is, in fact, an exaggerated form of the process of uralitization."

Heddle[6] and Merrill[7] have described instances in which the fibrous structure of anthophyllite has been accentuated by weathering. Hopkins[8] apparently adopts this view in part for the deposits at Sall Mountain, Georgia.

Heddle[9] argued that chrysotile must be pseudomorphic, since serpentine is a mineral incapable of assuming a chrystalline form, but, after describing the chrysotile found near "Hesta Ness, which terminates the south side of the bay of Gruting, in Fetlar, Shetland," he suggests that "As the magnetite here is itself saturated with a serpentinous basis, there is a possibility that the fibrous structure of the chrysotile may be the result of its protrusion, by an exfiltration process, through the interstices of the granular magnetite, which interstices acted upon the mineral in a manner similar to the holes in a draw-plate."

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[4] M. F. Heddle: Chapters on the Mineralogy of Scotland, *Transactions of the Royal Society of Edinburgh*, vol. 28, pp. 503–504 and 531 (1877–78). Also quoted by G. P. Merrill in Notes on Asbestos and Asbestiform Minerals, *Proceedings of the U. S. National Museum*, vol. 18, pp. 286–287 (1895).

[5] G. P. Merrill: Notes on Asbestos and Asbestiform Minerals, *Proceedings of the U. S. National Museum*, vol. 18, p. 289 (1895).

[6] M. F. Heddle: *Op. cit.*, pp. 502 and 504.

[7] *Op. cit.*, p. 288.

[8] O. B. Hopkins: Report on the Asbestos, Talc and Soapstone Deposits of Georgia. *Geological Survey of Georgia*, *Bulletin* No. 29, p. 106 (1914).

[9] M. F. Heddle: *Op. cit.*, pp. 535–536.

Dana,[10] in his *Systematic Mineralogy*, gave the following explanation of the origin of fibrous structure: "When a solution is spread thinly over a large surface, minute crystalline points encrust the whole, and if the solution be gradually supplied, as crystallization goes on, it is obvious that the minute points may elongate into crowded prisms of fibres, producing a fibrous structure. Such a structure is common in narrow seams in rocks, and the fibres are usually elongated across the seam."

Cirkel[11] quotes this explanation and then adds: "It seems that this paragraph refers to chrysotile; because the crystals of the same are elongated across the seam in the precise mode described." Dana's hypothesis is probably the correct explanation of the fibrous structure seen in the stalactitic, mammillary, and similar forms that have been deposited in open spaces from a thin layer of solution, but there is no evidence that any of the varieties of asbestos have been formed in this way.

The hypothesis advanced by Cirkel[12] is that, "chrysotile may have been an extreme example of crystallization which took place under conditions of high temperature and extreme pressure."

While the writer has not had the opportunity of making an exhaustive search of the literature, he believes that the various explanations of fibrous structure listed above include the more important hypotheses so far advanced by scientific investigators.

## *Discussion of the Evidence*

Microscopic examination shows that bundles consisting of thousands of small fibers of chrysotile or amphibole asbestos behave as crystal units, an entire bundle exhibiting the same optical properties as any one of the component fibers. The fact that all of the asbestiform minerals have prismatic cleavage suggests that the fibrous structure may be due to an abnormal development of the cleavage or at least that the separation of the fibers takes place along cleavage planes. This hypothesis is supported by the prismatic form shown by most of the smallest obtainable fibers of amphibole asbestos, when they are examined under the microscope, but fibers of chrysotile usually possess irregular polygonal or rounded cross-sections. The highly fibrous structure of the asbestiform minerals is not, however, a crystallization phenomenon in the sense that it is due solely to the inherent physical properties of the crystal molecule, for all minerals having asbestiform varieties occur in non-fibrous as well as fibrous forms. Moreover, there are intermediate gradations between

[10] J. D. Dana: *Systematic Mineralogy: Section II, Theoretical Crystallogeny*, p. 124 (1850).

[11] Fritz Cirkel: *Op. cit.*, p. 93.

[12] *Loc. cit.*, p. 93.

the non-fibrous and the asbestiform varieties of serpentine and of the amphiboles with the possible exception of crocidolite.

Why is it, then, that these minerals sometimes have a fibrous or asbestiform structure, while at other times they are non-fibrous? The answer to this question, the writer believes, is to be found chiefly in the physical conditions under which the minerals were formed.

The primary minerals of igneous rocks never show an asbestiform structure even in the case of minerals possessing perfect prismatic cleavage, and this statement is true in general for all minerals that have grown in free contact with supersaturated solutions. The latter generalization will be objected to by those who believe that cross-fiber veins have been deposited from solutions circulating along open fractures, but the writer has found no evidence supporting such an hypothesis. This question, however, is later discussed in detail under the origin of cross-fiber veins.

There are many minerals that have fibrous varieties. Most of them do not have a prismatic cleavage, some have no cleavage at all, and at least one such mineral, limonite, is always amorphous and never crystalline. The fibrous varieties of these minerals differ from asbestos chiefly in having fibers that are coarser, more brittle and not so easily separable. With fibers of equal size the flexibility and tensile strength are probably determined very largely by chemical composition.

In the case of those minerals that do not have a prismatic cleavage and do not normally have a columnar or prismatic habit, the fibrous structure is obviously due to physical conditions which have prevented crystal growth except in one direction. If a mineral having perfect prismatic habit and cleavage develops under physical conditions that limit growth to a direction parallel to the principal axis, then the fibrous structure may be accentuated to such an extent as to make the mineral truly asbestiform. A careful comparison of the occurrences of all the common fibrous minerals indicates that the commercially important deposits of the asbestiform minerals have been formed under the same physical conditions that result in the development of a well-defined fibrous structure in other minerals.

That the peculiar structure of asbestiform minerals is usually due to the accentuation of a normal prismatic habit and cleavage through the limitation of crystal growth by physical conditions is the author's thesis. Recently he conducted a series of laboratory experiments with the object of determining the essential conditions for the development of fibrous crystals. Some of these experiments have been referred to in another paper,[13] and it is planned to publish a detailed account of the others in the near future. All questions connected with the problem have not been

[13] Stephen Taber: The Growth of Crystals under External Pressure, *American Journal of Science*, Ser. 4, vol. 41, pp. 532–556 (1916).

cleared up, but some of the facts established have an important bearing on the present discussion.

### *Conclusions*

The shape of a growing crystal is controlled by one or more of three independent factors, namely: (1) the tendency to assume a regular polyhedral form because of the forces of surface tension and molecular orientation; (2) the relative and absolute magnitude of the external forces resisting growth in different directions; and (3) the accessibility of the material from which the crystal is built.

1. Some crystalline substances normally have a prismatic or columnar habit because of the intermolecular forces controlling the development of crystal faces, and under favorable conditions slender acicular or hair-like crystals may result; but a highly fibrous or asbestiform structure is never developed even in the case of minerals having perfect prismatic cleavage, without the assistance of one of the other two factors.

2. If crystals are under unequal pressure, growth may be limited to the direction of least pressure. When crystals grow through the addition of new material from solutions, growth takes place only where the solutions are supersaturated with respect to the crystal surfaces with which they are in contact, and growth must continue regardless of resisting forces as long as supersaturation is maintained. But pressure increases the solubility of all substances that go into solution with decrease in volume, and therefore any increase in pressure must be accompanied by a corresponding increase in the concentration of the solution, if growth is to continue. Crystals subjected to unequal pressure may even go into solution on the surfaces that are under the greater pressure, while at the same time deposition is taking place along the line of least pressure; and if the normal habit of the crystals is columnar, then those crystals that are oriented with their longer axes parallel to the least pressure will tend to grow at the expense of those that are less favorably oriented.

In the same way unequal pressure may control the direction of growth and therefore the shape of crystals, when crystallization from a state of fusion takes place, with increase in volume. There are, however, only a few substances, such as ice, that solidify with expansion in volume. Unequal pressure may also determine the shape of secondary minerals formed without going into solution, for, when the alteration of one mineral to another is accompanied by increase in volume, pressure tends to prevent the alteration, and therefore the effect of unequal pressure may be to limit alteration and the growth of the new mineral to the direction of least pressure.

Unequal pressure always sets up shearing stresses, and in the case of cleavable minerals these stresses are relieved most easily by slipping

along cleavage planes, thus making the cleavage more pronounced. The effect of shearing on minerals with prismatic cleavage may therefore be a factor of some importance in producing a fibrous structure.

The development of fibrous structure is undoubtedly in some instances to be attributed to growth under unequal pressure, but this is not the commonest cause of the phenomenon.

3. The study of fibrous minerals and their occurrence in nature, as well as the results of laboratory experiments in the growth of fibrous crystals, leads to the conclusion that in most cases the development of fibrous structure has been due primarily to the fact that the material from which the growing crystals were built was accessible only in one direction. This is the most probable reason why secondary minerals are so frequently fibrous. For example, when serpentine is formed from olivine, the alteration sets in from the exterior of the crystal and from cracks, and the resulting serpentine is usually in the form of microscopic fibers that develop normal to these surfaces. The fibrous form is gradually assumed by the microscopic crystals, because growth takes place only at their base where they may receive additions of new material as alteration of the olivine progresses. In a partly altered olivine the microscopic veinlets of serpentine are frequently similar in appearance and structure to the larger veins of cross-fiber chrysotile.

The efficacy of this method of producing fibrous structure in crystals, when growth takes place through the addition of new material from supersaturated solutions, has been proved experimentally.[14] Cups of porous porcelain were partly immersed in concentrated solutions of copper sulphate, alum and other salts. The solutions were drawn up through the capillary pores allowing evaporation to take place from the exposed surfaces. After a day or two, crystal growth began with the formation of irregular spots or thin crusts on the upper surfaces of the cups, and these gradually increased in size and thickness. Later, groups of fibrous crystals could be observed under the crusts slowly pushing them outward. The fibers continually increased in length as long as the material for growth was available, and at the end of 8 months some were over 2 cm. in length.

In these experiments crystallization also occurred at a few favorable places within the walls of the cups, and, as growth continued, the crystals developed sufficient pressure to produce rupture. The fractures thus formed were gradually extended and widened by the growth of fibrous veins, closely resembling in structure the cross-fiber veins of chrysotile and of asbestos, as well as the similar veins of fibrous calcite, gypsum and other minerals. In some of the experiments where alum was used, it was possible to change the color of the solution by adding chrome alum in varying amounts, and thus produce banded veins.

[14] Stephen Taber: *Op. cit.*, pp. 545–546.

Cavities or open spaces were usually present under the central portions of the larger crusts and within the veins, and these openings were sometimes partly lined with crystals normal in habit instead of fibrous. This lack of uniformity is to be explained by the fact that supersaturation of the solution was produced and maintained by evaporation, and this process was retarded or prevented where the crusts protected the solution from contact with the air. Better results were obtained when supersaturation was induced by cooling, so that practically uniform concentration could be obtained over the entire crystallizing surface.

The experiments so far conducted by the writer indicate that fibrous crystals may be produced in the manner described above only from substances that go into solution with decrease in volume.[15] Most, if not all, of the rock-forming minerals belong to this class.

The development of the fibrous structure is probably aided by small adjustments or slips along the surfaces of the fiber prisms and cleavage planes when the latter are present, since under the conditions of growth it is not conceivable that new material can be added at the end of each prism of a large group continuously and at the same uniform rate. Indeed observation shows that such slipping does take place in the groups of crystals grown in the laboratory, and both megascopic and microscopic examination of fibrous minerals furnishes evidence of similar slipping.

The fibers obtained in the experiments just described are brittle and therefore rather difficultly separable, but occasionally it is possible to procure individual fibers having a length of several millimeters and a thickness of less than 0.001 mm. Such fibers of copper sulphate and alum are flexible and somewhat elastic. The columnar or fibrous ice crystals found in clayey soils grow in practically the same way as the fibrous crystals described above, and Merrill has observed the growth of fibrous incrustations of gypsum on the walls of caves,[16] but did not attribute their fibrous structure to the manner of growth.

## Origin of Cross-fiber Veins

Many different theories, some of them rather hypothetical, have been advanced by geologists to explain the origin of veins of cross-fiber asbestos. These theories have been reviewed recently in three publica-

[15] Attempts were made to obtain similar results with ammonium chloride and other salts that go into solution with expansion in volume, but no veins were formed and the crusts were enlarged only through the addition of new material on the outer exposed surfaces, the solutions reaching these surfaces through capillary pores in the crystalline mass. In some instances tubular, hair-like crystals were formed, 0.01 mm. or less in diameter, and these apparently grew only at their outer ends, the new material being furnished by solutions drawn up through the capillary tubes.

[16] G. P. Merrill: On the Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings of the U. S. National Museum*, vol. 17, pp. 77–81 (1894).

tions, and it is not necessary, therefore, to give them in detail here. That no general agreement has been reached, is shown by the diversity of opinion expressed in the conclusions drawn by the authors of the three monographs just cited.[17] Most of the theories are limited to chrysotile veins in serpentine and could not be applied to veins of crocidolite and other varieties of amphibole asbestos. Since the origin of chrysotile is closely related to the origin of the massive serpentine with which it is associated, the origin of chrysotile veins will be considered séparately.

### *Chrysotile Veins*

All of the different theories attempting to explain the origin of chrysotile veins fall under one or the other of the following classes:

1. The veins were deposited in open fissures: (*a*) by circulating solutions; or (*b*) through the segregation of material; or (*c*) infiltration of serpentinous solutions from the walls.

2. The veins were formed by the replacement of the wall rock along small fissures that served as channels for the circulation of solution.

3. The veins are portions of the serpentine that have crystallized *in situ*, the crystals growing outward from preëxisting fractures through which water entered to alter the rock to serpentine.

All of these theories presuppose the presence of fractures, and much ingenuity has been used in accounting for the fractures. They have been attributed to: (1) the contraction of an igneous magma upon cooling and solidifying; (2) dynamic causes; (3) exfoliation, possibly aided by the increase in volume that accompanies the alteration of peridotite to serpentine; (4) shrinkage resulting from the loss of silica or other constituent; and (5) a partial dehydration of the serpentine.

It seems to the writer that there are serious objections to all of the theories outlined above, for, while some of them may explain many of the phenomena connected with the occurrence of asbestos veins, not one offers a complete and adequate explanation of all of the known facts. It is conceivable that some chrysotile veins may have been formed in open fissures, but it is mechanically impossible that all of them could have been formed in this way. Dresser[18] has pointed out the absurdity of this theory as applied to the chrysotile deposits of southern Quebec, Canada, where the veins, in places occupying over 10 per cent. of the entire rock, run in all directions from vertical to horizontal, and occasionally reach

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[17] Fritz Cirkel: *Op. cit.*

J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey, Memoir* No. 22 (1913).

O. B. Hopkins: *Op. cit.*

[18] J. A. Dresser: *Op. cit.*, p. 65.

a length as great as 100 ft. Pratt[19] and Diller[20] have described chrysotile veins in the Grand Canyon, Arizona, over 4,000 ft. below the rim, which extend horizontally parallel to the bedding of the inclosing rocks for distances of 150 ft. or more.

Moreover, there is evidence, in many instances at least, that the formation of the chrysotile veins and the alteration of the inclosing rock to form serpentine are contemporaneous processes. In the Black Lake district of Quebec, where the alteration of peridotite to serpentine is not complete, the chrysotile veins are bordered on both sides by bands of massive serpentine, and the width of the veins is proportional to the width of the inclosing bands. By careful measurements Dresser[21] found the ratio of the chrysotile vein to the entire band of serpentine to be 1:6.6. The alteration of peridotite to serpentine is accompanied by such an increase in the volume of the rock that it would be impossible for fissures to remain open during the process.

Little if any evidence has been offered in support of the replacement hypothesis. Where chrysotile veins occur in limestone, as they do in Arizona, it is possible for some replacement of the inclosing rock to take place, but this occurrence is exceptional. Nearly all chrysotile veins are found in massive serpentine, having a chemical composition that is practically identical with that of the vein material, and no reason has yet been given to explain why serpentine should replace serpentine of the same chemical composition. Moreover, chrysotile veins do not possess any of the characteristics that commonly distinguish replacement veins from other veins. They have their own peculiar structure and never show any trace of an inherited structure. Serpentine frequently occurs as a pseudomorph after other minerals, but pseudomorphs are never found in chrysotile veins. Replacement veins are characterized by great irregularity in width and lack of sharp boundaries, while chrysotile veins frequently show remarkable uniformity in width, and always have well-defined walls.

The theory that "the veins are crystallized portions of the serpentine walls, and that the crystals (fibers) have grown outward from the original crevices which are now represented by partings of iron ore found near the center of the veins,"[22] was evidently advanced in order to avoid some of the more obvious objections to the preceding hypothesis, but it completely fails to explain many of the characteristic phenomena of chrysotile veins. If the fibers grew outward in the way here postulated, some of

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[19] J. H. Pratt: Asbestos, *Mineral Resources*, 1904, *U. S. Geological Survey*, pp. 1137–1140 (1905).

[20] J. S. Diller: Asbestos, *Mineral Resources*, 1907, *U. S. Geological Survey*, pt. II, pp. 720–721 (1908).

[21] J. A. Dresser: *Op. cit.*, pp. 59–60.

[22] J. A. Dresser: *Op. cit.*, p. 66.

them would unquestionably penetrate the massive serpentine for greater distances than others, thus giving irregular boundaries to the veins, but chrysotile veins have sharply defined boundaries and are easily separated from the wall rock.

In discussing the deposits of southern Quebec, Dresser states: "the facts are self-evident that zones of the country rock have been altered to serpentine and proportionate parts of these have taken the form of asbestos veins."[23] But, if the veins are merely "portions of the serpentinized bands which have crystallized *in situ*".[24] why is the ratio of chrysotile to massive serpentine limited? Why are the veins limited to a width of 1, 2 or, in rare instances, 3 in.?

This theory does not satisfactorily explain the angular inclusions of massive serpentine that frequently mark the central parting of chrysotile veins and are also to be found irregularly distributed through them. It affords no explanation of the occasional presence of more than one parting or of bends in the fibers such as are shown in Figs. 4, 5 and 6. It does not explain the gradation of cross-fiber into slip-fiber as illustrated in Figs. 3 and 7. Most of the objections here given are equally applicable to the other theories of vein formation so far discussed, and, in view of all the facts, the conclusion is inevitable that none of these theories furnishes a true explanation of the origin of chrysotile veins.

As already noted in this paper, fibrous veins with structural features analogous to those of chrysotile veins have been produced in the laboratory, where their formation and growth could be carefully observed. The results obtained from these experiments, supplemented by a study of fibrous veins in different kinds of rock, have led the writer to certain definite conclusions. These conclusions have been formulated into a theory of vein formation that seems to explain satisfactorily the phenomena connected with chrysotile veins. Briefly stated, this theory is that all cross-fiber veins are formed through a process of lateral secretion, the growing veins making room for themselves by pushing apart the inclosing walls; and that the fibrous structure is to be attributed largely to the physical conditions which have limited crystal growth to a single direction. In the case of the asbestiform minerals, the fibrous structure is accentuated by a normal prismatic habit and cleavage.

In individual occurrences it may be difficult or impossible to determine why the fibrous mineral was taken into solution, the cause of its redeposition and the other details of origin; but, in the case of serpentine, pressure due to expansion in volume is probably the controlling factor.

Serpentine is a secondary mineral resulting, as a rule, from the altera-

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[23] J. A. Dresser: Asbestos in Southern Quebec, *Trans.*, vol. 50, p. 957 (1915).

[24] J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey, Memoir* No. 22, p. 65 (1913).

tion of preëxisting silicates of magnesia. The alteration is seldom one of simple hydration, for usually some material is added or subtracted, and in many cases the constituents out of which serpentine is formed are derived from several different minerals. Serpentine is therefore found in rocks that are either of sedimentary or of igneous origin. Locally it may be the dominant mineral thus forming serpentine rocks. Such rocks are derived chiefly from igneous rocks of the pyroxenite-peridotite family which contain a high magnesian content. Although serpentine is secondary after many minerals, the most important source is olivine followed by the magnesian pyroxenes and amphiboles.

The alteration to serpentine is accompanied by an increase in the volume of the rock mass, which results in the development of pressure whenever there is resistance to expansion. This pressure can not be explained, however, by attributing it to a chemical reaction taking place with increase of volume, as, for example, when plaster of Paris combines with water and sets, because, in many cases where serpentine is formed from anhydrous minerals, there is an actual decrease in the total volume of the system, if the volume of the water is taken into consideration. For instance, when serpentine is formed from olivine according to the equation:

$$3Mg_3FeSi_2O_8 + 6H_2O + O = 3H_4Mg_3Si_2O_9 + Fe_3O_4.$$

Neglecting the oxygen, the volume changes in this reaction are as follows: 281.3 parts of olivine combine with 108 parts of water to give 321 parts of serpentine and 44.5 parts of magnetite. Hence there is an increase in the volume of solid amounting to approximately 30 per cent., while there is a decrease of about 6 per cent. in the volume of the system as a whole. The pressure resulting from this reaction is, therefore, due to the fact that water in a liquid or gaseous state is able to penetrate the rock mass through capillary and subcapillary openings while the solid serpentine formed does not escape in like manner.

The water may penetrate the rock through fractures, along the contact between mineral grains and along the cleavages of minerals. As the alteration proceeds, readjustments in the rock mass, due to increase in volume, result in the formation of new fractures. Some of the water possibly travels for short distances through a process of diffusion or trading, by which a molecule of serpentine gives up its water to a neighboring anhydrous molecule and then takes up more water.

The maximum pressure that may be developed locally as a result of the alteration must be enormous, since, in most cases, it is limited solely by the resistance offered to expansion. Pressure alone can have no direct effect in preventing a reaction that takes place with decrease in volume; and, whether the reaction will take place or not is determined entirely

by the partial pressure of the water vapor and the vapor pressure of the serpentine.

The development of pressure in this way is nicely illustrated by the following laboratory experiment:

A porous porcelain cup was partly filled with anhydrous cupric chloride ($CuCl_2$), after which the open end was sealed with paraffine. The cup was then placed, together with a beaker containing water, under a bell jar, and allowed to stand undisturbed. Water vapor was absorbed through the capillary pores of the cup with the formation of the hydrous salt ($CuCl_2.2H_2O$). At the end of 5 days small fractures were observed, and these gradually increased in length and width as the volume of the salt continued to increase. The fracturing of bottles by salts that take up water of crystallization from the atmosphere is a common phenomenon in chemical laboratories.

The pressure developed as a result of the increase in rock volume when serpentine is formed is important as a factor in the formation of chrysotile veins, chiefly because the solution of the serpentine and its redeposition in the form of chrysotile seem to be determined very largely by pressure. The pressure is at a maximum where serpentine is being formed, and here the mineral is most readily taken into solution. The separation of serpentine from solution is accompanied by expansion in volume, and therefore takes place only where the forces opposing expansion are not prohibitive.

All existing fractures or joints, whether dynamic in origin or due to contraction on cooling, are favorable places for the deposition of chrysotile, because as a rule less force is required in pushing apart the walls of existing fractures than is necessary in forming new ones. The alteration of a rock to serpentine begins along the existing fractures which divide the rock mass into blocks of variable size. As the alteration slowly penetrates inward from the surfaces of a block, strains are set up between the expanded outer shell and the unaltered central part. These strains tend to produce exfoliation fractures, and, although the resisting pressure is probably sufficient in most cases to prevent the formation of open fissures, this pressure is locally reduced by the tendency to fracture, thus permitting the separation of serpentine from solution and the growth of new veinlets of chrysotile.

The chrysotile deposits of Thetford, Canada, furnish good illustrations of veins that have originated in this way,[25] and Merrill has described veins in the serpentine at Montville, N. J., that are difficult of explanation under any other hypothesis. At the latter locality the serpentine has been formed through the hydration of nodules of lime-magnesian

[25] J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey Memoir*, No. 22, pp. 58–59 and Fig. 7 (1913).

pyroxene occurring in dolomite, and the narrow chrysotile veins are in a general way parallel to the surface of the original nodule of pyroxene.[26]

Since the pressure due to expansion prevents the formation or maintenance of appreciable open spaces, water must reach the unaltered rock, at least partly, through very small capillary and subcapillary openings. Under these conditions circulation is extremely slow and the movement is probably limited almost entirely to a single direction, *i.e.*, toward the unaltered portion of the rock. It is not necessary to assume the presence of solutions circulating toward the chrysotile veins in order to explain their growth, as the transfer of serpentine through the short distance from the place of its formation to the walls of the growing vein is probably due chiefly to diffusion of the material through the solution.

If pressure due to expansion is the principal factor in bringing about solution of the serpentine, there must be a close relation between the increase in volume of the rock mass and the volume of material removed in solution and redeposited as chrysotile. This probably explains why the percentage of cross-fiber chrysotile in serpentine is less than the increased volume of the rock, and why the width of chrysotile veins in partly altered rock sometimes bears a definite ratio to the thickness of the inclosing bands of massive serpentine. When the rock is homogeneous and the alteration proceeds uniformly, a definite and limited proportion of the resulting serpentine is removed in solution, for the local pressure is relieved by the transfer of this excess material.

When the formation of serpentine is accompanied by little or no increase in the volume of the rock because of the complete removal of a portion of the products, no chrysotile may be formed. It is possible that in individual cases the resistance to expansion and various other factors prevent the formation of chrysotile veins.

The difference in the specific gravities of massive serpentine and chrysotile is unquestionably due to the difference in their modes of origin. According to Dana[27] the specific gravity of massive serpentine varies from 2.50 to 2.65, while that of chrysotile is only 2.219.

While it seems probable that pressure is the controlling factor in the solution of serpentine and its redeposition as chrysotile, this is not essential to the writer's theory of the origin of cross-fiber veins. The solution of mineral matter and its redeposition in fibrous veins may result from one or more of several different causes. In the laboratory experiments referred to above, supersaturation was induced by either evaporation or cooling. The essential conditions for the growth of cross-fiber veins are: (1) the growing fibers must be in contact at their base with supersaturated solutions; and (2) that the solutions must reach the grow-

[26] G. P. Merrill: On the Serpentine of Montville, New Jersey. *Proceedings of the U. S. National Museum*, vol. 11, pp. 105–112 (1888).

[27] E. S. Dana: *System of Mineralogy*, Ed. 6, pp. 269–271 (1914).

ing veins through the wall rock. So long as the veins are in contact with supersaturated solutions, growth will continue and the walls will be pushed apart regardless of resisting forces; but the greater the resisting pressure, the greater the concentration must be in order to produce supersaturation.

The force that enables growing veins to make room for themselves by pushing apart their inclosing walls is not due to the tendency of a crystalline substance to assume a regular polyhedral form, for the columnar or fibrous structure of most minerals occurring in cross-fiber veins is not a crystallization property, but is caused by the conditions of growth, as already explained. Under suitable conditions, the fibrous structure may develop in substances that crystallize in any of the systems of crystallization. In most cases it is not the normal habit, and therefore is unstable. The writer believes that the force is due chiefly to the expansion in volume which accompanies the separation of most solids from solution. When a substance separates from solution with increase in volume, the pressure developed depends on the magnitude of the forces resisting expansion, and may be many times the force required to crush the substance. It is altogether improbable that pressure alone could expel solutions occupying subcapillary pores in rocks, and, in serpentine and other rocks found inclosing cross-fiber veins, the openings are almost entirely subcapillary in size. As previously stated, the transfer of material is probably due to diffusion rather than to circulating solutions.

### *Structural Features of Cross-fiber Veins*

Nearly all of the structural features characteristic of the cross-fiber veins found in rocks have been reproduced in the course of laboratory experiments. The diagrammatic sketches shown in Figs. 1 to 7 are all drawn to scale from veins of chrysotile or from veins of other fibrous minerals, but, so far as the phenomena that they illustrate are concerned, they might just as well have been drawn from some of the veins produced in the laboratory.

Cross-fiber veins commonly show a central parting or break in the continuity of the fibers (see Fig. 4), but in some veins the fibers extend from wall to wall without interruption. The central parting is formed whenever fibers start to grow from both walls of a fracture, and apparently this always happens when veins develop along preëxisting fractures, unless growth is limited to one side only. In laboratory experiments, growth is frequently limited to one side of a vein by the development of the fissure in such a direction as to cut the other side off from the solution furnishing material for growth. That this may also occur in the case of chrysotile veins is indicated by the occasional presence of the parting very close to one of the walls, instead of near the center. Since growth

in such veins has been very largely limited to one side, it is more than probable, in some instances, that growth may be entirely confined to one side. In some of the laboratory veins, the absence of a parting is due to simultaneous growth at both ends of the fibers—a fact proved by changing the color of the solutions. This occurs only when vein growth and the inception of fracturing are contemporaneous. The absence of partings is most common in the small lenticular chrysotile veins that frequently narrow and pinch out without intersecting other veins, thus indicating that they were not formed along preëxisting joints. The absence of partings is also more noticeable where the inclosing serpentine is practically free from mineral impurities.

Occasionally two or more partings may be observed in the same chrysotile vein, and these are sometimes symmetrically arranged with respect to the center. Such partings may result from a pause in the process of growth or from a slight displacement of the walls along their contact with the vein. A parting may frequently be observed separating two stages of growth in the fibrous or needle-like ice columns found on clayey soils after a cold night, and a similar parting in fibrous gypsum has been noted by Merrill.[28]

When cross-fiber veins are examined closely, it may be seen that the partings are never plane surfaces. The specimen of chrysotile[29] sketched in Fig. 1 shows how very irregular the partings may be in individual cases. The same phenomenon can be observed on a smaller scale under the microscope; where the fibers look as though they had been pushed past one another, causing the ends to interpenetrate along the line of parting. The inequality in the length of fibers is due to the more rapid growth of those fibers that are most favorably situated for receiving additions of new material. The stresses resulting from unequal growth are relieved by slipping along cleavage planes and prism boundaries, and this tends to accentuate the development of the fibrous structure. Additional evidence of the unequal growth of fibers in chrysotile veins is furnished by the similar displacement of bands roughly paralleling the walls and marked by a slight difference in color from the rest of the vein material.

The partings in chrysotile veins are commonly marked by the presence of granular crystals of magnetite and chromite and angular inclusions of the massive serpentine wall rock, as in Figs. 2 and 7. In size, the fragments of wall rock range from the smallest grains to masses that are larger than the veins themselves. In other words, there is every gradation between branching veins and veins with inclusions of wall rock.

[28] G. P. Merrill: On the Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings U. S. National Museum*, vol. 17, p. 81 (1894).

[29] This specimen of chrysotile, from the deposits on Ash Creek near Globe, Ariz., was furnished through the courtesy of the U. S. Geological Survey.

The inclusions are also irregularly distributed through the veins without reference to partings, and occasionally a large number of fragments may be seen strung out in a broken train that may extend from the wall to a

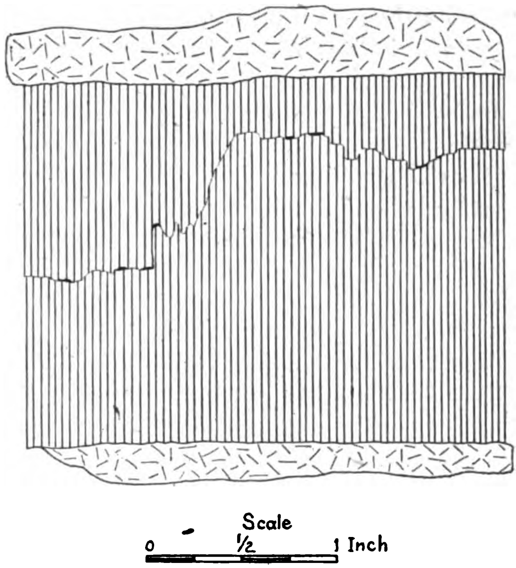


FIG. 1.—CHRYSOTILE VEIN FROM NEAR GLOBE, ARIZ. PARTING IS VERY IRREGULAR AND MARKED BY OCCASIONAL INCLUSIONS OF MASSIVE SERPENTINE WALL ROCK.

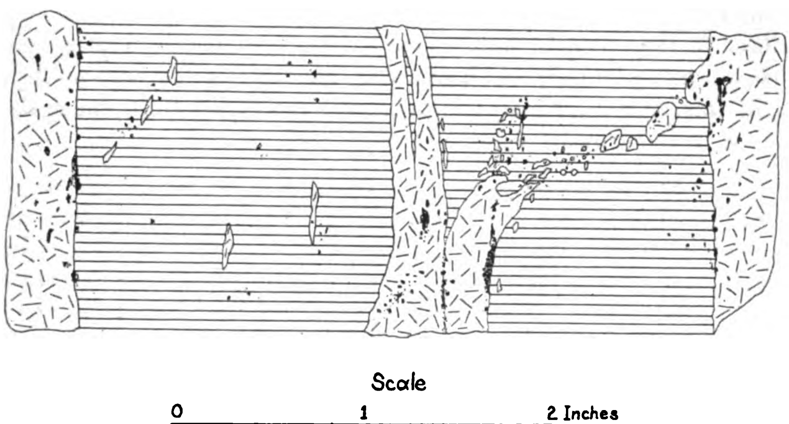


FIG. 2.—CHRYSOTILE VEIN FROM THETFORD, CANADA, CONTAINING CENTRAL INCLUSION OF MASSIVE SERPENTINE AND SMALLER INCLUSIONS OF SERPENTINE AND MAGNETITE EXTENDING IN A BROKEN TRAIN TOWARD ONE WALL. THERE IS A VEINLET OF CHRYSOTILE IN THE LARGE INCLUSION.

central parting, as in Fig. 2. Some of the larger inclusions contain veinlets of chrysotile.

Cross-fiber veins grown in the laboratory are frequently branching,

and they also contain numerous inclusions of wall material distributed through the veins in exactly the same way as in veins of chrysotile and other fibrous minerals, such as crocidolite, gypsum and calcite. The inclusions that mark a central parting represent fragments formed when rupture occurred, and their position is due to the growth of the vein on both sides as new material was added through the walls. When growth

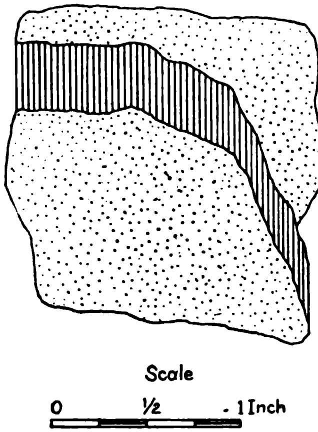


FIG. 3.—VEIN OF FIBROUS CALCITE IN LIMESTONE FROM ST. LAWRENCE COUNTY, NEW YORK.

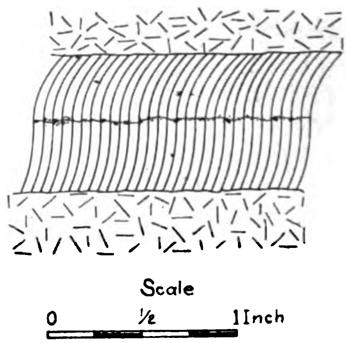


FIG. 4.—CHRYSOTILE VEIN FROM LOWELL, VERMONT, WITH CURVED FIBERS.

is more rapid on one side, the line of parting, together with the inclusions, is closer to the opposite side. Vein matter occasionally begins to crystallize out along an incipient fracture or line of weakness close to the vein, and in this way a fragment is gradually separated from the wall and included in the growing vein.

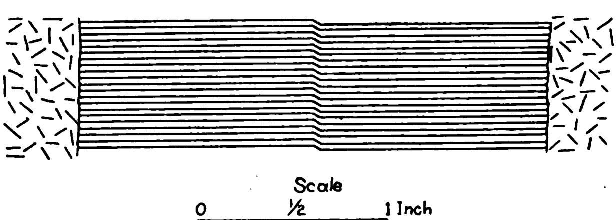


FIG. 5.—CHRYSOTILE VEIN WITH FIBERS SHOWING TWO ABRUPT BENDS.

The fibers are always parallel to one another, and, in most veins, they are approximately normal to the walls (see Fig. 2), but frequently they are more or less oblique. Laboratory experiments prove that the fibers always extend in the direction in which the walls move as they are pushed apart by vein growth. In most veins the fibers are normal to the walls because the walls are usually forced directly apart, but when the walls

have also a lateral displacement because of the simultaneous growth of adjacent non-parallel veins or other causes, the fibers grow in the direction of the resultant motion. If the course of a vein is not straight, the fibers may be normal to the walls at one place and oblique at another, as in the calcite vein from St. Lawrence County, New York, sketched in Fig. 3. In this way cross-fiber veins may grade imperceptibly into slip-fiber veins.

As long as the relative motion of the walls of a growing vein is in a straight line the fibers are straight; any change in the direction of motion is immediately recorded by the slowly lengthening fibers. If the change in the direction of relative motion is gradual and continuous, the fibers are curved, as in Fig. 4, and if the change is abrupt, it results in the development of sharp bends. In the chrysotile vein shown in Fig. 5, the initial movement of the walls had a lateral component, which soon

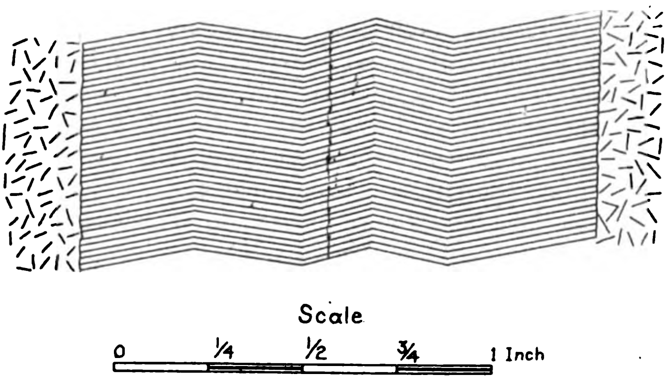


FIG. 6.—CHRYSOTILE VEIN WITH FIBERS SHOWING FOUR ABRUPT BENDS AND A CENTRAL PARTING.

disappeared as the vein continued to grow. The structure here suggests that the formation of the vein fracture was due, at least in part, to shearing stresses. Most of the fibers in this vein extend from wall to wall without a break, and their tensile strength does not seem to be appreciably affected by the bends.

Sometimes the fibers of a vein record several changes in the relative movement of the walls, and this gives a banded appearance due to the unequal reflection of light where the fibers run in different directions. The specimen of chrysotile from Thetford, Canada, sketched to scale in Fig. 6, is a beautiful example of this kind of banding. The beginning of the vein is recorded by a parting within the central band, and there are numerous inclusions of chromite and massive serpentine distributed along the line of parting. The movement of the walls during the first stage of vein growth is recorded by the direction of the fibers in the central band, and during the enlargement of the vein there were two sudden

changes in the relative motion of the walls with the formation of two additional bands on both sides of the central band, thus making five in all.

The structural phenomena discussed in the preceding paragraphs all furnish evidence tending to confirm the writer's theory of the origin of cross-fiber veins, and they are difficult or impossible of explanation under any other hypothesis of vein formation so far advanced.

### *Veins of Crocidolite and Other Fibrous Amphiboles*

All of the asbestiform minerals belonging to the amphibole group have been reported as occurring in cross-fiber veins, though the number of such instances is small, and, as yet, little information has been published concerning the details of individual occurrences. For this reason, it is not possible at the present time to discuss, except in a general way, the factors controlling the origin, solution and redeposition of the minerals, but the data now available indicate that these veins must also have been formed through a process of lateral secretion, and that they have made room for themselves by forcing apart the inclosing wall rock. Crocidolite is the only asbestiform mineral other than chrysotile known to occur in cross-fiber veins in commercial quantities, and only one occurrence of this mineral, so far reported, is of any importance.

Crocidolite has long been known to occur in West Griqualand, South Africa, and most specimens found in mineral collections have come from that source. The deposits are located in a range of quartzose schists called the asbestos mountains. The mineral is characterized by a dull lavender-blue color, due to the presence of considerable iron protoxide. In places it is altered by oxidation and infiltration of silica to a compact siliceous stone with a bright yellow color, and chatoyant luster, which has given it the popular name of *tiger-eye*. A brief account of this occurrence is given by Cirkel[30] in his monograph on chrysotile asbestos. He states that the crocidolite "is generally found in veins, seldom less than 2 in., and more often 4 in. and 5 in. wide, formed of closely compacted parallel fibers which run from wall to wall of the vein without break or fault. Several veins have been found, regular in extent, and the fiber always lies at right angles to the sides of the deposit. The inclosing rock is a dark brown shale."

A considerable number of specimens of crocidolite from South Africa have been examined megascopically and also under the microscope, by the present writer. All of the structural phenomena characteristic of

[30] Fritz Cirkel: Crysotile-Asbestos, Its Occurrence, Exploitation, Milling, and Uses, Ed. 2, *Canada Department of Mines, Mines Branch, Report* No. 69, pp. 239–240 (1910). The quotations here given appear to have been taken from a paper by H. T. Olds, Notes on Blue Asbestos, read before the *Institution of Mining and Metallurgy*, London, *Transactions*, vol. 7, pp. 122–123 (1899).

chrysotile veins may be found in veins of crocidolite. Specimens in the collections of the University of South Carolina show that the vein fibers are sometimes oblique instead of perpendicular to the walls, partings are frequently present and may be marked by inclusions of hematite, magnetite, quartz and fine-grained wall rock; bent and curved fibers are also common. Most of these features may likewise be observed in specimens of tiger-eye. In view of all the facts stated above, it must be concluded that, although consisting of different material, the crocidolite veins have grown in the same way as those of chrysotile.

Anthophyllite and true asbestos are rarely found in cross-fiber veins because the conditions prevailing at the time of their formation are not, as a rule, favorable for the solution and transportation of these minerals to a new place of deposition. According to Hopkins,[31] the veins of cross-fiber anthophyllite found in Georgia "are in every way similar to chrysotile, the fibers being perpendicular to the inclosing walls, and sometimes continuing from the one to the other, but more commonly jointed one or more times." This description indicates that their origin is similar to that of other cross-fiber veins.

## Origin of Slip-fiber

All asbestiform minerals, with the possible exception of crocidolite, are known to occur as slip-fiber, and true asbestos seldom occurs in any other way. The origin of the slip-fiber type is a question that has aroused much less diversity of opinion than the origin of the cross-fiber type. Merrill[32] and others have advocated the efficacy of pressure and shearing in the production of a fibrous structure in minerals. The common association of slip-fiber with slickensided surfaces and other evidences of faulting have apparently convinced most investigators that shearing is the essential factor in its formation. The various ways in which unequal pressure may bring about the development of a fibrous structure have been previously discussed, but there is reason for believing that all of the so-called slip fiber has not been formed directly as a result of unequal pressure.

Cirkel believed that the slip-fiber found in the Broughton district, Quebec, had been formed from cross-fiber chrysotile as "the result of the secondary readjustment, which took place immediately after the crystallization of the fiber in veins. Both rock and veins must have been in a semi-magmatic condition during this period, and pressure may have aided this process of physical alterations of the mass in a marked degree."[33]

[31] O. B. Hopkins: *Op. cit.*, pp. 104–105.

[32] G. P. Merrill: On the Serpentine of Montville, New Jersey, *Proceedings of the U. S. National Museum*, vol. 11, p. 105 (1888), and Notes on Asbestos and Asbestiform Minerals, *Proceedings U. S. National Museum*, vol. 18, p. 289 (1895).

[33] Fritz Cirkel: *Op. cit.*, pp. 94–95.

It is possible if not probable that some slip-fiber has been formed from cross-fiber, but there is absolutely no reason for postulating that the rock was "in a semi-magmatic condition."

Dresser[34] reached a somewhat different conclusion as to the origin of the Broughton deposits. He notes that the serpentine at Broughton was probably derived from pyroxenite, while that at Thetford, where cross-fiber predominates, was derived from peridotite. After calling attention to the fact that the asbestos is limited to "the sheared and shattered" portions of the rock mass, he argues that there is some connection between the two. He further[35] suggests that when the pyroxenite was altered to serpentine, "the upper portions may have had a development of asbestos in the form of 'mass-fibre' or asbestos irregularly distributed through the rock, possibly due to a greater action of magmatic waters near the top of the sills; and that this fibrous structure weakened

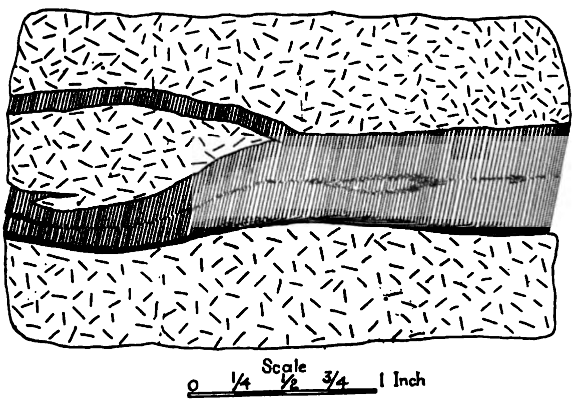


FIG. 7.—CHRYSOTILE VEIN FROM LOWELL, VERMONT, SHOWING GRADATION OF CROSS-FIBER INTO SLIP-FIBER ALONG THE LOWER WALL.

the resisting power of the rock and the shear zone was thus localized." The chief objection to the latter hypothesis is, that while chrysotile is common both as cross-fiber and as slip-fiber, it is not known to occur as mass-fiber.

The investigations of the present writer indicate that much so-called slip-fiber has really been formed in the same way as cross-fiber, as is evidenced by the presence of partings, inclusions and fibers that are oblique rather than parallel to the inclosing walls. In fact, every gradation may be found between cross-fiber veins in which the fibers are nor-

[34] J. A. Dresser: A Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey Memoir* No. 22, p. 69 (1913).

[35] J. A. Dresser: *Idem.*, p. 70.

mal to the walls, and slip-fiber veins in which the fibers are approximately parallel to the walls. In some instances the gradation may be observed in a single vein, as in the specimen[36] from near Lowell, Vt., sketched in Fig. 7. In this vein some of the fibers terminate at the abrupt bend, but others continue without a break, although they are less flexible where approximately parallel to the vein walls than where they are more nearly normal.

## ORIGIN OF MASS-FIBER

Asbestos of the mass-fiber type, so far as known, is always anthophyllite. Several occurrences of this kind are found in the vicinity of Sall Mountain, Georgia, which may be considered the type locality, as the deposits have been worked on a small scale for several years. More recently similar deposits have been opened up about 14 miles southeast of Kamiah, Idaho.

The deposits at Sall Mountain are roughly elliptical in shape, and the largest, which has been practically mined out, had a length of about 75 ft., a width, near the middle, of 50 ft., and apparently pinched out at a depth of 50 ft. It is estimated that considerably over 90 per cent. of the rock mass is realized as fiber.[37] The fibers are arranged in small groups or bundles, and range up to an inch in length, though averaging only about ½ in. The fibers show a strong tendency to form spherical bunches with radial structure, but because of mutual interference these bodies are, as a rule, only imperfectly developed, and therefore, in most cases, the rock consists of a mass of fibrous bundles and sheaves oriented in all directions. Occasionally, however, cross-fractures show well-formed rosettes of radiating fibers. Individual fibers sometimes appear jointed or broken. They are low in tensile strength and brittle, readily breaking into short lengths so that none of the material is of spinning grade. Hopkins states that, because of lack of flexibility, the fibers are broken so many times during the milling process "that they are exceptionally ¼ in. long, while the bulk is $\frac{1}{10}$ in. and less."[38]

According to Diller, the rock found near Kamiah, Idaho, "is very like that mined at Sall Mountain, Georgia, except that in Idaho the fibers are somewhat coarser and the radial groups larger."[39]

Hopkins, who studied and described the Georgia deposits in great

[36] This specimen was obtained through the courtesy of the U. S. National Museum.

[37] J. S. Diller: Asbestos, *Mineral Resources*, 1907, *U. S. Geological Survey*, pt. II, p. 717 (1908).

[38] O. B. Hopkins: Report on the Asbestos, Talc and Soapstone Deposits of Georgia, *Geological Survey of Georgia, Bulletin* No. 29, p. 88 (1914).

[39] J. S. Diller, Asbestos: *Mineral Resources*, 1909, *U. S. Geological Survey*, pt. II, p. 729 (1910).

detail, concludes that the origin of the mass-fiber is due largely to the alteration of enstatite.[40]

Since this alteration involves an increase in volume, it probably takes place in the katamorphic zone. The change of enstatite to anthophyllite is paramorphic, and therefore the presence of solutions is not essential, as in the case of reactions involving the addition or subtraction of material. This probably accounts for the difference in the origin and structure of serpentine and of mass-fiber anthophyllite. The alteration of a rock mass to serpentine begins along fractures and all accessible surfaces, gradually penetrating toward the unaltered interior; while the alteration to anthophyllite begins at a large number of centers, more or less evenly distributed through the rock, and then spreads out radially from these centers. The lack of flexibility in mass-fiber anthophyllite is attributed largely to the fact that physical conditions have not controlled and limited its growth to the same extent that they have controlled the formation of the cross-fiber and slip-fiber types. The separation of the fibers is made easier by the hydration of the anthophyllite in the belt of weathering.

[40] O. B. Hopkins: *Op. cit.*, p. 104.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# A Study of the Silica Refractories*

BY J. SPOTTS McDOWELL, B. S.†

(New York Meeting, February, 1917)

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* An investigation conducted by the author at the laboratories of the Massachusetts Institute of Technology from October, 1915, to February, 1916, and published with the consent of Professor H. O. Hofman, Acting Head of the Department of Mining and Metallurgy. The supplementary report represents work done later while associated with the Harbison-Walker Refractories Co. at Pittsburgh, Pa.

† Research Department, Harbison-Walker Refractories Co., Pittsburgh, Pa.

## Acknowledgments

To a very considerable degree I am indebted to the kindness of others for the material herein presented. The subject was suggested to me by Kenneth Seaver, Chief Engineer of the Harbison-Walker Refractories Co., Pittsburgh, Pa., and represents, in fact, the continuation of an investigation begun by him.*

The materials studied were furnished by the Harbison-Walker Refractories Co., and the test brick manufactured at the Hays Station plant of that company under the supervision of R. H. H. Pierce, Chief Chemist. The conditions under which the test brick were made were devised by members of the Harbison-Walker Refractories Co.'s engineering department.

Whatever value the microscopic determinations may possess is due entirely to the careful instruction and kindly counsel of Professor C. H. Warren of the Massachusetts Institute of Technology. I am particularly grateful to Victor Dolmage, who was extremely generous with his time in assisting with some of these determinations. The photomicrographs are the work of W. L. Whitehead. The crushing and cross-breaking tests were made with the assistance of I. H. Cowdrey of the Mechanical Engineering Department of the Institute of Technology. I wish to express my thanks to all those named, as well as to others who have rendered assistance in various ways.

## Introduction

The magnitude of the thermal expansion of silica brick, and its inability to withstand rapid temperature changes, present problems of considerable importance in the manufacture of siliceous refractories.

In the study represented by the experimental data herein described, these problems are considered from the point of view afforded by the results of recent investigators: Fenner and his associates of the Geophysical Laboratory, Washington, D. C., who have established the stability relations of the silica minerals, and Endell and co-workers in Berlin, who have studied the changes brought about in the constitution of siliceous refractory materials upon the application of heat.

In the process of manufacture, the quartzite from which the brick is made changes in part into cristobalite and tridymite. The attempt is herein made to determine by microscopic methods the degree of transformation in various specimens of test bricks manufactured at slightly varying temperatures of burning, with varying coarseness of grain, and to determine the effect of repeated burning. The effect of variations

* Kenneth Seaver: Manufacture and Tests of Silica Brick for the Byproduct Coke Oven, *Trans.*, vol. 53, pp. 125–139 (1916).

in these conditions upon crushing strength and modulus of rupture has also been considered.

Coincidently, a study has been made, from the available literature, of the properties of the silica minerals and the silica refractories. From the knowledge so gained, combined with the experimental data regarding the constitution of the test bricks, deductions have been made as to the conditions of manufacture under which silica brick of decreased thermal expansion and increased power to withstand rapid temperature changes might conceivably be produced. It is hoped that these deductions may serve to indicate possible starting points for future investigations.

The first part of this report comprises data compiled from various publications. Although much of this has no bearing upon the specific problem in hand, it is believed to be of interest in any thorough study of the properties of siliceous refractory materials.

## THE SILICA MINERALS

### Stability Relations

The crystal modifications of silica important in this connection are quartz, tridymite and cristobalite, each of which possesses $\alpha$ and $\beta$ phases. Any one of these three modifications may be converted into either of the others by appropriate heat treatment. The formation of tridymite seems always to require the presence of a flux or catalyzer, while cristobalite is formed in the absence of a catalyzer. The inversion temperatures[1] are as follows:

870°C $\pm$ 10 quartz $\rightleftharpoons$ tridymite
1,470° $\pm$ 10 tridymite $\rightleftharpoons$ cristobalite
575° $\alpha$ quartz $\rightarrow$ $\beta$ quartz; 570° $\beta$ quartz $\rightarrow$ $\alpha$ quartz
117° $\alpha$ tridymite $\rightarrow$ $\beta_1$ tridymite; 163° $\beta_1$ tridymite $\rightarrow$ $\beta_2$ tridymite; reversions on cooling not sharp.
274.6° – 219.7° $\alpha$ cristobalite $\rightarrow$ $\beta$ cristobalite
240.5° – 198.1° $\beta$ cristobalite $\rightarrow$ $\alpha$ cristobalite

At ordinary temperatures each mineral exists only in the $\alpha$ phase, changing into the $\beta$ phase as the temperature is raised. The $\alpha$ to $\beta$ transformations are all rapid, while considerable time is required for the transformation of one mineral into another.

*In the presence of a flux*, when any form of silica is heated a sufficient length of time below 870°, quartz is always formed;[2] between 870° and 1,470° tridymite and from 1,470° upward to the melting point of silica, cristobalite. *Heated without a flux*, the inversion from quartz to tridymite does not occur, but the transformation is direct to cristobalite,

[1] C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, p. 383 (1913).

[2] *Ibid.*, p. 358.

and the temperature of incipient formation of cristobalite is depressed to about 1,250°; under the same conditions, tridymite is converted into cristobalite, with the inversion temperature raised to about 1,570°. Fenner's experiments on quartz powder heated without a flux gave the following results:

108 hr. at 1,250°: a very small per cent. of cristobalite formed.
90 hr. at 1,360°: resultant product two-thirds cristobalite, one-third quartz.
1 hr. at 1,570°: transformation into cristobalite nearly complete.

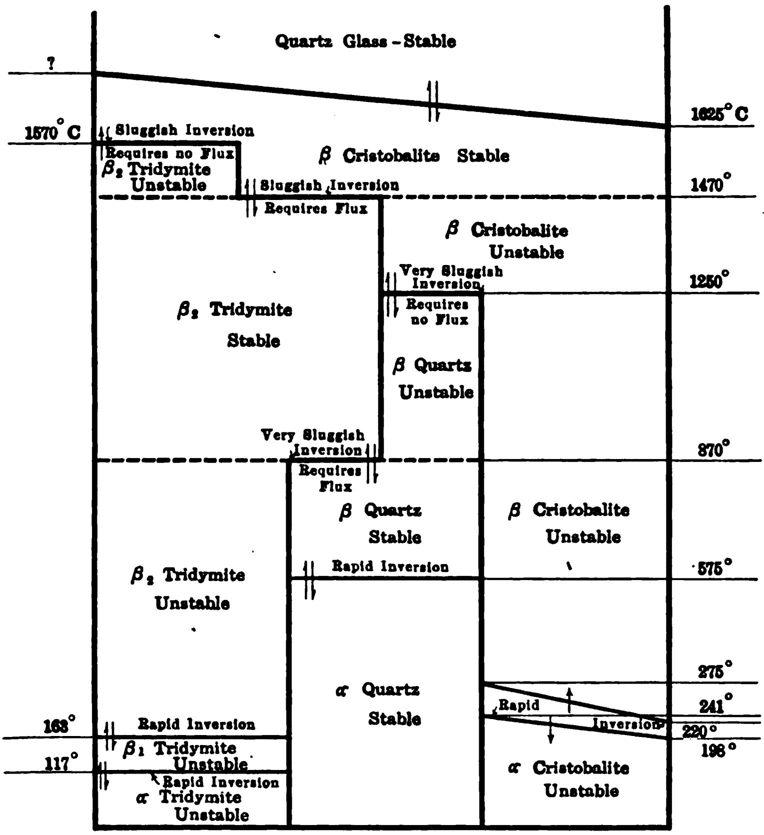


FIG. 1.—STABILITY RELATIONS OF THE SILICA MINERALS.

Quartz glass heated above 1,200° becomes gradually converted into cristobalite.[3] The percentages of cristobalite formed under various conditions of heating are shown in the accompanying table, taken from the work of Rieke and Endell.

[3] Rieke and Endell: Devitrification of Quartz Glass, *Silikat Zeitschrift*, vol. 1, No. 1, p. 8 (Jan., 1913).

Table 1.—*Cristobalite Formed by Heating Quartz Glass under Various Conditions (Rieke and Endell)*

| Time of Heating, Hours | Temperature | | | | |
|---|---|---|---|---|---|
| | 1,200° | 1,300° | 1,400° | 1,500° | 1,600° |
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| ½ | — | — | 5 | 34 | 60 |
| 1 | 0 | 2 | 17 | 50 | 90 |
| 2 | 0 | 5 | 25 | 100 | |
| 4 | 0 | 10 | — | | |

Quartzite heated three to five times to 1,450° in a porcelain kiln is converted chiefly into cristobalite, with here and there scattered residual

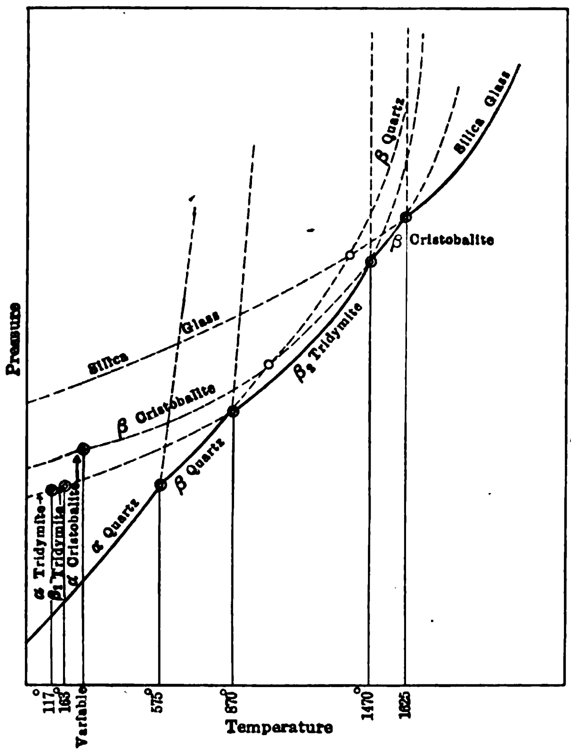


Fig. 2.—Stability Relations of the Silica Minerals (after Fenner).

quartz grains; after 10 burnings at 1,450° wedge-shaped twin crystals of tridymite begin to appear.[4] Silica brick, which consists principally of

[4] K. Endell: *Stahl und Eisen*, vol. 33, p. 1855 (Nov., 1913).

cristobalite when placed in use, gradually changes at high temperatures in the furnace to tridymite, as explained later on in this paper.

The stability fields and existence ranges of the various forms of silica are shown in Fig. 1; an equilibrium diagram (after Fenner) in Fig. 2.

TABLE 2.—*Optical Properties**

| Composition | Crystal System | Crystal Habit | Elongation | Optical Orientation | nNa | | Optical Character | Remarks |
|---|---|---|---|---|---|---|---|---|
| | | | | | α | γ | | |
| α Quartz...... | Hexagonal Trapezohedral Tetartohedral | Pyramidal | Y | c = γ | 1.544 | 1.553 | + | Stable below 575°. |
| β Quartz...... | Hexagonal Trapezohedral Hemihedral | Pyramidal | Y | c = γ | | | + | Stable between 575° and 870°. Inverts on cooling to α quartz. |
| α Tridymite... | Orthorhombic | Thin pseudo-hexagonal plates | | | 1.469 | 1.473 | + | Exists below 117°. Usually finely intergrown aggregates. Optic axial angle large. |
| β Tridymite... | Hexagonal | Hexagonal plates | | c = γ | | | + | Exists only above 117°. Inverts on cooling to α tridymite. |
| α Cristobalite. | Probably tetragonal | Equaint grains | | c = α | 1.484 | 1.487 | − | Exists below 275°. |
| β Cristobalite.. | Isometric | Equaint grains | | | | | | Exists only above 198°. Inverts on cooling to α cristobalite. |

* C. N. Fenner: *American Journal of Science*, ser. 4, vol. 36, pp. 351–356 (1913).
N. L. Bowen: *American Journal of Science*, ser. 4, vol. 38, p. 245 (1914).
Rankin and Wright: *American Journal of Science*, ser. 4, vol. 39, pp. 4 and 74 (1915).

Tridymite crystals usually occur as crystalline aggregates of random orientation, as broad, thin hexagonal plates, which appear as needles or laths when turned on edge, and as wedge-shaped twin crystals. The hexagonal plates of usual thinness appear perfectly isotropic when lying on the base; the needle or lath-like sections have weak double refraction, parallel extinction and negative elongation.

Cristobalite does not develop a well-defined crystal form but usually occurs as minute branching skeleton crystals, with the branches often showing octahedral terminal caps. The higher temperature form of cristobalite is isometric; on cooling it becomes weakly birefringent. The birefringence in thin sections is barely discernible with the sensitive tint plate.

TABLE 3.—*Density*

| | Specific Gravity | |
|---|---|---|
| Quartz | 2.65 | |
| Tridymite | 2.270 | Artificial tridymite (Fenner). |
| | 2.28 | Natural tridymite (Mallard). |
| | 2.32 | (Endell). |
| Cristobalite | 2.333 | Artificial cristobalite (Fenner). |
| | 2.34 | Natural cristobalite (Mallard). |
| | 2.33 | (Endell). |
| Cristobalite | ..... | At 1,500° has same density as quartz glass; at 300°, slightly greater density than quartz glass (Rieke and Endell). |
| Quartz glass | 2.21 | (Dana, Endell). |
| | 2.194 | (Schwarz). |

## MELTING POINTS

"It appears that the fusing point of quartz is lower than 1,470°, but that at this temperature it passes into cristobalite almost as rapidly as it melts. The fusing point of tridymite should lie between those of quartz and cristobalite."[5] "It is clear that the indicated melting point of cristobalite must be higher than 1,625°, the value found by Fenner, higher than 1,685° even, the value found by Endell and Rieke.[6] . . . . . As Dr. Fenner has suggested, cristobalite may have a variable molecular constitution and a similarly variable melting point according to the conditions under which it was formed."[7] Kanolt[8] finds that pure silica flows distinctly at 1,750°, which is, therefore, the apparent melting point and higher than the true melting point. Quartz glass softens at 1,500° and melts between 1,700° and 1,800°.[9]

## THERMAL EXPANSION

The results of LeChatelier's[10] experiments are shown in Fig. 3; those of Day, Sosman and Hostetter[11] in Fig. 4.

[5] C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, p. 383 (1913).

[6] Endell and Rieke: *Zeitschrift für Anorganische, Chemie.*, vol. 79, pp. 239–259 (1913).

[7] N. L. Bowen: Ternary System: Diopside-Forsterite-Silica, *American Journal of Science*, ser. 4, vol. 38, p. 218 (1914).

[8] C. W. Kanolt: Melting Point of Fire-Bricks, *U. S. Bureau of Standards, Technologic Paper* No. 10, p. 14 (1912).

[9] A. E. Marshall: Fused Silica Ware, Its Manufacture, Properties and Uses, *Metallurgical and Chemical Engineering*, vol. 10, pp. 248–249 (Apr., 1912).

[10] LeChatelier: La Silice, *Revue Universelle des Mines*, ser. 5, vol. 1, p. 90 (1913).

[11] Day, Sosman and Hostetter: Determination of Mineral and Rock Densities at High Temperatures, *American Journal of Science*, ser. 4, vol. 37, pp. 1–39 (1914).

LeChatelier gives the mean linear expansion of quartz glass as $0.0_67$; Kaye[12] gives the following figures: 0° to 30°C., $0.0_642$; 30° to 100°, $0.0_653$; 100° to 500°, $0.0_658$; 500° to 900°; $0.0_650$; 900° to 1000°; $0.0_680$. Quartz glass, partly devitrified by long-continued heating at a high tem-

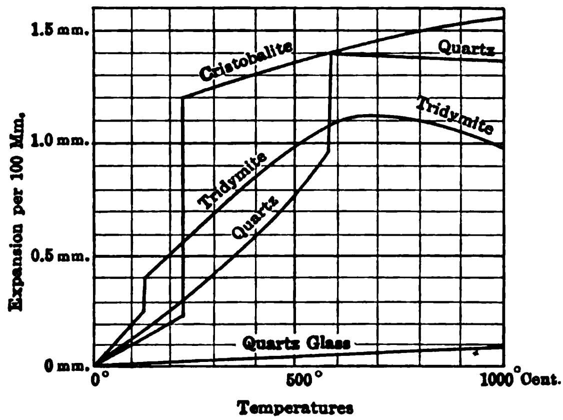


FIG. 3.—RESULTS OF LE CHATELIER'S EXPERIMENTS IN THERMAL EXPANSION.

perature, so that it contains much $\beta$ cristobalite, if cooled rapidly to 300° remains completely clear and transparent, and but few cracks appear. When further cooled, however, to 230°, the sudden formation of numerous

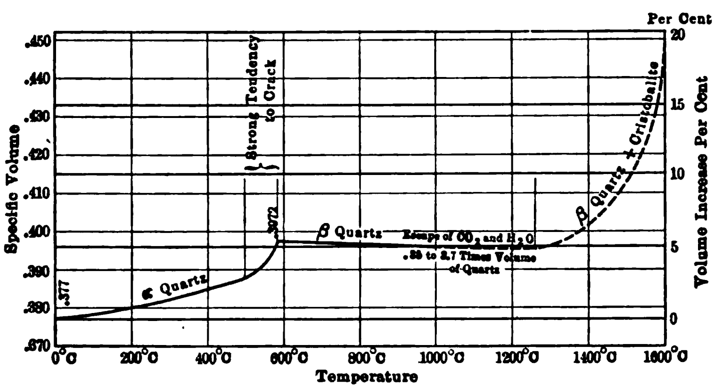


FIG. 4.—THERMAL EXPANSION OF QUARTZ.

fine cracks causes the glass to become white and opaque, coincident with the inversion of $\beta$ to $\alpha$ cristobalite. The specific volume and coefficient of expansion of $\beta$ cristobalite are, therefore, nearly the same as those of quartz glass; the presence of a few cracks at 300° indicate that the

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[12] G. W. C. Kaye: Expansion and Thermal Hysteresis of Fused Silica, *Philosophical Magazine*, ser. 6, vol. 20, pp. 718–728 (Oct., 1910).

cristobalite probably has a slightly greater coefficient of expansion than the glass.[13]

## Solubilities

Rammelsberg,[14] and Lange and Milberg[15] studied the solubility of quartz, opal and amorphous silica in alkaline solutions, but not that of tridymite or cristobalite. Cramer found the solubility in KOH of quartzite burned once in a porcelain kiln to be 41 per cent.; burned 10 times, 70 per cent. After the first burning, this material probably consisted of a mixture of cristobalite and quartz; after the tenth burning, of cristobalite and tridymite, with a small amount of quartz.[16] Schwarz[17] obtained the results shown in Table 4 on boiling powders with grains approximately 0.04 mm. in diameter.

Table 4

| Reagent | Solubility of | | | | Time, Hours |
|---|---|---|---|---|---|
| | Quartz, Per Cent. | Tridymite, Per Cent. | Cristobalite, Per Cent. | Amorphous Silica, Per Cent. | |
| 5 per cent. $Na_2CO_3$ solution | 2.11 | 2.77 | .... | .... | ½ |
| 5 per cent. HF solution... | 30.17 | 76.30 | 74.3 | 96.6 | ½ |
| 1 per cent. HF solution... | 5.20 | 20.30 | 25.8 | 52.9 | 1 |

## Identification of Cristobalite and Tridymite

The method of identification most readily applied is by determination of the indices of refraction by the immersion method, described later on in this paper. Tridymite may also be recognized by its characteristic needles, laths and wedge-shaped twin crystals.

The methods of Rieke and Endell,[18] who have gone into this subject with great care, are given in the following paragraphs. Specific-gravity determinations were of little value in the case of quartz converted by heating into cristobalite or tridymite, for several reasons. The specific gravities of the two last-named are not greatly different; the material always contains some unaltered quartz; and the end products are so

[13] Rieke and Endell: Devitrification of Quartz Glass, *Silikat Zeitschrift*, vol. 1, No. 1, p. 6 (Jan., 1913).

[14] *Annalen der Physik und Chemie, Poggendorf*, vol. 112, pp. 177 (1861).

[15] *Zeitschrift für Angewandte Chemie*, 1897, pp. 393 and 425.

[16] K. Endell: *Stahl und Eisen*, vol. 33, 1770 *et seq.* (Oct., 1913).

[17] R. Schwarz: Chemical Relations of the Various Modifications of $SiO_2$, *Zeitschrift für Anorganische Chemie*, vol. 76, p. 424 (1912).

[18] Rieke and Endell: Volume Change of Some Ceramic Raw Materials on Burning, *Silikat Zeitschrift*, vol. 1, No. 4, pp. 67, 85 (Apr., 1913).

filled with microscopic cracks that the results of specific-gravity determinations are always low.

These fine cracks interfered with microscopic investigations. On account of total reflection at the borders, according to Rieke and Endell, the greater part of the end product appears isotropic, and the birefringence can be detected only upon very careful observation.

By the use of the thermal microscope, it was seen that cristobalite becomes isotropic at 225° to 230°. The effect was particularly evident in cristobalite derived from quartz glass or from quartz heated above 1,600°; it could not be used as a means of identification in the case of cristobalite formed below 1,450°, which, on account of the cracks

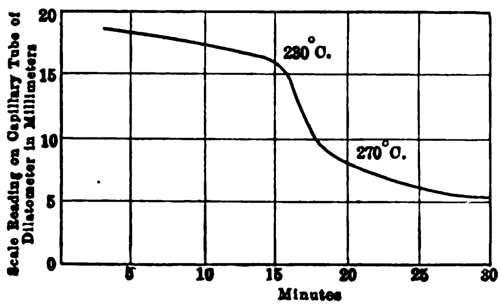


FIG. 5.—CURVE SHOWING EFFECT OF HEAT ON A SPECIMEN OF SILICA BRICK COMPOSED ESSENTIALLY OF CRISTOBALITE.

above mentioned, appears nearly isotropic. The change in volume of cristobalite at 230° is considerable. For that reason, quicksilver dilatometer observations offered a sure way of distinguishing cristobalite from tridymite. A specimen of silica brick, composed essentially of cristobalite, on heating[19] gave the curve shown in Fig. 5.

Rieke and Endell have designated the sudden clearing up shown by cristobalite heated carefully over the flame at about 230°, as the "cristobalite reaction." This effect can easily be observed with the naked eye in the case of cristobalite derived from quartz glass.

## THE SILICA REFRACTORIES

### RAW MATERIAL

#### *American Deposits*

The raw material used for the manufacture of high-grade American silica brick is a true quartzite; the deposits most extensively used are described by Seaver.[20] Named in the order of their importance, they

[19] *Stahl und Eisen*, vol. 33, p. 1856, Fig. 4 (Nov., 1913).

[20] K. Seaver: Manufacture and Tests of Silica Brick for the By-Product Coke Oven, *Trans.* vol. 53, pp. 125–139 (1916).

are the so-called Medina or Tuscarora sandstone of Huntingdon and Blair Counties, Pennsylvania, the Baraboo quartzite of the Devil's Lake

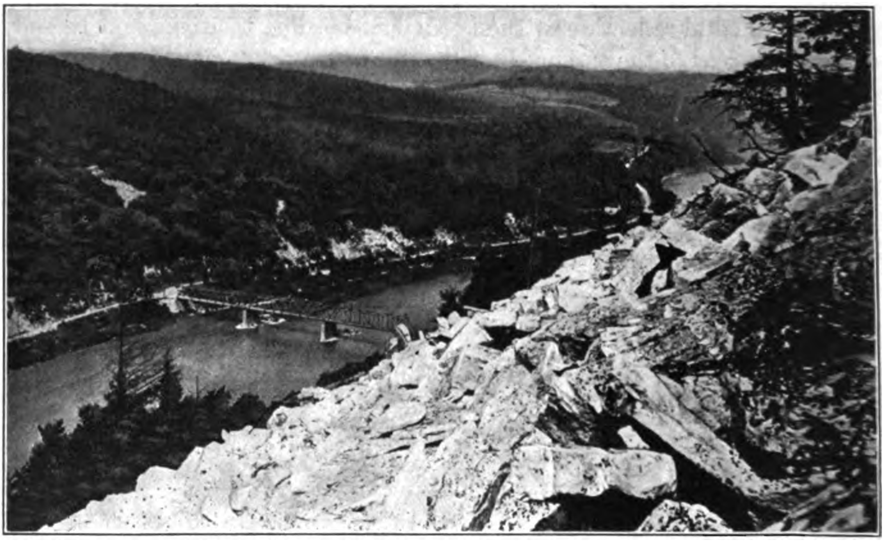

FIG. 6.—GANISTER FLOE IN THE GORGE OF THE JUNIATA RIVER, NEAR MOUNT UNION, HUNTINGDON CO., PENNSYLVANIA.

FIG. 7.—GANISTER FLOE ON THE NORTH SIDE OF THE JUNIATA RIVER, NEAR MOUNT UNION, HUNTINGDON CO., PENNSYLVANIA.

region, Wisconsin, and the deposits of eastern Alabama and of Colorado.

Where quarried for refractory purposes, the Medina is white, hard

and resistant. Its steeply pitching beds form the crests of many of the mountains of central Pennsylvania, and the steep slopes below the summit are covered by great bodies of talus, composed of blocks of the rock, varying in size from pieces weighing a few pounds to those weighing many tons. The talus slopes sometimes cover areas of a thousand acres or more where the beds have been cut through by streams. For the manufacture of silica brick, most of the rock used is taken from these bodies of loose rock, locally called "ganister floes," although rock from the solid measures is sometimes used.

Figs. 6 and 7 are pictures of ganister floes near Mt. Union, Huntington Co., Pa., where the Juniata River cuts through Jack's Mountain.

TABLE 5.—*Typical Analyses**

| Constituent | Medina Quartzite from Pennsylvania | Baraboo Quartzite | Alabama Quartzite |
|---|---|---|---|
| $SiO_2$ | 97.80 | 97.15 | 97.70 |
| $Al_2O_3$ | 0.90 | 1.00 | 0.96 |
| $Fe_2O_3$ | 0.85 | 1.05 | 0.80 |
| CaO | 0.10 | 0.10 | 0.05 |
| MgO | 0.15 | 0.25 | 0.30 |
| Alkalies | 0.40 | 0.10 | 0.31 |

* K. Seaver: *Op cit.*, p. 127–128.

### *Properties Determining Usability*

Not all pure quartzites are adapted to the manufacture of refractories. For refractory purposes three properties are to be considered: mechanical strength, melting point and behavior on burning. Experience shows that to produce a physically strong brick the stone should be hard, dense, and give splintery, angular fragments of heterogeneous size and shape on crushing; one that crushes down to rounded grains can not be used. The strength of the brick and its melting point are conditioned by the analysis of the rock. The silica content may vary from 96 to 98 per cent.; if less silica is present the fusion point is lowered. Attempts to use pure quartzite of 99 per cent. $SiO_2$ have not met with success. The presence of $Al_2O_3$ and $Fe_2O_3$ is necessary to form a bond; their combined percentage is usually about 1.75 per cent., and should not exceed 2.5 per cent. The amount of alkalies should be less than 0.5 per cent.

The melting interval of the stone depends upon its content of bases. For good quartzite it is usually considered to be cone 35 to 36; that is, 1,755 to 1,775°C.[21] Endell[21] obtained the following results, measuring temperatures by means of an iridium-iridium-ruthenium thermoelement

[21] K. Endell: *Stahl und Eisen*, vol. 33, p. 1857 (Nov., 1913).

in the calibration of which the melting points of gold, palladium and platinum were taken as 1,063°, 1,549° and 1,755°, the values determined by Day and Sosman:[22]

At 1,650°C., fine powder of good quartzite sintered together; pea-sized pieces showed no signs of fusion.

At 1,690°C., fine powder of good quartzite melted together into drops; the pea-sized pieces were superficially melted with visible rounding of the edges.

At 1,710°C., little pieces of pure Brazilian quartz 1 to 2 mm. in cross-section were melted round after 10 min. heating; a piece of ¼-in. cross-section had strongly rounded edges. Pea-sized pieces of good quartzite were strongly vitrified and showed fully rounded edges.

### *Expansion on Heating*[23]

Chemical analysis alone does not afford an adequate index to the behavior of a quartzite on heating. Material suitable for refractory purposes does not weaken or crack perceptibly under the influence of high temperatures, and shows considerable volume increase after a first burning, retaining essentially the same volume later after repeated burning. Unsuitable quartzites, notably the pure, coarsely crystalline variety, become badly cracked or completely disintegrated on heating; they usually expand but little on a first burning and considerably thereafter.

TABLE 6

| Material | Volume Increase after Burns | | | | |
|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 |
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| Good quartzite (average of three samples) | 9.4 | 10.7 | 11.0 | 11.3 | 11.5 |
| Quartzite unsuitable for refractory purposes (average of three samples) | 5.3 | 6.5 | 6.5 | 9.7 | 10.7 |
| Melting point of all samples cone 35 to 36. | | | | | |

In Cramer's[24] experiments, hard, fine-grained quartzites, which did not weaken in the heat treatment, gave the greatest expansion on the first burn. The volume increase of one specimen of good quartzite heated

[22] Day and Sosman: *Zeitschrift für Anorganische Chemie*, vol. 72, pp. 1–10 (Aug., 1911).

[23] F. T. Havard: *Refractories and Furnaces*, p. 36, McGraw-Hill Book Co., New York, 1912.

[24] *Stahl und Eisen*, vol. 21, p. 772 (July, 1901); *Tonindustrie Zeitung*, vol. 25, p. 864 (1901).

to cone 16 (1,460° ±) was 17.1 per cent. after the first burn, 22 per cent. after the sixth. This enlargement was due to two factors: (1) an increase in porosity and (2) an expansion of the mineral itself, resulting from the gradual transformation of the quartz into cristobalite and tridymite. Only the latter effect is shown in the figures of Table 6, calculated from specific gravities of samples of quartzite heated repeatedly to 1,450° in a porcelain kiln.[25]

### *Texture*

The insufficiency of analyses and melting-point determinations to indicate the value of a given quartzite for refractory purposes led Wernicke and Wildschrey[26] to seek an explanation in extensive microscopic studies of the texture of the rock. Their conclusions are as follows:

Typical quartzites, consisting principally of differently oriented intergrown quartz grains fairly uniform in size, metamorphosed quartzites, or those showing undulatory extinction under the microscope, can not be used. Good quartzites consist of quartz grains, mostly rounded, in a groundmass or cement of amorphous silica or crypto-crystalline quartz. No muscovite was found in such quartzites. Since expansion upon heating presents no difficulties in the case of quartzites containing a cement, only their melting point need be considered. They expand without cracking and acquire nearly their whole expansion on the first burn. In consequence of the fineness of division of the impurities in the cement and the slight sintering caused thereby, they are much stronger after burning than the typical quartzites in which the impurities are not so finely divided. The latter crack on burning and attain their complete expansion more slowly.

Grum-Grzimailo,[27] on the other hand, expresses the opinion that any quartzite with more than 94 to 95 per cent. $SiO_2$ may be used. He states that the purer the material and the larger the single quartz grains the longer must be the time of burning; the finer the quartz and its impurities the more rapidly will the required changes take place on heating.

Wernicke and Wildschrey's views are not supported by microscopic examinations of the most important American quartzites as shown later in this paper.

## Silica Brick

The material herein discussed is termed "silica brick" in American practice, and is made from quartzite of 96 to 98 per cent. $SiO_2$ content to

[25] Calculated by the writer from data of K. Endell: *Stahl und Eisen*, vol. 33, p. 1774 (Oct., 1913).

[26] Wernicke and Wildschrey: Quartzite and Its Application in the Refractories Industry, *Tonindustrie Zeitung*, vol. 34, pp. 688, 723 and 768 (1910).

Wernicke: Quartzite and Silica Brick, *Stahl und Eisen*, vol. 33, p. 1860 (Nov., 1913).

[27] *Stahl und Eisen*, vol. 31, p. 225 (1911).

which about 2 per cent. of lime is added to serve as a bond. It is not to be confused with the so-called "quartzite" brick, which contains much less silica, and in which fire clay is the bonding material.

### *Manufacture.*

The processes of manufacture are discussed by Seaver,[28] Hofman,[28] and Havard.[28] After grinding in a wet pan, where the lime is added, the charge is molded into brick, dried, and burned in a down-draft kiln.

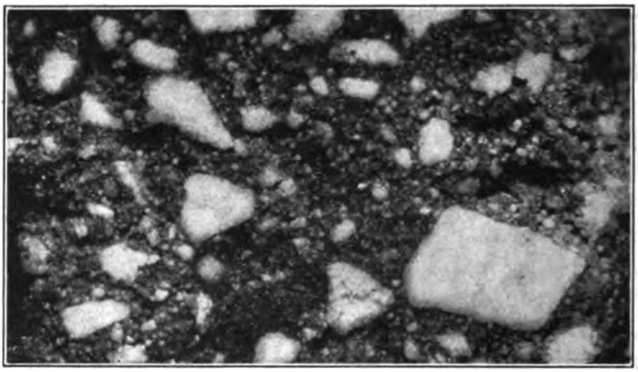

FIG. 8.—PHOTOMICROGRAPH OF POLISHED SURFACE OF 9-IN. SILICA BRICK. × 2.

The fineness of grinding is determined by the character of brick to be made; the grind used for special shapes has a larger proportion of fines

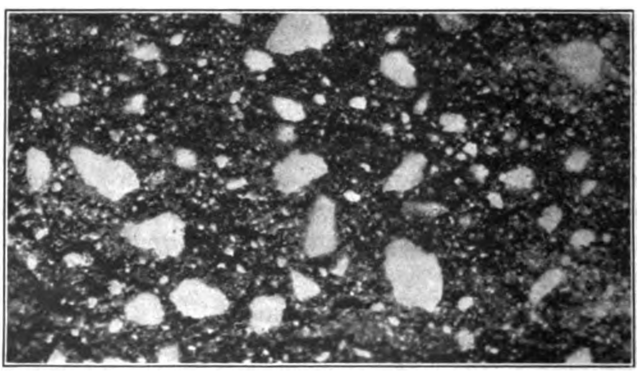

FIG. 9.—PHOTOMICROGRAPH OF POLISHED SURFACE OF SILICA SHAPE. × 2.

than that used for standard 9-in. brick. In either case, the ground material consists of a mixture of grains varying in size from that of a pea to the finest powder.[29] All silica brick are made in molds smaller than

[28] See Bibliography.
[29] See screen analyses, Table 13.

the proposed size of the finished brick; 3/8 to 7/16 in. per foot is allowed for expansion in burning.

The brick must be burned to such a temperature that as much as possible of the permanent expansion shall be acquired in the kiln, and not later in the furnace. The burning temperature is usually given as cone 15 to 16, or 16 to 17, and the time of burning as 20 days, including the time required for cooling. Lange[30] states that cone 15 should be melted down, cone 16 beginning to melt.

### Properties of Silica Brick

*Analysis.*—A typical analysis[31] of brick made from Pennsylvania rock is as follows:

| | Per Cent. |
|---|---|
| $SiO_2$ | 96.25 |
| $Al_2O_3$ | 0.88 |
| $Fe_2O_3$ | 0.79 |
| CaO | 1.80 |
| MgO | 0.14 |
| Alkalies | 0.39 |
| | 100.25 |

Brick with less than 94 per cent. $SiO_2$ or more than 3 per cent. CaO do not have the necessary refractory qualities; if less than 1.5 per cent. CaO is present, the brick are weak.

*Density.*—Principally on account of the progressive transformation of quartz into other modifications of silica upon the application of heat, the density of a silica brick is dependent upon the time and temperature of burning. Within certain limits, long-continued heating or burning at higher temperatures causes the density to diminish. The specific gravity of a silica brick should presumably lie between 2.65 (the value for quartz) and 2.28 (the value for tridymite). Microscopic cracks permeating the material render accurate determinations difficult, and results obtained in the ordinary way by the use of large pieces are quite sure to be inaccurate, even when the sample has been previously boiled.

For more accurate results, the finely powdered substance, boiled to remove adherent air bubbles, is usually tested in a pycnometer. Although somewhat laborious, this method with homogeneous materials gives results accurate in the second decimal place. As burned quartz is not homogeneous (see Figs. 16 and 17), and as the very fine cracks and pores are not always filled by the pycnometer liquid, the accuracy of the determinations in the case of silica brick is considered to be ± 0.02.[32]

[30] *Stahl und Eisen*, vol. 32, p. 1729 (Oct., 1912).

[31] K. Seaver: *Op. cit.*, p. 134.

[32] K. Endell: *Stahl und Eisen*, vol. 33, p. 1773 (Oct., 1913).

In this way Goerens and Gilles[33] determined the specific gravity of certain samples as 2.44. The figures given for the volume density of silica brick range from 1.50 to 1.88, averaging about 1.67 for a well-burned 9-in. brick. The values quoted by different writers for the "true specific gravity" of this material are quite inconsistent, varying from 2.05[34] to 2.75.[35]

Inconsistencies inexplicable by variations in the materials studied may perhaps be explained on the following grounds:

Let $W$ = weight of specimen in grams.
$V$ = entire volume of body in cubic centimeters.
$V_s$ = volume of rock substance.
$V_p$ = volume of pores.
$V_o$ = volume of open pores, which absorb water.
$V_c$ = volume of closed pores and cracks, which absorb no water.
$V_p = V_o + V_c$.
$V = V_s + V_p = V_s + V_o + V_c$.

It is evident that three possible values may be obtained for the specific gravity.

1. $G_b$. Bulk specific gravity $= \dfrac{W}{V} = \dfrac{W}{V_s + V_o + V_c}$.

2. $G$. True specific gravity $= \dfrac{W}{V_s}$. As already pointed out, for silica brick this value should probably lie somewhere between 2.28 and 2.65, depending on the conditions of burning.

3. $G'$. Apparent specific gravity, determined from the weights of dry specimen and water displaced after saturation. Apparent specific gravity $= \dfrac{W}{V_s + V_c}$. Its value lies between that of $G$ and $G_b$ and is dependent on the relation of closed to open pore space. When $V_c = O$, $G' = G$; when $V_o = O$, $G' = G_b$.

In many cases, the values quoted for the "true specific gravity" of silica brick are obviously too low. It is not improbable that in such cases the value designated above as $G'$ has been considered to be the true density and so reported.

*Porosity.*—The porosity of silica brick varies within wide limits, dependent upon the methods of manufacture. The importance of this property lies in its relation to thermal conductivity and permeability

[33] Goerens and Gilles, *Ferrum*, vol. 12, p. 17 (Nov., 1914).

[34] F. T. Havard: *Op. cit.*, p. 37.

[35] Wologdine and Queneau: Conductivity, Porosity, and Gas Permeability of Refractory Materials, *Electrochemical and Metallurgical Industry*, vol. 7, p. 433 (Oct., 1909).

to gases.[36] Under ordinary conditions pore space acts as a heat insulator.[37] Following the notation of the preceding paragraph,

Total pore space in per cent. of volume of body $= P_t = \frac{100\,V_p}{V} = 100\,\frac{(1 - G_a)}{G}$.

Open pore space in per cent. $= P_o = \frac{100\,V_o}{V} = 100\,\frac{(1 - G_b)}{G'}$.

$P_o$ is determined by absorption tests; $P_t$, by ascertaining the true specific gravity. Many writers upon silica brick do not make this distinction, but assume that $P_o = P_t$. The figures commonly given for porosity range from 18 to 43 per cent. of volume of the brick.

*Strength.*[38]—The figures in Table 7 are taken from Havard.

TABLE 7.—*Figures Taken from Havard**

| Silica Brick Made in | Crushing Strength, Pounds per Square Inch | | |
|---|---|---|---|
| | Side | Edge | End |
| Illinois | 2,345 | 1,546 | 2,119 |
| Pennsylvania | 2,896 | 1,044 | 1,551 |
| Missouri | 2,370 | 1,792 | 1,803 |

Seaver[39] studied the effect of reburning on crushing strength and modulus of rupture with silica bricks made in Pennsylvania burned repeatedly to 1,540° ±. His results (Table 8) show that with brick burned three times, the strength of the brick is increased by each reburning.

TABLE 8

| Number of Burns | Average Modulus of Rupture, Pounds per Square Inch. Brick Stood on Edge, Supports 6 In. apart | Average Crushing Strength, Pounds per Square Inch. Brick Tested Flatwise |
|---|---|---|
| 1 | 624 | 4,313 |
| 2 | 809 | 4,396 |
| 3 | 1,001 | 4,500 |

At high temperatures, the strength of a brick is much less than at

* F. T. Havard: *Op. cit.*, p. 37.

[36] Wologdine and Queneau: *Op. cit.*
Goerens and Gilles: *Op. cit.*

[37] Dougill, Hodsman and Cobb: Thermal Conductivity of Refractory Materials, *Journal of the Society of Chemical Industry*, vol. 34, pp. 465–471 (May, 1915).

[38] See also under Cross-breaking and Crushing Tests.

[39] K. Seaver: *Op. cit.*, p. 138.

ordinary temperatures.[40] Subjected to a load of 50 lb. per square inch and gradually heated, chrome bricks fail suddenly at 1,450°, magnesia bricks at 1,550°. Silica bricks under the same conditions are unaffected at 1,450°.

*Melting Point.*—The melting interval of silica brick is generally considered to lie between cones 34 and 36 (1,740 to 1,775° C.)[41]; Havard[42] states that silica brick softens between 1,750 and 1,800°. Kanolt[43] measured the temperature at which pieces of hazelnut size visibly changed their form. For three different pieces the temperatures were 1,700°, 1,705°, 1,700°. Endell[44] tested brick consisting essentially of cristobalite, as well as brick consisting essentially of tridymite. Pea-sized pieces of both became completely melted at 1,700° ± 10.

The apparent discrepancy between the melting temperature of silica brick as determined in the laboratory, and the temperatures to which they are sometimes subjected in practice, is explained as follows:[45] Only the immediate surface of the brick reaches the furnace temperature. This is soon melted when 1,700° is exceeded, forming a viscous glaze, which protects the portion of the brick immediately adjacent. This portion does not reach its melting point on account of the temperature drop toward the exterior of the furnace.

TABLE 9.—*Specific Heat of Silica Brick**

| $t°$ | Heyn | | Steger | Norton† |
|---|---|---|---|---|
| | True Specific Heat at $t°$ | Mean Specific Heat between 20° and $t°$ | Mean Specific Heat between 0° and $t°$ | Mean Specific Heat between 20° and $t°$ |
| 0° C. | 0.200 | 0.200 | | |
| 200° C. | 0.237 | 0.219 | 0.205 | |
| 400° C. | 0.270 | 0.237 | | |
| 600° C. | 0.282 | 0.250 | | 0.225 |
| 800° C. | 0.285 | 0.261 | | 0.236 |
| 1,000° C. | 0.298 | 0.262 | | |
| 1,100° C. | | | | 0.255 |
| 1,200° C. | 0.291 | 0.266 | | |

* E. Heyn: Thermal Conductivity of Refractory Building Materials, *Mitteilungen aus dem Königlichen Materialprüfungsamt*, vol. 32, p. 180 (1914).

W. Steger: Specific Heats of Refractory Products, *Silikat Zeitschrift*, vol. 2, No. 11, p. 203 (Nov., 1914).

† C. L. Norton: Private communication to K. Seaver.

[40] G. H. Brown: Note on Load Tests Made on Magnesite, Chrome and Silica Brick, *Transactions of the American Ceramic Society*, vol. 14, pp. 391–393 (1912).

[41] K. Endell: *Stahl und Eisen*, vol. 33, p. 1856 (Nov., 1913).

[42] F. T. Havard: *Op. cit.*, p. 29.

[43] *U. S. Bureau of Standards, Technologic Paper* 10 (1912).

[44] K. Endell: *Op. cit.*, pp. 1857, 1858.

[45] K. Endell: *Op. cit.*, pp. 1857, 1858.

*Thermal Conductivity.*—The divergence in the results obtained by different observers in the determination of thermal conductivity has occasioned considerable discussion. As Dudley points out, these variations are due to variations in the materials tested, as well as to differences between, or inaccuracies of, the methods employed. For comparative purposes, the coefficients are given in Table 10 for some fire-clay brick also. The coefficient $k$ represents the flow of heat in calories per second, per square-centimeter area, through 1-cm. thickness, for a temperature difference of 1° C. To convert these figures into B.t.u. per 24 hr. per square-foot area per inch thickness per 1° F. temperature difference, they should be multiplied by 69,700.

TABLE 10

| Material | Conductivity | | | Reference |
|---|---|---|---|---|
| | Temperature | | Mean $k$ between $t_1$ and $t_2$ | |
| | $t_1$ | $t_2$ | | |
| Silica brick, "Star" brand 95.9 per cent. $SiO_2$ | 0 | 100 | 0.0021 | Calculated from results of Boyd Dudley, Jr. |
| | 0 | 1,000 | 0.0031 | |
| Clay brick, "Woodland" brand 52.9 per cent. $SiO_2$, 42.7 per cent. $Al_2O_3$ | 0 | 100 | 0.0016 | *Transactions of the American Electrochemical Society,* vol. 27, p. 336 (1915). |
| | 0 | 1,000 | 0.0025 | |
| "Quartzite" brick, 73.9 per cent. $SiO_2$, 22.9 per cent. $Al_2O_3$. | 0 | 100 | 0.0020 | |
| | 0 | 1,000 | 0.0027 | |
| Silica brick, 96 per cent. $SiO_2$, specific gravity, 2.44 | 0 | 100 | 0.0028 | Calculated from results of Goerens and Gilles, *Ferrum,* vol. 12, pp. 1, 17 (Oct., Nov., 1914). |
| | 0 | 1,000 | 0.0031 | |
| Claybrick, 53.9 $SiO_2$, 40.2 $Al_2O_3$ | 0 | 100 | 0.0022 | |
| | 0 | 1,000 | 0.0027 | |

Notwithstanding the lack of numerical correspondence in results, Dudley, as well as Goerens and Gilles, finds that the coefficients are greater at high temperatures than at low, and that the conductivity of silica brick is greater than that of clay brick. The latter conclusion is corroborated by Brown,[46] who compared the flow of heat through the following materials:

| | | |
|---|---|---|
| Silica brick................ | 95.5 per cent. $SiO_2$, | 1.9 per cent. $Al_2O_3$. |
| Quartzite brick............ | 83.4 per cent. $SiO_2$, | 11.8 per cent. $Al_2O_3$. |
| Clay brick................. | 63.2 per cent. $SiO_2$, | 33.0 per cent. $Al_2O_3$. |

Cylindrical specimens were heated at one end to 1,300° C. while the other end was exposed to the atmosphere at 31° C. Under these condi-

[46] G. H. Brown: Relative Thermal Conductivities of Silica and Clay Refractories, *Transactions of the American Ceramic Society,* vol. 16, pp. 382–385 (1914).

tions, after equilibrium had been attained, the conductivity of the silica brick was 9 per cent. greater than that of the clay brick, and 3 per cent. greater than that of the quartzite brick.

Wologdine[47] shows that the conductivity of both silica and clay refractories increases with higher temperatures of burning. His figures indicate that silica is a poorer conductor than clay, for material burned at, or lower than 1,300°. Marshall[48] quite properly holds that while Wologdine's data are probably correct for the materials investigated, his figures are not practically applicable in a comparison of American refractories, since commercial silica brick are burned at a higher temperature than 1,300°.

The data given by Boyd Dudley, Jr.,[49] are of particular interest, in that the materials studied are three well-known brands of American refractories. The mean conductivities between any two temperatures $t_1$ and $t_2$ can be calculated from his results expressed by the following straight-line formula based upon measurements taken up to about 1,000° C.:

"Woodland" brick.................... $k = 0.0015 + 0.0_{6}10\ (t_1 + t_2)$
"Quartzite" brick.................... $k = 0.0019 + 0.0_{6}08\ (t_1 + t_2)$
"Star" silica brick.................... $k = 0.0020 + 0.0_{6}11\ (t_1 + t_2)$

These formulæ should be used with caution above 1,000°, and the formula for silicia brick should probably not be applied above 1,050°, as Heyn[50] finds a sharp rise in the conductivity of this material between 1,050° and 1,100°.

*Expansion.*—The problem of the expansion of silica brick is thus stated by Seaver:[51] In the process of manufacture, a permanent expansion occurs under the influence of heat, swelling the brick from the mold or "green" brick size to the required or "burned" size. It is desired that the permanent expansion should, if possible, be completed in the burning, so as to prevent further swelling in subsequent use. "In every burned brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. This thermal or temporary expansion occurring upon heating a burned brick is not to be confused with the permanent expansion produced in the body of the brick during its manufacture. It is argued that if a silica brick is not properly burned,

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[47] *Electrochemical and Metallurgical Industry*, vol. 7, pp. 383 and 433 (Sept.-Oct., 1909).

[48] S. M. Marshall: Relative Thermal Conductivity of Silica and Clay Brick, *Metallurgical and Chemical Engineering*, vol. 12, p. 74 (Feb., 1914).

[49] See Table 10.

[50] *Mitteilungen aus dem Königlichen Materialprüfungsamt*, vol. 32, p. 181 (1914),

[51] K. Seaver: *Op. cit.*, p. 133.

then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion, but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for."

Since the final increments of permanent expansion are acquired but slowly under the influence of high temperatures it is found that reburned brick show a slight increase in volume after each of several heat treatments. It is generally held that the manufacturer can do nothing to diminish the temporary expansion.[52]

The linear expansion in the burning from "green brick" size to size of finished product is 3/8 to 7/16 in. per foot, that is, 3.1 to 3.6 per cent. It may be here noted that the change of volume in the complete transformation of quartz to cristobalite corresponds to a linear expansion of 4.35 per cent.; quartz to tridymite, 5.3 per cent.

With a number of brick burned repeatedly to cone 16 to 17, Cramer[53] found an average permanent linear expansion of 3.2 per cent. on the first burn, an increment of 0.6 per cent. on second burn and 0.2 per cent. on the third. In a test mentioned by Havard,[54] a Pennsylvania silica brick reheated to 1,400° C. had a total linear expansion of 1.7 per cent., of which 0.4 per cent. was permanent, 1.3 per cent. temporary. Stockman and Foote,[55] at the Massachusetts Institute of Technology, studied the temporary thermal expansion of two brands of silica brick with the results shown in Table 11.

TABLE 11

| Brand | Average Total Linear Expansion at | | | |
|---|---|---|---|---|
| | 300°, | 600°, | 900°, | 1,200°, |
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| I. Reese & Sons | 0.9 | 1.5 | 1.9 | |
| Star | ... | 0.7 | 0.9 | 1.1 |

Practical experience shows that the thermal expansion of well-burned brick amounts to 3/16 to 1/4 in. per foot (1.6 to 2.1 per cent.) of which the greater part occurs below 600° C.

"In laying silica brick it is important to leave room for expansion;

[52] Experimental results hereinafter given indicate that this view is subject to certain qualifications.

[53] *Tonindustrie Zeitung*, vol. 25, p. 876 (1901); *Stahl und Eisen*, vol. 21, p. 772 (July, 1901).

[54] F. T. Havard: *Op. cit.*, p. 37.

[55] Massachusetts Institute of Technology Mining Department Thesis No. 226, 1902.

thus in a reverberatory furnace for melting sulphide-copper ores, having 12-in. silica brick, ¼ in. is allowed to 1 ft. in the roof, 5/16 in. to 2 ft. in the bottom; the expansion space is occupied by cardboard when the brick is being laid."[56]

*Effect of Rapid Temperature Change.*—Closely connected with the expansion on heating is the inability of silica brick to withstand rapid changes of temperature. For this reason, both while being manufactured and later when put into use, heating and cooling must take place evenly and slowly. "By quick burning" (in the kilns) "brick may be easily expanded by double or treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick . . . In the early stages of firing the heat must be raised very slowly."[57] If cooled too rapidly, silica brick become friable, crack and spall. An underburned brick is less likely to spall than a well-burned one, but is weak, and has an excessive expansion, largely of a permanent character, when heated in the furnace.

## *Constitution*

The constitution of silica brick and its relation to the phenomena of expansion have engaged the attention of many investigators. K. Endell[58] gives a historical sketch of the most important work done in this connection. In 1890, E. Mallard[59] discovered tridymite crystals in brick which had been in use a year and a half; this observation was confirmed in 1911 by Grum-Grzimailo.[60] The latter found that silica brick, when first made, consisted of grains of unaltered quartz surrounded by an apparently amorphous groundmass, which was held to be glass, together in some cases with varying amounts of tridymite; upon several months' use in the open-hearth furnace, the brick gradually and completely went over into a mass of interlocking twin crystals of tridymite. To avoid continual expansion in the furnace it was proposed to burn the brick in such a way as to effect the complete transformation to tridymite in the kilns.

Holmquist[61] discovered cristobalite in brick which had melted and hung down from the roof of the furnace as stalactites.

Endell[62] shows that the texture and constitution of silica brick are quite different before use, after 10 charges in the open-hearth furnace, after 50 charges, and again after having been melted. After the first burn, about two-thirds of the material was found to be converted into an

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[56] H. O. Hofman: *General Metallurgy*, p. 368, McGraw-Hill Book Co., New York, 1913.

[57] K. Seaver: *Op. cit.*, p. 139.

[58] *Stahl und Eisen*, vol. 32, p. 392 (Mar., 1912).

[59] *Bulletin, Société française de minéralogie*, vol. 13, p. 172 (1890).

[60] *Stahl und Eisen*, vol. 31, pp. 224–226 (Feb., 1911).

[61] *Tonindustrie Zeitung*, vol. 35, p. 1325 (1911).

[62] *Stahl und Eisen*, vol. 32, p. 392–397 (Mar., 1912).

apparently isotropic modification of silica, enclosing unaltered quartz grains of unequal size. This apparently isotropic form was at first believed to be glass, but later shown to be cristobalite.[63] Since the transformation of the original quartz was not complete in the initial burn, the conclusion was reached that only about two-thirds of the expansion takes place in the kilns, and the rest must occur in the furnace. In a brick which had been through 10 charges in the open-hearth furnace, the groundmass was found to be converted to a large degree into tridymite, while the larger quartz grains were transformed into the apparently isotropic condition (cristobalite). After about 50 charges, a good silica brick consisted almost wholly of interlaced tridymite crystals. A brick was examined in which a rind 2 or 3 cm. thick had been melted, while the rest of the piece showed no signs of fusion. Three zones could be distinguished. The one lying nearest the heat had been fully melted, and later, in cooling, crystallized out in part as cristobalite, which was surrounded by a yellow iron-containing glass. The birefringence and other crystal properties of cristobalite formed in this way were much more easily recognized than with cristobalite formed by the transformation of quartz. The colorless cristobalite grains were penetrated by pearly cracks and twinned in a very complicated manner. In the thermal microscope they became isotropic at 220°. Analysis of this fused zone showed a silica content of but 90 per cent.; vaporization of $SiO_2$ had presumably been favored by intermediate formation of silicon carbide. The central zone showed the typical constitution of a tridymite brick which still contains some cristobalite. As the outer zone was approached, grains of quartz appeared; the latter zone contained considerable unaltered quartz in a groundmass of cristobalite, and in constitution appeared like an unused brick. It had evidently been kept cool by the outside air.

Endell[64] studied also the changes in the constitution of quartzites after repeated burning in a porcelain kiln to 1,450° C. It was found that the finely divided impurities of the quartzite serve as inversion centers and hasten the transformations.

Rock heated three times was converted principally into cristobalite, with some unaltered quartz grains remaining. The latter went over into cristobalite on longer heating and caused a continual expansion. After 10 heatings, there were still a few unaltered quartz particles, and a small proportion of tridymite had appeared in the form of the characteristic wedge-shaped twin crystals. Quartzite burned four times in the porcelain kiln, and later heated 2 hr. to 1,600° was completely transformed into cristobalite.

Seaver and Klein[65] made quantitative determinations of the constitution of silica brick and quartzite burned repeatedly to 1,540° C.,

[63] K. Endell: *Stahl und Eisen*, vol. 33, p. 1856 (Nov., 1913).
[64] *Op. cit.*, p. 1856.
[65] K. Seaver: *Op. cit.*, p. 137.

that temperature being maintained each time for about 40 hr. Their results are shown in Table 12.

TABLE 12

| | Silica Brick | | | Quartzite | |
|---|---|---|---|---|---|
| | Burn | | | Burn | |
| | 1, Per Cent. | 2, Per Cent. | 3, Per Cent. | 1, Per Cent. | 2, Per Cent. |
| Per cent. cristobalite | 77 | 83 | 84 | 49 | 69 |
| Per cent. quartz and silicates | 23 | 17 | 16 | 51 | 31 |

No tridymite was to be expected in these tests, since that mineral is not known to be formed under any conditions above 1,470° C.

The relation of the constitution of silica brick to the expansion on heating is indicated by the facts that the volume increase in the transformation of quartz to cristobalite is 13.6 per cent.; quartz to tridymite, 16.7 per cent.

## MICROSCOPIC STUDY AND EXPERIMENTAL DATA

### Microscopic Examination of Quartzites

The accompanying photomicrographs represent the texture of specimens of quartzite from the deposits most important in the American refractories industry. All of these quartzites consist essentially of individual quartz crystals of random orientation closely packed together and interlocking with sharp angles. No trace of a groundmass or binding material is to be seen, except in a few small and isolated areas, which do not appear in the figures. Small inclusions are visible in great number. The accessory minerals are extremely small in amount, and have not been studied.

Fig. 10 shows the Medina quartzite from Mount Union, Huntingdon Co., Pa. This consists essentially of a mass of interfering quartz crystals, each with a sand grain as a nucleus. The rounded or subangular outlines of the original sand grains are marked by a row of inclusions, and are perfectly distinct. The spaces between these are filled with secondary quartz which has attached itself to the original grains in crystalline continuity with them, so that the original quartz grain and the new quartz border have the same extinction positions and interference colors. The borders contain but few inclusions, and consequently are much clearer than the nuclei.

The Baraboo Quartzite from Ablemans, Wis., is shown in Fig. 11. The grain is much coarser than that of the other quartzites examined, and the line along which the interlocking individuals join is peculiarly jagged. Many of the grains show a marked undulatory extinction with crossed nicols; that is, the whole of the crystal is not extinguished at

the same time, but a wave-like shadow passes over it on rotating the stage.

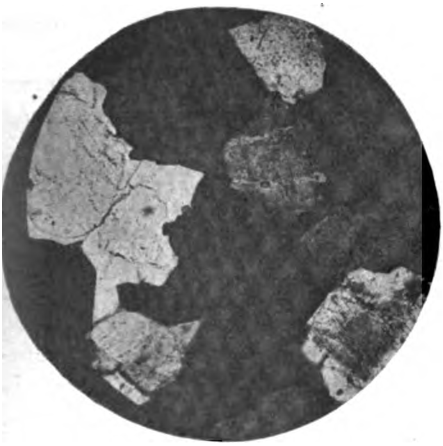


FIG. 10.

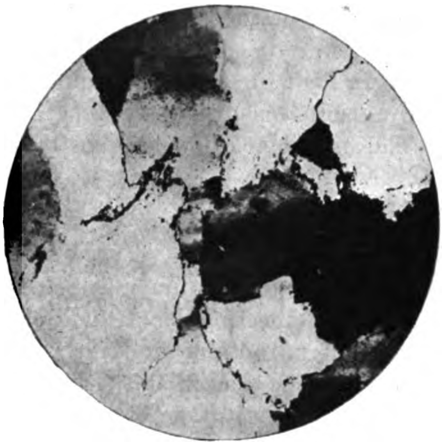


FIG. 11.

FIG. 10.—MEDINA QUARTZITE FROM MT. UNION, HUNTINGDON CO., PENNSYLVANIA (WHITEHEAD). THE FIELD SHOWS A MASS OF INTERFERING QUARTZ CRYSTALS EACH WITH A SAND GRAIN AS A NUCLEUS. THE OUTLINES OF THE ORIGINAL GRAINS ARE WELL SHOWN IN THE TWO CRYSTALS TO THE LEFT. CROSSED NICOLS. × 48.

FIG. 11.—BARABOO QUARTZITE FROM ABLEMANS, WISCONSIN (WHITEHEAD). UNDULATORY EXTINCTION IS SHOWN BY A GRAIN NEAR THE CENTER AND BY ONE NEAR THE TOP OF THE SECTION. CROSSED NICOLS. × 48.

FIG. 12.

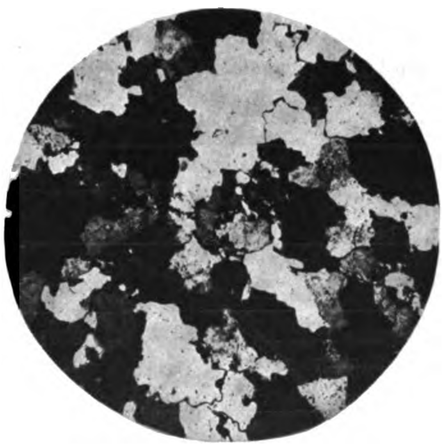


FIG. 13.

FIG. 12.—QUARTZITE FROM NORTHEASTERN ALABAMA, USED IN BRICK MAKING AT WYLAM, ALABAMA (WHITEHEAD). NOTE THE STRIATED APPEARANCE OF SOME OF THE GRAINS. CROSSED NICOLS. × 48.

FIG. 13.—QUARTZITE USED IN BRICK MAKING AT ANACONDA, MONT. (WHITEHEAD). CROSSED NICOLS. × 48.

The crystals of the Alabama quartzite contain numerous inclusions, arranged in nearly parallel lines, which with low magnifications appear under the microscope as fine striations. The Anaconda quartzite is extremely fine in grain.

The textural relations above described, and illustrated in the photomicrographs, do not sustain the conclusion[66] that only quartzites containing a fine-grained groundmass or cement can be used in the manufacture of high-grade silica refractories.

In Figs. 10, 11, 12, 13,[67] quartz grains in extinction positions are shown by black areas; quartz grains in other positions appear as lighter areas. The small black dots are mostly inclusions.

## Study of Silica Brick

### *Description of Test Brick*

The conditions under which the test brick were made were devised by the engineering department of the Harbison-Walker Refractories Co., with the purpose of studying the effects of variations in grind and temperature of burning, and of repeated burning, upon the physical properties and constitution of silica brick. Accordingly, three lots of 9-in. brick (approximately 9 by 4½ by 2½ in. in size) were made up. These lots will be hereinafter referred to as "regular", "medium", and "fine grind" respectively. The regular-grind specimens were made by the usual method for 9-in. brick; those of medium grind were given the grind used for silica shapes, and pounded into loose-side molds, as silica shapes are treated. The material used for the fine grind was screened through an 8-mesh hand screen, returned to the pan and reground long enough to add and mix lime thoroughly; the brick were pounded into loose-side molds. Screen analyses are shown in Table 13.

TABLE 13.—*Screen Analyses of Test Brick*

| Grind | Screen Numbers in Meshes per Linear Inch | | | | | |
|---|---|---|---|---|---|---|
| | Held on | | | | | Through |
| | 8, Per Cent. | 16, Per Cent. | 20, Per Cent. | 30, Per Cent. | 40, Per Cent. | 40, Per Cent. |
| Regular | 15.3 | 13.4 | 4.6 | 5.4 | 4.8 | 56.5 |
| Medium or shape | 10.2 | 13.1 | 2.2 | 5.0 | 4.7 | 64.8 |
| Fine | | 14.3 | 4.1 | 6.5 | 4.1 | 71.0 |

A number of brick from each lot were burned under the ordinary conditions of manufacture at three positions in the kiln, designated for convenience as positions $A$, $B$ and $C$. The brick were burned a number of times, being under fire approximately 10 days each time, and specimens were set aside for study after each burning.

[66] See under *Texture*.

[67] Figs. 10 to 12 inclusive are from specimens furnished by the Harbison-Walker Refractories Co. Fig. 13 is from specimen furnished by the Anaconda Copper Mining Co.

The temperatures attained on the various burns as measured by Orton-Seger cones are shown in Table 14.

TABLE 14.—*Temperature Measured by Orton-Seger Cones*

| Number of Burn | Position | | |
|---|---|---|---|
| | *A* | *B* | *C* |
| 1 | Cone 15 | Cone 13 | Cone 14 |
| 2 | Cone 15 | Cone 13 | Cone 14 |
| 3 | Cone 15 | Cone 15 | Cone 15 |
| 4 | Cone 13 | Cone 14 | Cone 14 |
| 5 | Cone 15 | Cone 15 | Cone 15 |
| 6 | Cone 16 | Cone 14 | Cone 14 |

The fusion points of these cones are as follows:

| Cone | Orton Scale, Degrees Centigrade | Seger Revised Scale,[a] Degrees Centigrade |
|---|---|---|
| 13 | 1,390 | 1,380 |
| 14 | 1,410 | 1,410 |
| 15 | 1,430 | 1,435 |
| 16 | 1,450 | 1,460 |

[a] Advertisement in 1914 Calendar of *Tonindustrie Zeitung*

The fusion points in silica brick kilns probably average 70° lower than the figures given by the manufacturers,[68] but can not be told with exactness, since cones give very erratic results when used in such kilns.[69]

## *Physical Tests*

*Density and Porosity.*—The bulk and apparent specific gravities and per cent. of open pore space of certain brick of the first burn are shown in Table 15. The term "apparent" specific gravity, is used in the sense defined earlier in this paper under the heading, "Properties of Silica Brick." The composition of the brick (hereinafter discussed) suggests that the value of the true specific gravity, of which no determinations were made, probably lies between 2.40 and 2.45. The per cent. of open pore space was calculated from the two specific gravities (see discussion under heading just mentioned).

From the figures given it appears that the greatest porosity is shown by the coarse (regular) grind, and the least by the medium (shape) grind.

[68] Hoffman: *Tonindustrie Zeitung*, vol. 35, pp. 1099–1100 (1911).

S. Geijsbeek: Melting Points of Pyrometric Cones under Various Conditions, *Transactions of the American Ceramic Society*, vol. 14, 849–871 (1912).

[69] K. Seaver: *Op. cit.*, p. 133.

The data are insufficient to justify any conclusions as to the effect upon porosity of variations in burning temperature.

TABLE 15.—*Tabulated Data: Density and Porosity*

| First Burn | | | | | | | |
|---|---|---|---|---|---|---|---|
| Mark | Grind | Burned to Cone | Bulk Specific Gravity | Apparent Specific Gravity | Open Pore Space, Per Cent. of Volume | Open Pore Space, Mean Per Cent. of Volume | Open Pore Space, Mean Per Cent. of Volume |
| 83–A–1 | Regular | 15 | 1.678 | 2.259 | 25.7 | 26.1 | |
| 279–A–1 | Regular | 15 | 1.655 | 2.254 | 26.5 | | |
| 69–C–1 | Regular | 14 | 1.637 | 2.229 | 28.5 | 28.5 | |
| 228–B–1 | Regular | 13 | 1.671 | 2.251 | 25.7 | 25.7 | 26.8 |
| 585–A–1 | Medium | 15 | 1.770 | 2.247 | 21.3 | 21.8 | |
| 601–A–1 | Medium | 15 | 1.748 | 2.249 | 22.3 | | |
| 679–C–1 | Medium | 14 | 1.723 | 2.232 | 22.7 | 22.7 | |
| 443–B–1 | Medium | 13 | 1.839 | 2.277 | 19.2 | 19.2 | 21.2 |
| 1,148–A–1 | Fine | 15 | 1.757 | 2.272 | 22.7 | 22.5 | |
| 1,149–A–1 | Fine | 15 | 1.765 | 2.272 | 22.3 | | |
| 854–C–1 | Fine | 14 | 1.725 | 2.242 | 23.2 | 23.2 | |
| 997–B–1 | Fine | 13 | 1.774 | 2.264 | 21.7 | 21.7 | 22.5 |

### *Cross-Breaking and Crushing Tests*

In the transverse tests the brick were laid edgewise with a span of 6 in. and the load applied at midspan. Modulus of rupture was calculated from the formula $R = \frac{3\,Wl}{2\,bd^2}$, in which $l$ is the distance between supports in inches, $b$ the breadth and $d$ the depth of the specimen in inches, and $W$ the load in pounds at which failure occurred. Compression tests were made on half bricks resulting from the transverse tests. The bricks were bedded flatwise on blotting paper in the testing machine to secure a uniform bearing surface. The crushing strength was obtained by dividing the crushing load in pounds by the mean area in square inches of the two surfaces of the brick.

Unfortunately, but three brick of each lot were available for these tests, a number insufficient to permit the determination of the mean strengths with the proper degree of accuracy. The proposed specifications[70] of the American Society for Testing Materials direct that at least five specimens shall be used in transverse and compression tests upon building brick, and the same requirement should undoubtedly apply to the testing of refractory brick.

The results obtained are shown in Tables 16, 17, and 18.

[70] *Proceedings of the American Society for Testing Materials*, vol. 13, p. 284 (1913).

TABLE 16.—*Sample Log and Data Sheet*
*Cross-Breaking and Crushing Tests*

| Cross-Breaking Span, 6 In. | | | | | | Crushing: Half Brick Crushed Flat | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Mark | Breadth of Brick, In. | Depth of Brick, In. | *W* | *R* | Mean *R* | Crushing Area, Square Inches 1 | 2 | Mean | *W* | *C* | Mean *C* |
| 83–A–1 | 2.48 | 4.45 | 4,970 | 910 | | 19.80 | 17.58 | 18.69 | 51,900 | 2,780 | |
| 279–A–1 | 2.46 | 4.45 | 4,820 | 890 | 920 | 19.54 | 17.97 | 18.75 | 45,600 | 2,430 | 2,680 |
| 289–A–1 | 2.52 | 4.43 | 5,260 | 960 | | 16.46 | 16.16 | 16.31 | 46,100 | 2,830 | |
| 585–A–1 | 2.66 | 4.73 | 7,010 | 1,060 | | 21.75 | 21.33 | 21.54 | 101,000 | 4,690 | |
| 601–A–1 | 2.66 | 4.71 | 7,450 | 1,140 | 1,050 | 21.18 | 20.24 | 20.71 | 120,500 | 5,820 | 4,780 |
| 715–A–1 | 2.68 | 4.71 | 6,320 | 960 | | 20.26 | 20.00 | 20.13 | 77,000 | 3,830 | |

*W* = load causing failure in pounds.
*R* = modulus of rupture, pounds per square inch.
*C* = compressive strength, pounds per square inch.

TABLE 17.—*Collected Data*
*Cross-Breaking and Crushing Tests*

First burn — Brick Tested Edgewise 6-in. Span

| Mark | Grind | Burned to Cone | Cross-Breaking, Load in Pounds | Modulus of Rupture: Modulus, Pounds per Square Inch | Mean | Average Deviation, Per Cent. of Mean | Crushing Strength, Brick Laid Flat: Crushing Strength, Pounds per Square Inch | Mean | Average Deviation, Per Cent. of Mean |
|---|---|---|---|---|---|---|---|---|---|
| 83–A–1 | Regular | 15 | 4,970 | 910 | | | 2,780 | | |
| 279–A–1 | Regular | 15 | 4,820 | 890 | 920 | 3 | 2,430 | 2,680 | 6.4 |
| 289–A–1 | Regular | 15 | 5,260 | 960 | | | 2,830 | | |
| 135–B–1 | Regular | 13 | 3,240 | 600 | | | 2,680 | | |
| 228–B–1 | Regular | 13 | 3,630 | 670 | 690 | 10 | 2,390 | 2,730 | 8.5 |
| 327–B–1 | Regular | 13 | 4,280 | 800 | | | 3,120 | | |
| 69–C–1 | Regular | 14 | 4,010 | 730 | | | 2,210 | | |
| 102–C–1 | Regular | 14 | 4,140 | 750 | 770 | 5 | 2,050 | 2,100 | 3.3 |
| 156–C–1 | Regular | 14 | 4,520 | 820 | | | 2,040 | | |
| 585–A–1 | Medium | 15 | 7,010 | 1,060 | | | 4,690 | | |
| 601–A–1 | Medium | 15 | 7,450 | 1,140 | 1,050 | 6 | 5,820 | 4,780 | 14.0 |
| 715–A–1 | Medium | 15 | 6,320 | 960 | | | 3,830 | | |
| 443–B–1 | Medium | 13 | 4,990 | 800 | | | 5,430 | | |
| 445–B–1 | Medium | 13 | 4,310 | 690 | 910 | 24 | 3,890 | 4,680 | 11.0 |
| 476–B–1 | Medium | 13 | 7,760 | 1,230 | | | 4,710 | | |
| 679–C–1 | Medium | 14 | 5,430 | 830 | | | 4,340 | | |
| 680–C–1 | Medium | 14 | 5,610 | 860 | 920 | 11 | 4,930 | 4,820 | 6.7 |
| 744–C–1 | Medium | 14 | 7,050 | 1,070 | | | 5,180 | | |
| 1,148–A–1 | Fine | 15 | 5,790 | 930 | | | 7,270 | | |
| 1,149–A–1 | Fine | 15 | 9,450 | 1,500 | 1,170 | 19 | 4,360 | 5,880 | 17.0 |
| 1,151–A–1 | Fine | 15 | 6,770 | 1,070 | | | 6,010 | | |
| 997–B–1 | Fine | 13 | 5,360 | 860 | | | 6,300 | | |
| 998–B–1 | Fine | 13 | 6,180 | 990 | 910 | 6 | 5,280 | 5,840 | 6.3 |
| 999–B–1 | Fine | 13 | 5,560 | 890 | | | 5,940 | | |
| 854–C–1 | Fine | 14 | 3,270 | 520 | | | 5,180 | | |
| 934–C–1 | Fine | 14 | 2,840 | 450 | 790 | 52 | 5,040 | 5,250 | 3.4 |
| 1,052–C–1 | Fine | 14 | 9,010 | 1,410 | | | 5,520 | | |

TABLE 17.—*Collected Data.—Continued*

Second burn

| Mark | Grind | Burned to Cone | Cross-Breaking, Load in Pounds | Modulus of Rupture: Modulus, Pounds per Square Inch | Modulus of Rupture: Mean | Modulus of Rupture: Average Deviation, Per Cent. of Mean | Crushing Strength, Brick Laid Flat: Crushing Strength, Pounds per Square Inch | Crushing Strength, Brick Laid Flat: Mean | Crushing Strength, Brick Laid Flat: Average Deviation, Per Cent. of Mean |
|---|---|---|---|---|---|---|---|---|---|
| 275-A-2 | Regular | 15-15 | 5,820 | 1,080 | | | 2,230 | | |
| 288-A-2 | Regular | 15-15 | 5,160 | 980 | | | 2,750 | | |
| 290-A-2 | Regular | 15-15 | 3,480 | 620 | 910 | 15 | 2,990 | 2,680 | 8.6 |
| 293-A-2 | Regular | 15-15 | 5,140 | 940 | | | 2,760 | | |
| 138-B-2 | Regular | 13-13 | 5,620 | 1,040 | | | 2,770 | | |
| 218-B-2 | Regular | 13-13 | 4,730 | 860 | 1,010 | 10 | 2,540 | 2,760 | 5.4 |
| 330-B-2 | Regular | 13-13 | 6,070 | 1,120 | | | 2,970 | | |
| 10-C-2 | Regular | 14-14 | 5,420 | 980 | | | 2,690 | | |
| 14-C-2 | Regular | 14-14 | 4,730 | 880 | 920 | 3 | 2,490 | 2,680 | 4.6 |
| 110-C-2 | Regular | 14-14 | 4,950 | 920 | | | 2,850 | | |
| 567-A-2 | Medium | 15-15 | 10,440 | 1,600 | | | 5,780 | | |
| 599-A-2 | Medium | 15-15 | 10,730 | 1,630 | 1,680 | 5 | 7,010 | 6,030 | 11.0 |
| 628-A-2 | Medium | 15-15 | 11,800 | 1,810 | | | 5,290 | | |
| 467-B-2 | Medium | 13-13 | 9,350 | 1,460 | | | 7,030 | | |
| 472-B-2 | Medium | 13-13 | 10,390 | 1,630 | 1,310 | 24 | 5,200 | 6,270 | 11.0 |
| 579-B-2 | Medium | 13-13 | 5,360 | 840 | | | 6,590 | | |
| 572-C-2 | Medium | 14-14 | 10,340 | 1,570 | | | 5,600 | | |
| 658-C-2 | Medium | 14-14 | 10,620 | 1,620 | 1,680 | 7 | 5,130 | 5,570 | 5.2 |
| 699-C-2 | Medium | 14-14 | 12,150 | 1,860 | | | 5,970 | | |
| 824-A-2 | Fine | 15-15 | 4,440 | 680 | | | 7,870 | | |
| 848-A-2 | Fine | 15-15 | 4,830 | 770 | | | 7,850 | | |
| 850-A-2 | Fine | 15-15 | 12,450 | 1,950 | 1,270 | 43 | 6,700 | 6,890 | 14.0 |
| 970-A-2 | Fine | 15-15 | 10,550 | 1,670 | | | 5,150 | | |
| 961-B-2 | Fine | 13-13 | 5,420 | 860 | | | 5,610 | | |
| 1,019-B-2 | Fine | 13-13 | 6,770 | 1,080 | 1,070 | 13 | 7,390 | 6,290 | 12.0 |
| 1,042-B-2 | Fine | 13-13 | 7,820 | 1,280 | | | 5,870 | | |
| 805-C-2 | Fine | 14-14 | 13,650 | 2,160 | | | 8,560 | | |
| 836-C-2 | Fine | 14-14 | 8,050 | 1,260 | 1,590 | 24 | 7,150 | 7,830 | 6.2 |
| 1,012-C-2 | Fine | 14-14 | 8,730 | 1,360 | | | 7,780 | | |

TABLE 17.—*Collected Data.—Continued*

Third burn

| Mark | Grind | Burned to Cone | Cross-Breaking, Load in Pounds | Modulus of Rupture: Modulus, Pounds per Square Inch | Modulus of Rupture: Mean | Modulus of Rupture: Average Deviation, Per Cent. of Mean | Crushing Strength, Brick Laid Flat: Crushing Strength, Pounds Per Square Inch | Crushing Strength, Brick Laid Flat: Mean | Crushing Strength, Brick Laid Flat: Average Deviation, Per Cent. of Mean |
|---|---|---|---|---|---|---|---|---|---|
| 39-A-3 | Regular | 15-15-15 | 9,260 | 1,680 | | | 3,320 | | |
| 53-A-3 | Regular | 15-15-15 | 4,060 | 740 | | | 3,000 | | |
| 144-A-3 | Regular | 15-15-15 | 5,770 | 1,040 | 1,200 | 25 | 2,940 | 3,110 | 4.5 |
| 249-A-3 | Regular | 15-15-15 | 7,330 | 1,340 | | | 3,170 | | |
| 174-B-3 | Regular | 13-13-15 | 7,190 | 1,310 | | | 3,000 | | |
| 202-B-3 | Regular | 13-13-15 | 5,460 | 990 | 1,010 | 20 | 2,590 | 2,830 | 5.7 |
| 280-B-3 | Regular | 13-13-15 | 3,890 | 720 | | | 2,910 | | |
| 66-C-3 | Regular | 14-14-15 | 6,290 | 1,170 | | | 2,650 | | |
| 153-C-3 | Regular | 14-14-15 | 6,380 | 1,160 | 1,110 | 7 | 3,490 | 2,990 | 11.0 |
| 179-C-3 | Regular | 14-14-15 | 5,470 | 990 | | | 2,840 | | |
| 427-A-3 | Medium | 15-15-15 | 4,910 | 770 | | | 6,570 | | |
| 428-A-3 | Medium | 15-15-15 | 6,550 | 1,030 | | | 6,520 | | |
| 439-A-3 | Medium | 15-15-15 | 7,840 | 1,230 | 1,210 | 25 | 6,890 | 6,550 | 2.8 |
| 501-A-3 | Medium | 15-15-15 | 11,680 | 1,820 | | | 6,220 | | |
| 431-B-3 | Medium | 13-13-15 | 3,040 | 470 | | | 4,860 | | |
| 568-B-3 | Medium | 13-13-15 | 10,520 | 1,610 | 1,320 | 43 | 6,220 | 6,380 | 17.0 |
| 622-B-3 | Medium | 13-13-15 | 12,280 | 1,880 | | | 8,070 | | |
| 513-C-3 | Medium | 14-14-15 | 8,280 | 1,270 | | | 4,620 | | |
| 672-C-3 | Medium | 14-14-15 | 12,790 | 1,950 | 1,580 | 16 | 4,520 | 4,830 | 7.0 |
| 674-C-3 | Medium | 14-14-15 | 9,770 | 1,510 | | | 5,360 | | |
| 1,028-A-3 | Fine | 15-15-15 | 5,780 | 910 | | | 7,390 | | |
| 1,046-A-3 | Fine | 15-15-15 | 6,340 | 1,000 | | | 6,920 | | |
| 1,057-A-3 | Fine | 15-15-15 | 6,900 | 1,100 | 1,240 | 28 | 8,000 | 7,060 | 9.0 |
| 1,066-A-3 | Fine | 15-15-15 | 12,130 | 1,930 | | | 5,920 | | |
| 846-B-3 | Fine | 13-13-15 | 13,900 | 2,180 | | | 8,680 | | |
| 882-B-3 | Fine | 13-13-15 | 10,510 | 1,650 | 1,730 | 17 | 8,610 | 8,120 | 9.0 |
| 906-B-3 | Fine | 13-13-15 | 8,470 | 1,350 | | | 7,010 | | |
| 880-C-3 | Fine | 14-14-15 | 6,800 | 1,070 | | | 7,070 | | |
| 974-C-3 | Fine | 14-14-15 | 3,300 | 520 | 680 | 38 | 6,240 | 6,600 | 11.0 |
| 1,074-C-3 | Fine | 14-14-15 | 2,790 | 440 | | | 5,860 | | |

TABLE 18.—*Summary*

*Cross-Breaking and Crushing Tests*

| Burn | Burned to Cone | Modulus of Rupture, Brick Tested Edgewise, 6-in. Span | | | | | | Crushing Strength, Brick Laid Flat | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Regular Grind | | Medium Grind | | Fine Grind | | Regular Grind | | Medium Grind | | Fine Grind | |
| | | Mean | a.d.* | Mean | a.d. | Mean | a.d. | Mean | a.d. | Mean | a.d. | Mean | a.d. |
| | 15 | 920 | 3 | 1,050 | 6 | 1,170 | 19 | 2,680 | 6 | 4,780 | 14 | 5,880 | 17 |
| 1 | 14 | 770 | 5 | 920 | 11 | 790 | 52 | 2,100 | 3 | 4,820 | 7 | 5,250 | 3 |
| | 13 | 690 | 10 | 910 | 24 | 910 | 6 | 2,730 | 8 | 4,680 | 11 | 5,840 | 6 |
| | 15–15 | 910 | 15 | 1,680 | 5 | 1,270 | 43 | 2,680 | 9 | 6,030 | 11 | 6,890 | 14 |
| 2 | 14–14 | 920 | 3 | 1,680 | 7 | 1,590 | 24 | 2,680 | 5 | 5,570 | 5 | 7,830 | 6 |
| | 13–13 | 1,010 | 10 | 1,310 | 24 | 1,070 | 13 | 2,760 | 5 | 6,270 | 11 | 6,290 | 12 |
| | 15–15–15 | 1,200 | 25 | 1,210 | 25 | 1,240 | 28 | 3,110 | 5 | 6,550 | 3 | 7,060 | 9 |
| 3 | 14–14–15 | 1,110 | 7 | 1,580 | 16 | 680 | 38 | 2,990 | 11 | 4,830 | 7 | 6,600 | 11 |
| | 13–13–15 | 1,010 | 20 | 1,320 | 43 | 1,730 | 17 | 2,830 | 6 | 6,380 | 17 | 8,120 | 9 |

* a.d. represents the average deviation in per cent. of mean.

## *Discussion of Cross-Breaking and Crushing Tests*

*Average Deviation and Index of Uniformity.*—It is highly desirable not only that the brick should be mechanically strong, but also that different specimens of the same lot should be of uniform strength. For this reason, in Table 19, obviously the lot of brick of "regular grind, mark C–3," is a more satisfactory material, so far as cross-breaking is concerned, than the "fine grind, mark A–2," notwithstanding the fact that the mean modulus of rupture of the former is 1,110 lb. per square inch, of the latter, 1,270 lb. per square inch. Since, plainly, mean values in themselves are insufficient as a basis of comparison for the strength of such brick, the "average deviation" of the single observations from the mean (as developed in the principles of precision of measurements) has been calculated as a measure of uniformity of the different specimens, for each lot of brick tested.[71]

The deviation of single observations from the mean is due to two factors: (1) lack of precision of the measurements; and (2) lack of uniformity of different specimens of the material. In the series of tests herein described, variations due to the former factor amount to less than 2 per cent. and are practically negligible as compared with variations due to the latter. For this reason, the "average deviation" can be considered a true measure of uniformity. A very low "average deviation" is an indication of a high degree of

[71] Similarly, "average deviations" are given by the Mechanical Engineering Department of the Massachusetts Institute of Technology, in reporting results obtained in testing the strength of timber.

ɔncordance in the values found for different specimens of the same ıaterial; a high "average deviation" indicates a lack of concordance.

The method of derivation of the "average deviation" for two ɔecific cases is shown in Table 19.

TABLE 19

| Grind | Mark | Modulus of Rupture | Mean Modulus | Deviation from Mean | Average Deviation or "a.d." | a.d. in Per Cent. of Mean |
|---|---|---|---|---|---|---|
| ine........... | 824–A–2 | 680 | | 590 | | |
| ine........... | 848–A–2 | 770 | | 500 | | |
| ine........... | 850–A–2 | 1,950 | 1,270 | 680 | 540 | 43.0 |
| ine........... | .970–A–2 | 1,670 | | 400 | | |
| ˌegular........ | 66–C–3 | 1,170 | | 60 | | |
| ˌegular........ | 153–C–3 | 1,160 | 1,110 | 50 | 80 | 7.2 |
| ˌegular........ | 179–C–3 | 990 | | 120 | | |

In the graphical representation of the relative uniformity of different ɔts of specimens (Fig. 14) use has been made of an "index of uniformity,"

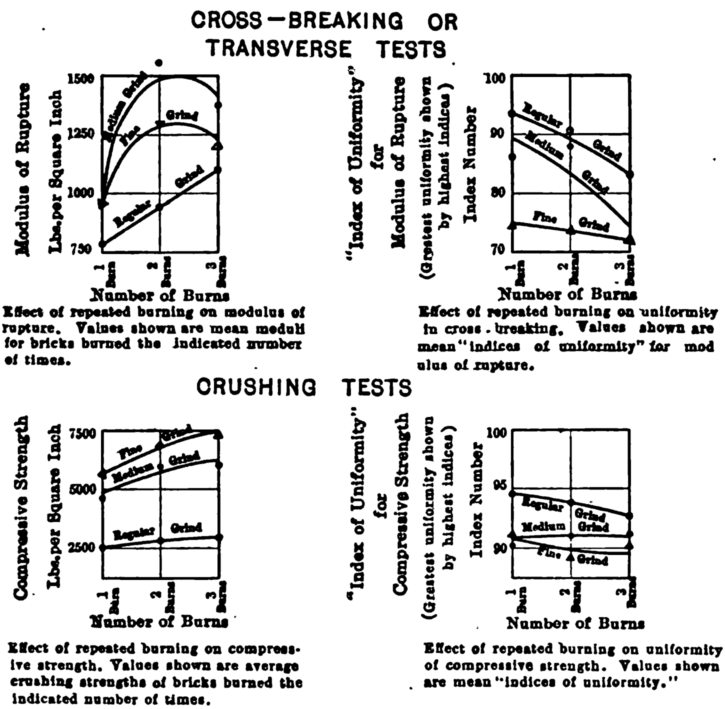
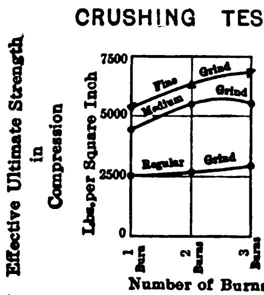



FIG. 14.

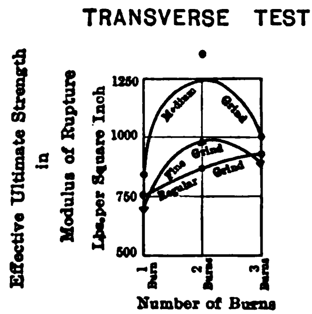
calculated from the average deviation (a.d.) thus: "Index of uniformity" = 100 − (a.d. in per cent. of mean). In the above table, the "index

of uniformity" for modulus of rupture is 57 on this scale for the first lot, 92.8 for the second.

*Effective Ultimate Strength.*—In comparing the strength of different lots of brick, their relative uniformity must be considered. As shown above, the mean ultimate strength is not a satisfactory basis for comparison. Since the "average deviation" is a measure of uniformity, it seems logical to consider the value obtained by subtracting the average deviation from the mean strength, as a more practical comparative standard. This value may be called the "effective ultimate strength;" and to this, rather than to the mean, the factor of safety should be




* "Effective ultimate strengths" were calculated by deducting average deviations from mean compressive strengths. Average values are shown.




"Effective ultimate strengths" were calculated by deducting average deviations from mean moduli of rupture. Average values are shown.

FIG. 15.

applied in calculating a safe working load for materials of which the load-carrying capacity is an important consideration. The effect of repeated burning upon the "effective ultimate strength," as so defined, is shown in Fig. 15.

*Effect of Variation in Method of Manufacture.*—The effects of the variables considered upon the strength of the test brick are represented graphically in Figs. 14 and 15.

The mean modulus of rupture of medium-grind brick is greatest; of regular-grind, least. The burning temperatures which produce the strongest brick in cross-breaking are as follows:

| | First Burn | Second Burn |
|---|---|---|
| Regular-grind.......... | Cone 15 | Cone 15 |
| Medium-grind.......... | Cone 15 | Cone 14 and 15 |
| Fine-grind............. | Cone 15 | Cone 14 |

Bricks of regular grind increase in cross-breaking strength after each burning; after the first burn, specimens burned at cone 15 are considerably stronger than those burned at cone 13 or 14, after the second burn this is no longer true. Modulus of rupture of medium- and fine-grind brick, in most cases, reaches a maximum on the second burn.

Crushing strength of regular-, medium- and fine-grind brick are to each other approximately as 41:84:100. There is an increase in strength after

each burn; a slight increase for regular grind, a considerable increase for medium and fine grinds.

Uniformity of strength of a given lot of brick diminishes with repeated burning; slightly in compression, considerably in cross-breaking. The most uniform brick are those of regular grind; fine-grind brick are fairly uniform in crushing, very non-uniform in cross-breaking strength.

The figures indicate that the medium-grind brick is the most satisfactory in point of strength, the regular-grind least so. While the compressive strength of the medium-grind is somewhat lower than that of the fine-grind, its cross-breaking strength is greater and it is a much more uniform material. In cross-breaking, the fine-grind shows little if any advantage over the regular-grind, for while its mean strength is high, it lacks uniformity; in compression, it is far stronger than the regular-grind.

There is, in all cases, an advantage in burning the brick a second time; for the medium and fine grinds, a third burn is usually harmful, decreasing the cross-breaking strength considerably, and increasing the compressive strength very slightly. Brick of regular grind increase in strength on each burn up to the third, slightly in compression, considerably in cross-breaking.

The alternate expansions and contractions incurred in repeated burning may tend to cause the formation of minute cracks in the brick and to have an unfavorable effect upon their strength. For that reason it is probable that stronger brick would be produced by maintaining the maximum temperature a greater length of time during a single burn than by resorting to repeated burning.

## Microscopic Study of Constitution of Brick [72]

### *Method in General*

Quantitative determinations of the proportions of quartz, tridymite and cristobalite in specimens of the various lots of brick were made by the Rosiwal micrometric method.[73] The measurements were all made upon powdered material, the minerals being identified through comparison of indices of refraction by immersion in liquids of definite indices. The Zeiss mechanical stage, and a lens system giving a magnification of about 390 diameters, were used.

[72] The microscopic work was performed under the direction of Dr. C. H. Warren, without whose instruction and assistance the writer would not have ventured to undertake a study of this character.

[73] A. Johannsen: *Manual of Petrographic Methods*, p. 291. McGraw-Hill, New York, 1914. Lincoln and Rietz: Determination of the Relative Volumes of the Components of Rocks by Mensuration Methods, *Economic Geology*, vol. 8, pp. 120–139 (1913).

The indices of refraction of quartz are 1.544 to 1.533; cristobalite, 1.484 to 1.487; tridymite, 1.469 to 1.473;[74] calcium silicates, higher than those of quartz.[75] Separate portions of each powder were studied in two liquids whose indices of refraction were respectively 1.495 and 1.479. The higher-refracting quartz and compounds not identified were distinguished in the former liquid from the lower-refracting cristobalite and tridymite. A large part of the higher-refracting material had the refractive index and general properties of quartz.

The proportion of tridymite was determined by immersion of the powder in a liquid with refractive index of 1.479, which is between those of tridymite and cristobalite, but extremely close to them. On account of the latter fact, variations in room temperature caused some difficulty, since the index of the liquid was diminished 0.0005 to 0.0006 for each degree temperature rise. All material with an index lower than 1.479 was reported as tridymite, and the amount of cristobalite obtained by difference.

### *Sampling*

A sample of about 300 grams from each lot of test specimens (100 grams from each brick of the lot) was crushed in a small Blake crusher and a disk grinder, to pieces approximately 1/16 in. in diameter. After cutting down to $\pm 20$ grams with a split shovel, the sample was reground until all passed through a 140-mesh sieve. For the microscopic study, that fraction was reserved (about 5 per cent. of the whole) which passed through a 240-mesh and was caught on a 260-mesh sieve. The size of these grains as determined microscopically varied from 0.05 to 0.06 mm. in diameter.

### *Identification*

The principal optical property used in identifying the minerals was the index of refraction, by means of the Becke effect. The single grains were commonly made up of two, sometimes all three minerals, and the correct positions of the boundaries between tridymite and cristobalite could be told only by crossing the nicols and rotating the stage.

The more highly refracting material (chiefly quartz in the first burns) was easily recognized by its high relief in the liquid used, and by well-defined Becke lines. Tridymite was distinguished by its low index, low double refraction, and by the lath-shaped and wedge-shaped outlines of its crystals. Identification was confirmed by the facts that the lath-shaped sections had parallel extinction and negative elongation, and that

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[74] See Table 2.

[75] Rankin and Wright: The Ternary System $CaO-Al_2O_3-SiO_2$, *American Journal of Science*, ser. 4, vol. 39, p. 74 (1915).

the index of refraction of the faster ray, as determined by the immersion method in sodium light, was 1.469.[76]

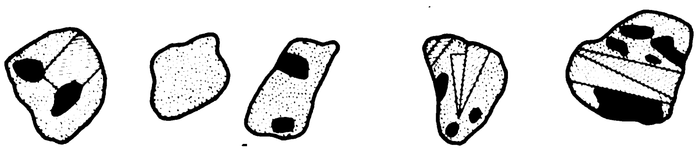
The low relief, the index of refraction, and the extremely low double refraction, which could usually be detected only with the aid of the sensitive tint plate, served to identify cristobalite. In the liquid with index 1.479, the Becke lines were faint, but their directions of motion could be

Quartz shown in black; cristobalite dotted

FIG. 16.—SKETCHES OF GRAINS OF POWDERED BRICK. FROM BRICK OF REGULAR GRIND, BURNED ONCE AT CONE 14. × 218 APPROXIMATELY.

detected upon close observation so long as the room temperature was kept at the proper point.

The lack of homogeneity of the grains led to the introduction of certain unavoidable quantitative errors. Quartz could, as a rule, be recognized even on the interior of the grains by the clearness of the Becke lines and the high relief; yet numerous small particles of residual quartz



Quartz shown in black; cristobalite dotted
tridymite cross-hatched

FIG. 17.—SKETCHES OF GRAINS OF POWDERED BRICK. FROM BRICK OF REGULAR GRIND, BURNED FOUR TIMES AT CONE 14–14–15–14. × 218 APPROXIMATELY.

were often so mingled with the cristobalite that an accurate quantitative separation of the two was impossible. The double refraction of tridymite was the only means of determining with exactness the boundaries between tridymite and cristobalite, but in sections of certain orientation could not be used. The presence of numerous microscopic cracks caused anomalous effects, and in many cases interfered with the Becke tests.

The texture of representative grains is shown in Figs. 16 and 17.

*Precision*

A coarser powder than that used was found to give less accurate results. The very fine portion, which went through the 260-mesh sieve,

[76] See C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, p. 351 (1913).

was exceedingly difficult to study. Since the latter fraction comprised about 85 per cent. of the original powder, it is evident that any great variation in its composition from that of the study sample will very seriously affect the results. For that reason, check determinations were made in a few cases of the amount of quartz in both the fraction which had passed through the 260-mesh sieve, and that which had been caught upon it. Results so obtained averaged 2 per cent. lower in the former case than in the latter, a difference which lies within the error of observation.

Errors introduced into the tridymite determinations by discarding the very fine material are probably serious, yet it was not found possible to obtain concordant results with this portion of the powder. As shown in Fig. 19, tridymite forms most rapidly in the fine-grained groundmass of the brick, and comparatively slowly in the larger grains. A much larger proportion of the groundmass than of the grains will undoubtedly crush down in the grinding so as to pass through a 260-mesh sieve. For that reason, the amount of tridymite in the fraction of the powder reserved for study, is probably considerably less than in the brick itself.

TABLE 20.—*Results of Microscopic Study—Tabulated Data*
*Constitution of Test Brick*

| Lot | Mark | Grind | Burned to Cone | Number of Burns | Constitution in Per Cent. by Volume | | |
|---|---|---|---|---|---|---|---|
| | | | | | $n > 1.495$ | $n < 1.479$ | $n > 1.479, < 1.495$ |
| | | | | | Quartz (+Silicates) | Tridymite | Cristobalite |
| A-1 | 277, 279 | Regular | 15 | 1 | 26 | 4 | 70 |
| C-1 | 102, 156 | Regular | 14 | 1 | 25 | 4 | 71 |
| B-1 | 135 | Regular | 13 | 1 | 40 | 1 | 59 |
| A-1 | 715, 730 | Medium | 15 | 1 | 24 | 6 | 70 |
| A-1 | 1,150, 1,151 | Fine | 15 | 1 | 26 | 4 | 70 |
| C-1 | 934, 1,052 | Fine | 14 | 1 | 26 | 5 | 69 |
| A-2 | 275, 288, 290, 293 | Regular | 15-15 | 2 | 19 | | |
| A-3 | 39, 53, 144, 249 | Regular | 15-15-15 | 3 | 16 | 19 | 65 |
| C-3 | 66, 153, 179 | Regular | 14-14-15 | 3 | 14 | 20 | 64 |
| B-3 | 174, 202, 280 | Regular | 13-13-15 | 3 | 18 | 10 | |
| A-3 | 427, 428, 439, 501 | Medium | 15-15-15 | 3 | 13 | 17 | 70 |
| A-3 | 1,028, 1,066, 1,057, 1,046 | Fine | 15-15-15 | 3 | 13 | 17 | 70 |
| C-4 | 12 | Regular | 14-14<br>15-14 | 4 | 12 | 30 | 58 |
| C-6* | 74 | Regular | 14-14-15<br>14-15-14 | 6 | 14 | 44 | 42 |
| B-6* | 326 | Regular | 13-13-15<br>14-15-14 | 6 | 12 | 45 | 43 |

* Analysis of No. 74 and No. 326 by V. Dolmage.

The figures given as the proportions of quartz and tridymite in the test brick represent in each case results of measurements taken on 150 to 250 grains. As a check on the work, percentages were calculated for each 75 to 80 grains. The first two results usually agreed within 3 to 4 per cent.; if not, an additional 80 to 100 grains were studied. The figures so obtained are believed to be truly comparative as showing the direction and approximately the velocity of the transformations occurring, notwithstanding the constant errors due to the difficulties above-mentioned, and to the personal equation. The latter errors are believed to be small, since results obtained on the same powder by Mr. Victor Dolmage (who kindly assisted with many of the measurements) and by the writer, usually checked within 3 per cent.

*Effect of Variations in Grind and Burning Temperature, and of Repeated Burning, upon Constitution of the Brick*

The effect of repeated burning on the constitution of brick of regular grind, burned several times, is shown in Fig. 18. The analyses of this brick after repeated burnings are shown in Table 21.

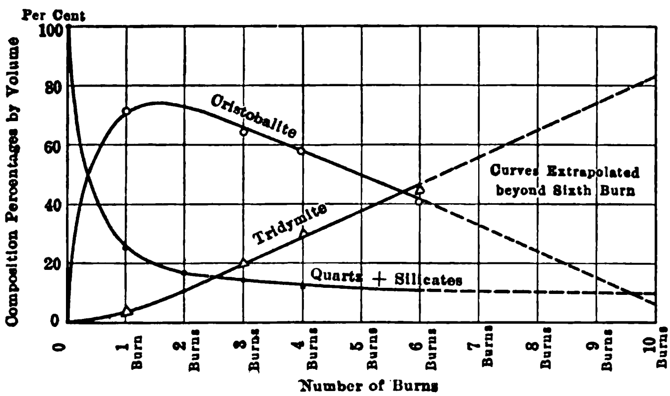



FIG. 18.—EFFECT OF REPEATED BURNING UPON THE CONSTITUTION OF SILICA BRICK. BRICK OF REGULAR GRIND. BURNING TEMPERATURE, CONE 14–14–15–14–15–14.

TABLE 21

| Mineral | Volume, Per Cent. | | | | | |
|---|---|---|---|---|---|---|
| | Number of Burns | | | | | |
| | 1 | 2 | 3 | 4 | 5 | 6 |
| Quartz (+Silicates)..... | 25 | .... | 14 | 12 | .... | 14 |
| Cristobalite............ | 71 | .... | 64 | 58 | .... | 42 |
| Tridymite.............. | 4 | .... | 20 | 30 | .... | 44 |

It is evident that in the first burn the larger part of the quartz is transformed into cristobalite, and that on repeated burning the remaining quartz is very slowly transformed.[77] The cristobalite at first very slowly and later more rapidly, inverts to tridymite. These results are apparently in accordance with the conclusions of Grum-Grzimailo and Endell, that silica brick heated a sufficient length of time at the proper temperature go slowly but completely over into tridymite.

Time and temperature of burning appear to be the controlling factors. Bricks burned a single time at cones 14 and 15 show essentially the same amount of transformation; those burned at cone 13 show considerably less. The effect of variations in grind is comparatively slight.

### *Study of Thin Sections*

Examination of thin sections cut from certain specimens of test brick show that the inversions proceed most rapidly in the groundmass, most slowly in the larger grains. This is illustrated in Fig. 19, a sketch made from thin section of brick of fine grind, burned three times at cone 15. The groundmass and the rims of the grains are made up of cristobalite and tridymite, with a little residual quartz; the interior of the grains consists of cristobalite and quartz, with a very little tridymite.

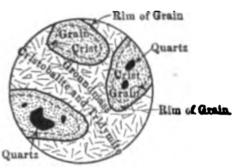



Fig. 19.—Sketch Made from Thin Section of Brick of Fine Grind, Burned Three Times at Cone 15.

Photomicrographs of the thin sections are shown in Figs. 20 to 23 inclusive. The dark areas represent principally cristobalite, the lighter areas, quartz and tridymite. From a comparison of the first two figures with the second two, the increased amount of tridymite in the later burns is at once evident.

## The Relation of the Constitution of Silica Brick to the Physical Properties

### *The Effect of the Addition of Lime*

It has already been shown that fineness of grind is not the controlling factor in the formation of tridymite, since that mineral evidently forms nearly as rapidly in the brick of coarse (or regular) grind, as in those of finer grinds. A seeming contradiction exists between this fact and the fact that the inversion to tridymite takes place more rapidly in the fine-grained groundmass of the brick than in the larger quartzite fragments (see Fig. 19).

The relative concentrations of lime in the groundmass of brick of

[77] Compare with results obtained by Seaver, shown in Table 12.

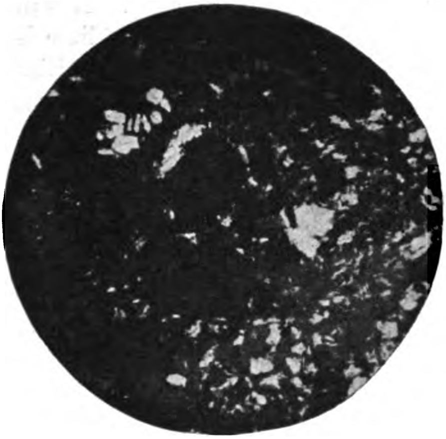


FIG. 20. FIG. 21.

FIG. 20.—SILICA BRICK OF REGULAR GRIND, BURNED ONCE AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

The white areas represent residual quartz grains of irregular size and shape, surrounded by a dark field of cristobalite. Little, if any, tridymite is to be seen.

FIG. 21.—SILICA BRICK OF FINE GRIND, BURNED ONCE AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

The figure shows residual quartz grains surrounded by cristobalite. There are a few narrow, elongated white areas in the field, which may represent tridymite.

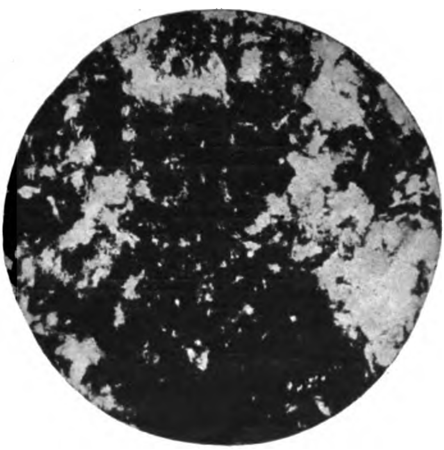


FIG. 22. FIG. 23.

FIG. 22.—SILICA BRICK OF REGULAR GRIND, BURNED THREE TIMES AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

Tridymite and residual quartz are shown by the white areas; cristobalite by the dark areas.

FIG. 23.—SILICA BRICK OF REGULAR GRIND, BURNED THREE TIMES AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

The larger white areas indicate places in which the conversion to tridymite is nearly complete. The smaller white areas in the darker portion of the field represent residual quartz and tridymite, surrounded by cristobalite (shown in black). Figs. 22 and 23 are taken from different parts of the same thin section.

various grinds probably has some bearing in this connection. It will be recalled that in the manufacture of the brick, 2 per cent. of lime is added to ground quartzite containing about 2 per cent. of impurities (mainly $Al_2O_3$ and $Fe_2O_3$). Since all of this enters the groundmass, the proportion of lime therein is much higher than 2 per cent.

The greatest concentration of lime in the groundmass will occur in that brick with the least proportion of fines (*i.e.*, the regular grind), in which, therefore, conditions will be most favorable for the formation of flux compounds during the burning. The catalytic action of the flux compounds probably explains the observed inversion of cristobalite to tridymite, which as Fenner has shown, does not form except in the presence of a flux. According to this point of view, an increase in the amount of lime in [illegible] should accelerate the formation of tridymite.

## *Specific Volumes of the Silica Minerals*

As an aid in interpreting the results of the microscopic study, use has been made of the specific volume curves of Fig. 24, based upon the following data:

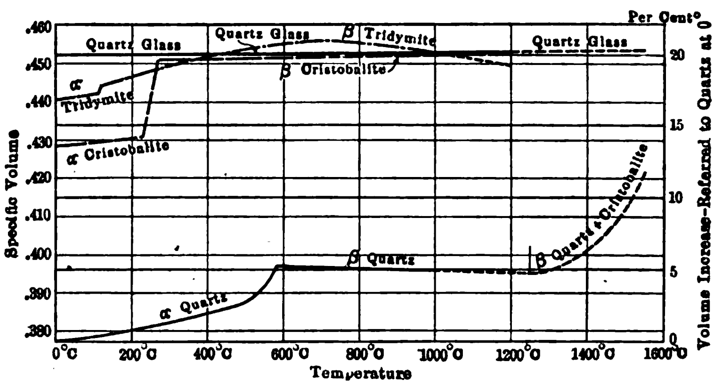



FIG. 24.—SPECIFIC VOLUMES OF THE SILICA MINERALS AND QUARTZ GLASS.

The quartz curve is a reproduction of that shown in Fig. 4, taken from the work of Day, Sosman and Hostetter.

The specific volumes at 20° of quartz glass, tridymite and cristobalite were calculated from their specific gravities, using the values shown below:[78]

| | Specific Gravity |
|---|---|
| Quartz glass................................ | 2.210 (Dana, Endell) |
| Tridymite................................ | 2.270 (Fenner) |
| Cristobalite................................ | 2.333 (Fenner) |

[78] See Table 3.

The quartz-glass curve was obtained by computing the volume expansion from the linear expansion[79] as given by LeChatelier, and applying these figures to the specific volume at 20°.

The curve for cristobalite is based upon the work of Rieke and Endell, and of LeChatelier. Both show that the expansion of cristobalite from 0° to 230° is slight, and that the volume changes at 230° to 270° is considerable,[80] due to the $\alpha$ to $\beta$ cristobalite inversion. Rieke and Endell find that the specific volume and coefficient of expansion of cristobalite above 300° are nearly the same as those of quartz glass; the specific volumes, as computed from the expansion data of LeChatelier[79] are considerably lower than those of quartz glass. The view of Rieke and Endell is taken in plotting the cristobalite curve of Fig. 24.

The tridymite curve was calculated from the thermal expansion as given by LeChatelier (Fig. 3). The specific volume of tridymite from 500° to 1,000° is seemingly a little greater than that of quartz glass. While this is probably incorrect, the difference between the specific volumes of the two materials is doubtless small.

The curves of Fig. 24 have been used hereinafter in discussing the probable effect of the changes in constitution upon certain properties of silica brick. It is believed that the inaccuracies of these curves are merely slight errors of degree; and that, while conclusions drawn from them can not be numerically exact, they will serve to indicate the general trend of the changes considered.

## *The Phenomena of Expansion*

*Permanent Expansion, Temporary or Thermal Expansion, and the Effect of Rapid Temperature Changes.*—The permanent expansion of silica brick after the first burning (in which $\pm 5$ per cent. of tridymite and $\pm$ 70 per cent. of cristobalite are formed) amounts to 10 to 11 per cent. by volume. The gradual decrease of quartz and increase of tridymite caused by repeated burning, theoretically requires a slight additional expansion after each burn until equilibrium is reached, since the complete transformation of quartz to cristobalite is accompanied by a volume increase of 13.6 per cent.; quartz to tridymite, 16.8 per cent.

At 1,250° C. the specific volumes of cristobalite and tridymite are approximately 14 to 15 per cent. greater than that of quartz. Since the residual quartz of silica brick gradually goes over into the other forms of silica at furnace temperatures, an underburned brick will expand considerably after being placed in use.

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[79] Fig. 3.
[80] Fig. 3; also Fig. 5.

Tridymite expands upon heating much less than does cristobalite. Its specific volume at 20° is 2.8 per cent. greater than that of cristobalite; at 1,000°, nearly the same as that of cristobalite. The gradual transformation of cristobalite into tridymite, shown to occur upon repeated burning of silica brick, probably takes place also in furnace linings. This inversion is accompanied by little, if any, change in volume at furnace temperature.

From the data of Fig. 24, specific volumes at different temperatures were computed for two materials assumed to consist of mixtures of quartz, tridymite, and cristobalite; expansion curves derived from these figures are shown in curves I and II, Fig. 25. Curve I is based upon an assumed composition of 25 per cent. quartz, 70 per cent. cristobalite, 5 per cent. tridymite; curve II on one of 12 per cent. quartz, 43 per cent. cristobalite,

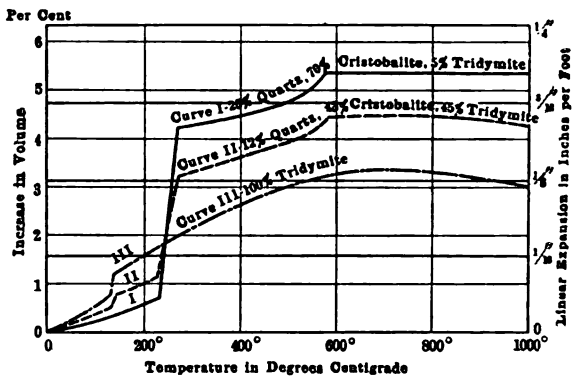


FIG. 25.—THEORETICAL EXPANSION CURVES FOR MIXTURES OF THE SILICA MINERALS.

Composition of Curve I corresponds closely to brick of regular grind *burned once* at cone 14.

Composition of Curve II corresponds closely to brick of regular grind *burned six times* at cone 14–14–15–14–15–14.

42 per cent. tridymite. These figures represent approximately the compositions of brick of regular grind, after one and after six burnings respectively. Curve III is taken from the expansion data of LeChatelier.[81]

Fig. 25 is intended merely to indicate the probable character of the expansion curve of silica brick, and of the effect upon the same of the observed changes in constitution. Obviously, the theoretical curves shown will not correspond precisely with those that would be obtained by actual measurements, being affected by inaccuracies in the data from which they were computed, and, in addition, modified by changes in

[81] Fig. 3.

porosity of the brick, the action of flux compounds, and the lag in expansion due to the temperature gradient between the exterior and interior of the brick.

These curves indicate that the temporary or thermal expansion of silica brick is decreased by repeated burning, and that an all-tridymite brick would have an expansion amounting to about 60 per cent. of that of an ordinary commercial brick, as represented in Curve I.

It is well known that silica brick crack or spall badly if subjected to rapid temperature changes. The phenomenon is probably due chiefly to the rapid expansion at 230° to 270°, accompanying the $\alpha$ to $\beta$ cristobalite inversion. This effect is diminished by repeated burning. The calculated volume increase at 230° to 270° amounts to 3.5 per cent. for brick burned once, 2.0 per cent. for brick burned six times. An all-tridymite brick should show very little tendency to crack upon rapid heating or cooling, since the only sharp break in the tridymite expansion curve, occurring at 130°, is attended by a volume increase of but 0.3 per cent.

For several reasons, therefore, a large proportion of tridymite in silica brick is desirable, provided the brick are not to be heated continuously above 1,470° C.

1. Tridymite has the least thermal or temporary expansion of the three silica minerals.

2. Tridymite has the greatest specific volume of the three minerals, and can not show any permanent expansion after repeated burning below its melting point.

3. A tridymite brick could probably be subjected to fairly rapid changes of temperature with little danger of cracking.

Comparison of curves I and II seems to show that, while repeated burning produces very desirable effects, the advantages derived up to the sixth burn are entirely disproportional to the additional expense involved, on account of the extreme slowness with which tridymite is formed. Moreover, if used continuously at a temperature exceeding 1,470°, a tridymite brick would slowly revert to cristobalite.

Since the time element appears to be the most important factor in the formation of tridymite, it appears that long-continued burning at the proper temperatures should be more satisfactory than repeated burning. The maximum temperature is maintained 40 hr.[82] in the burning of silica brick; maintaining this temperature for 2½ weeks should presumably give rise to the same changes in constitution as 10 separate burnings under the ordinary conditions of manufacture, and produce brick consisting almost wholly of tridymite (see Fig. 16).

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[82] K. Seaver: *Op. cit.*, p. 136.

### *Burning Temperature*

In order to make a brick containing the maximum proportion of tridymite, the burning temperature should be that at which the inversion to that mineral proceeds most rapidly. It appears that this must occur below 1,470°, since tridymite is not known to form under any conditions above that point. While the inversion proceeds more rapidly at cones 14 and 15 than at cone 13, no conclusions can be drawn from the data herein given as to the most favorable temperature for burning.

## Suggestions for Future Study

It seems desirable that the theoretical advantages of a high-tridymite brick should be investigated by comparing the thermal expansion of an ordinary commercial brick with that of a brick high in tridymite. The effect of rapid temperature changes might be compared by means of spalling tests.

If it should be proven that the production of a tridymite brick would be advantageous, the possibility of making such a brick on a commercial basis for certain purposes might well be considered. A small furnace, in which close temperature regulation is possible, should preferably be employed; and the results thereby obtained, if promising, should later be tried out on a larger scale.

The temperature at which the formation of tridymite proceeds most rapidly in silica brick, and the inversion-velocity at that temperature, are as yet undetermined. The possibility of accelerating the inversion by a slight increase in the amount of lime used or by the addition of some other catalyzing material, is worth considering.

It is not impossible that by long-continued heating at the proper temperature, with possibly a slight additional amount of lime, a tridymite brick might be produced on a commercial basis. The results of tests so far made, however, indicate that this is extremely doubtful.

## Summary

A brief review is given of the literature pertaining to the silica minerals and the silica refractories.

Microscopic study of the quartzites most important in the American refractories industry shows that they consist essentially of interlocking quartz crystals of random orientation, without a groundmass or cement. This is not in accordance with Wernicke's conclusion that no quartzites are suitable for the manufacture of high-grade refractories except those

which consist of quartz grains in a cement of amorphous silica or cryptocrystalline quartz.

Physical tests and microscopic studies were made of silica brick manufactured under varying conditions of grind, and burned one to four times at slightly varying temperatures. The grinds considered were (1) "regular," or coarse, (2) "shape," or medium, and (3) fine; the burning temperatures ranged from cone 13 to cone 16.

In all cases, a second burning increases the strength of the brick; a third burning has a harmful effect upon the strength in some cases; in others, causes little change. Brick of shape grind are the strongest, those of regular grind the weakest. With a single burning, a temperature of cone 15 produces a stronger material than cone 13 or 14. Uniformity of strength of different specimens of a given lot diminishes upon repeated burning. The most uniform brick are those of regular grind; the least uniform, those of fine grind. Attention is called to the insufficiency of average values as a proper basis for the comparison of strength of non-uniform materials, and for practical purposes the use of certain derived values, termed the "index of uniformity" and the "effective ultimate strength" is suggested.

Upon repeated burning, the quartz, of which the brick is made, is transformed through cristobalite into tridymite. By microscopic methods, estimates were made of the degree of transformation in various bricks. The figures obtained, while not numerically exact, indicate the general trend of the changes considered. The powder studied from brick burned a single time under the ordinary conditions of manufacture consists by volume of approximately 25 per cent. quartz + silicates, 70 per cent. cristobalite and 5 per cent. tridymite; after a sixth burn the composition is approximately 12 per cent. quartz + silicates, 43 per cent. cristobalite, and 45 per cent. tridymite. The inversions proceed more rapidly in the groundmass of the brick than in the larger grains.

It is considered desirable that silica brick should contain a large proportion of tridymite because (1) tridymite has the least thermal expansion of the silica minerals; (2) it will show no permanent expansion after repeated burning; (3) a tridymite brick could probably be subjected to fairly rapid changes of temperature with little danger of injury.

The advantages in these respects theoretically derived up to the sixth burn are believed to be wholly disproportional to the additional expense involved.

## SUPPLEMENTARY DATA

### Spalling Tests

*General Discussion.*—It has been shown in the preceding pages that a relation probably exists between the amount of cristobalite in a silica

brick, and its tendency to spall upon rapid change of temperature; and that since the percentage of cristobalite decreases on repeated burning, the spalling tendency should likewise diminish. It was desired to confirm this conclusion through experimental data.

In spalling tests as ordinarily conducted, brick are alternately heated and cooled rapidly a definite number of times between fixed temperature limits; they are usually cooled in the air, but sometimes by plunging into water. The loss of weight in per cent. is taken as a measure of the spalling. This method is not altogether satisfactory, since it does not indicate the internal condition of the brick, which may possibly weaken considerably in the test without much spalling, or lose surface spalls without much weakening; and if the hot test pieces are water-cooled, the formation of steam in the pores may disrupt the brick and lead to inaccurate conclusions.

For the purposes of this study it was decided to make use of a milder heat treatment than that usually employed; one sufficiently severe to cause internal cracks, yet not severe enough to cause spalls to chip off. For each lot of brick studied, the cross-breaking strength was determined upon a certain number of specimens which had been thus treated, and also upon an equal number which had not. The loss in mean strength, expressed in per cent., is considered to represent the "comparative spalling tendency."

*Spalling Test in Detail.*—The test was carried out as follows: The brick to be subjected to the heat treatment were placed in a gas-fired test kiln, 16 at a time, and heated at a uniform rate of 15° C. per hour (so as to cause no injury to the specimens during the heating) until 600° C. was attained. They were kept at that temperature for 3 hr. and at the end of that time were withdrawn and placed on edge on inverted steel pallets about 2 ft. apart, in a place protected from drafts, and allowed to cool in the air of the laboratory. They were not moved until the following day.

This treatment had the desired effect. No pieces spalled off, nor were the brick visibly shattered; yet when any two brick were struck together they gave a dead ring.

In Table 22 are shown the cross-breaking strengths before and after this heat treatment, of brick of regular grind burned a number of times varying from one to six. The burning temperatures are those shown in Table 14.

In Table 23, these values are summarized and the loss in strength, assumed to represent the comparative spalling tendency, is shown. Figs. 26 and 27 are graphic representations of the data given in Table 23.

**Table 22.**—*Cross-Breaking Tests Made to Determine Spalling Tendency*

Brick Tested on Edge; Supports 6 In. Apart; Load Applied at Midspan

| Cross-Breaking Strength of Brick *Not* Subjected to Heat Treatment | | | | | | Cross-Breaking Strength of Brick Subjected to Heat Treatment | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| No. of Burns | Mark | Lot | Cross-breaking, Load in Pounds | Modulus of Rupture: Modulus, Pounds per Square Inch | Modulus of Rupture: Mean | No. of Burns | Mark | Lot | Cross-breaking, Load in Pounds | Modulus of Rupture: Modulus, Pounds per Square Inch | Modulus of Rupture: Mean |
| 1 | 83 | A | 4,970 | 910 | | 1 | 268 | A | 1,730 | 310 | |
| 1 | 279 | A | 4,820 | 890 | | 1 | 295 | A | 2,480 | 443 | |
| 1 | 289 | A | 5,260 | 960 | | 1 | 299 | A | 2,290 | 412 | |
| 1 | 69 | C | 4,010 | 730 | 848 | 1 | 56 | C | 2,080 | 373 | 339 |
| 1 | 71 | C | 4,790 | 877 | | 1 | 73 | C | 1,900 | 341 | |
| 1 | 102 | C | 4,140 | 750 | | 1 | 97 | C | 780 | 138 | |
| 1 | 156 | C | 4,520 | 820 | | 1 | 166 | C | 2,000 | 357 | |
| 2 | 275 | A | 5,820 | 1,080 | | 2 | 18 | A | 3,430 | 605 | |
| 2 | 288 | A | 5,160 | 980 | | 2 | 88 | A | 2,360 | 421 | |
| 2 | 290 | A | 3,480 | 620 | | 2 | 125 | A | 3,120 | 564 | |
| 2 | 293 | A | 5,140 | 940 | | 2 | 240 | A | 2,030 | 365 | |
| 2 | 138 | B | 5,620 | 1,040 | | 2 | 196 | B | 2,440 | 434 | |
| 2 | 218 | B | 4,730 | 860 | 919 | 2 | 22 | B | 1,600 | 288 | 444 |
| 2 | 330 | B | 6,070 | 1,120 | | 2 | 324 | B | 2,350 | 420 | |
| 2 | 335 | B | 3,260 | 606 | | 2 | 351 | B | 2,360 | 425 | |
| 2 | 10 | C | 5,420 | 980 | | 2 | 96 | C | 2,310 | 414 | |
| 2 | 14 | C | 4,730 | 880 | | 2 | 108 | C | 2,300 | 419 | |
| 2 | 33 | C | 5,650 | 997 | | 2 | 151 | C | 2,730 | 489 | |
| 2 | 110 | C | 4,950 | 920 | | 2 | 318 | C | 2,740 | 490 | |
| 3 | 39 | A | 9,260 | 1,680 | | 3 | 70 | A | 2,850 | 505 | |
| 3 | 53 | A | 4,060 | 740 | | 3 | 123 | A | 2,900 | 518 | |
| 3 | 144 | A | 5,770 | 1,040 | | 3 | 263 | A | 2,390 | 425 | |
| 3 | 249 | A | 7,330 | 1,340 | | 3 | 303 | A | 2,620 | 468 | |
| 3 | 82 | B | 6,250 | 1,100 | | 3 | 51 | B | 2,770 | 495 | |
| 3 | 174 | B | 7,190 | 1,310 | 1,120 | 3 | 208 | B | 3,230 | 572 | 489 |
| 3 | 202 | B | 5,460 | 990 | | 3 | 267 | B | 2,800 | 494 | |
| 3 | 280 | B | 3,890 | 720 | | 3 | 276 | B | 3,020 | 531 | |
| 3 | 66 | C | 6,290 | 1,170 | | 3 | 32 | C | 3,040 | 538 | |
| 3 | 78 | C | 6,480 | 1,170 | | 3 | 152 | C | 1,960 | 355 | |
| 3 | 153 | C | 6,380 | 1,160 | | 3 | 155 | C | 2,730 | 491 | |
| 3 | 179 | C | 5,470 | 990 | | 3 | 171 | C | 2,650 | 478 | |
| 4 | 41 | A | 5,160 | 947 | | 4 | 149 | A | 3,000 | 547 | |
| 4 | 85 | A | 6,070 | 1,100 | | 4 | 247 | A | 1,830 | 325 | |
| 4 | 91 | A | 3,900 | 691 | | 4 | 305 | A | 2,800 | 500 | |
| 4 | 238 | A | 5,600 | 1,010 | | 4 | 310 | A | 1,720 | 306 | |
| 4 | 188 | B | 5,170 | 923 | | 4 | 89 | B | 3,040 | 549 | |
| 4 | 219 | B | 4,750 | 848 | 883 | 4 | 140 | B | 2,150 | 384 | 458 |
| 4 | 291 | B | 4,220 | 764 | | 4 | 195 | B | 2,280 | 406 | |
| 4 | 329 | B | 5,480 | 990 | | 4 | 350 | B | 2,050 | 366 | |
| 4 | 8 | C | 6,790 | 1,200 | | 4 | 22 | C | 2,950 | 534 | |
| 4 | 170 | C | 4,420 | 789 | | 4 | 77 | C | 2,580 | 471 | |
| 4 | 175 | C | 3,300 | 594 | | 4 | 130 | C | 2,850 | 521 | |
| 4 | 313 | C | 4,150 | 737 | | 4 | 148 | C | 3,230 | 586 | |
| 5 | 67 | A | 4,290 | 769 | | 5 | 19 | A | 2,450 | 434 | |
| 5 | 104 | A | 4,650 | 830 | | 5 | 76 | A | 2,270 | 407 | |
| 5 | 147 | A | 7,360 | 1,320 | | 5 | 132 | A | 3,020 | 543 | |
| 5 | 158 | A | 5,730 | 1,040 | | 5 | 173 | A | 2,900 | 516 | |
| 5 | 186 | B | 6,270 | 1,150 | | 5 | 80 | B | 3,010 | 528 | |
| 5 | 215 | B | 4,920 | 886 | 1,050 | 5 | 216 | B | 3,350 | 601 | 558 |
| 5 | 224 | B | 6,000 | 1,060 | | 5 | 308 | B | 3,800 | 691 | |
| 5 | 230 | B | 5,350 | 961 | | 5 | 342 | B | 2,600 | 466 | |
| 5 | 257 | C | 5,950 | 1,090 | | 5 | 122 | C | 2,820 | 510 | |
| 5 | 262 | C | 6,920 | 1,240 | | 5 | 145 | C | 3,840 | 688 | |
| 5 | 306 | C | 5,950 | 1,080 | | 5 | 165 | C | 3,410 | 611 | |
| 5 | 338 | C | 6,650 | 1,200 | | 5 | 297 | C | 3,860 | 605 | |
| 6 | 4 | A | 5,620 | 1,010 | | 6 | 87 | A | 3,570 | 638 | |
| 6 | 201 | A | 5,230 | 935 | | 6 | 126 | A | 3,170 | 573 | |
| 6 | 266 | A | 4,780 | 850 | | 6 | 157 | A | 3,250 | 572 | |
| 6 | 325 | A | 5,600 | 1,010 | | 6 | 159 | A | 2,810 | 504 | |
| 6 | 136 | B | 5,580 | 1,010 | | 6 | 30 | B | 2,610 | 468 | |
| 6 | 211 | B | 5,750 | 1,030 | 951 | 6 | 185 | B | 2,640 | 471 | 522 |
| 6 | 250 | B | 5,000 | 908 | | 6 | 232 | B | 1,940 | 349 | |
| 6 | 255 | B | 3,680 | 668 | | 6 | 320 | B | 2,750 | 479 | |
| 6 | 6 | C | 5,150 | 916 | | 6 | 91 | C | 3,300 | 600 | |
| 6 | 13 | C | 3,980 | 704 | | 6 | 94 | C | 2,990 | 549 | |
| 6 | 34 | C | 6,590 | 1,190 | | 6 | 160 | C | 2,870 | 517 | |
| 6 | 150 | C | 6,560 | 1,180 | | 6 | 317 | C | 3,010 | 549 | |

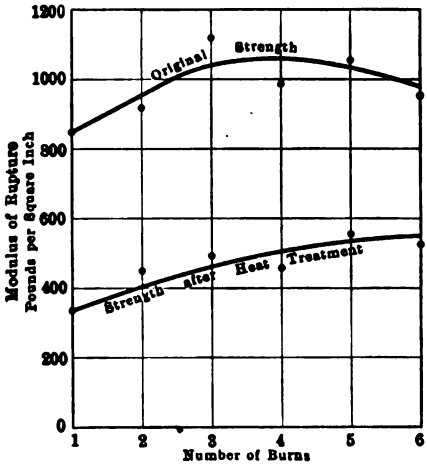


FIG. 26.—VARIATIONS IN THE CROSS-BREAKING STRENGTH OF BRICK OF REGULAR GRIND UPON REPEATED BURNING, AND THE EFFECT UPON THE STRENGTH OF A CERTAIN HEAT TREATMENT, CONSISTING OF SLOW HEATING TO 600° C. FOLLOWED BY COOLING IN THE AIR.

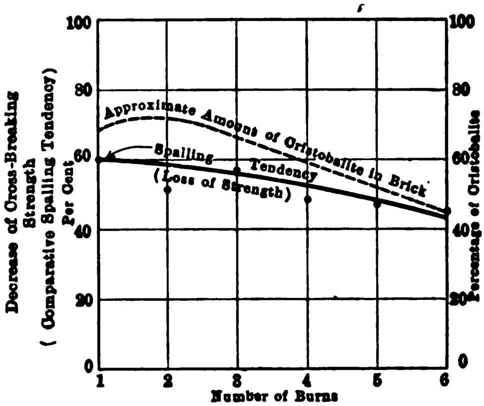


FIG. 27.—DECREASE IN CROSS-BREAKING STRENGTH EXPRESSED IN PER CENT. DUE TO HEAT TREATMENT DESCRIBED. THIS PERCENTAGE DECREASE IS ASSUMED TO REPRESENT THE COMPARATIVE SPALLING TENDENCY.

TABLE 23.—*Summary of Cross-breaking Tests Showing Loss of Strength due to Heat Treatment*

| Burn | Modulus of Rupture, Pounds per Square Inch | | Loss of Strength due to Heat Treatment | |
|---|---|---|---|---|
| | Before Heat Treatment | After Heat Treatment | Pounds per Square Inch | Per Cent. (= Comparative Spalling Tendency) |
| 1 | 848 | 339 | 509 | 60.0 |
| 2 | 919 | 444 | 475 | 51.8 |
| 3 | 1,120 | 489 | 631 | 56.2 |
| 4 | 883 | 458 | 425 | 48.1 |
| 5 | 1,050 | 558 | 492 | 46.9 |
| 6 | 951 | 552 | 429 | 45.1 |

*Results.*—The experimental data verify the conclusion that the spalling tendency of silica brick diminishes on repeated burning. In Fig. 27 there is a marked correspondence between the curve representing approximately the cristobalite content of the brick and that representing the spalling tendency. The latter, however, does not decrease as rapidly as the former. It is evident that, while reburning the brick diminishes

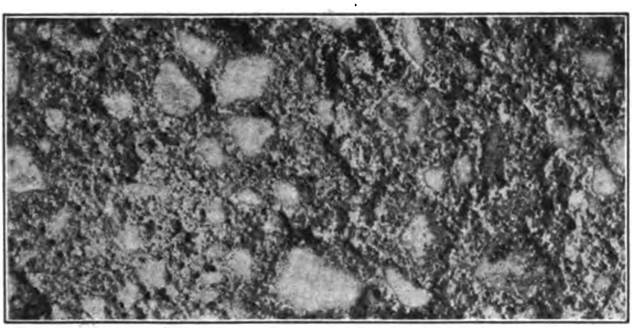

FIG. 28.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF REGULAR GRIND, BURNED ONCE IN SILICA BRICK KILN. × 4.

the spalling tendency, the change therein from 60 per cent. on a first burn to 45 per cent. after a sixth is too slight to be of any commercial importance.

A number of first-quality fire-clay brick, subjected to the same heat treatment, lost about 4.5 per cent. in strength. This result agrees with the well-known fact that the spalling tendency of clay brick, at the temperature of this test, is but a fraction of that of commercial silica brick.

FIG. 29.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF REGULAR GRIND, BURNED SIX TIMES. × 4.

FIG. 30.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF SHAPE GRIND, BURNED ONCE. × 4.

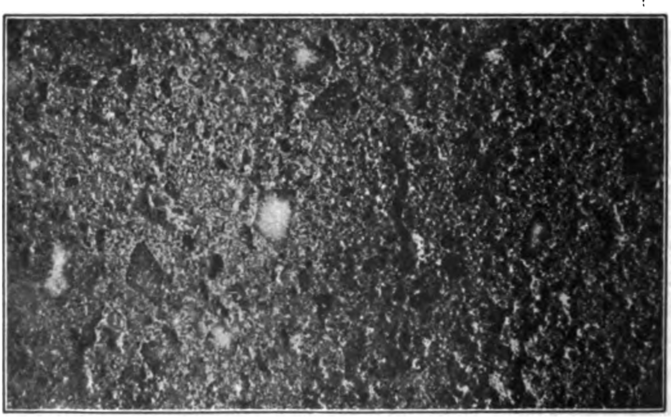

FIG. 31.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF SHAPE GRIND, BURNED SIX TIMES. × 4.

### *Changes in Texture of Brick Due to Burning*

The photomicrographs shown in Figs. 28 to 31 bring out very clearly the change in the texture of the brick brought about by repeated burning. After a single burning the larger grains of quartzite appear nearly white; after six burnings the grains stand out much more distinctly in the pictures and have become darker and apparently more dense. A number of grains are seen in which the center is still white, while the margins are altered in the manner described.

## Bibliography

### *Part I.—The Silica Minerals*


Allen, White and Wright: Wollastonite and Pseudo-Wollastonite, Polymorphic Forms of Calcium Metasilicate. *American Journal of Science*, ser. 4, vol. 21, p. 89 (1906).

Day and Shepherd: Lime-Silica Series of Minerals. *American Journal of Science*, ser. 4, vol. 22, p. 265 (1906).

Shepherd, Rankin and Wright: Binary Systems of Alumina with Silica, Lime and Magnesia. *American Journal of Science*, ser. 4, vol. 28, p. 293 (1909).

W. P. White: Melting Point Methods at High Temperatures. *American Journal of Science*, ser. 4, vol. 28, pp. 474–489 (1909).

Clarence N. Fenner: The Stability Relations of the Silica Minerals. *American Journal of Science*, ser. 4, vol. 36, p. 331 (1913).

Bowen and Anderson: Binary System $MgO-SiO_2$. *American Journal of Science*, ser. 4, vol. 37, p. 487 (1914).

Day, Sosman and Hostetter: Determination of Mineral and Rock Densities at High Temperatures. *American Journal of Science*, ser. 4, vol. 37, p. 1 (1914).

N. L. Bowen: The Ternary System. Diopside-Forsterite-Silica. *American Journal of Science*, ser. 4, vol. 38, pp. 207–264 (1914).

Rankin and Wright: The Ternary System $CaO-Al_2O_3-SiO_2$. *American Journal of Science*, ser. 4, vol. 39, p. 1 (1915).

A. E. Marshall: Fused Silica Ware, Its Manufacture, Properties and Uses in the Chemical Industries. *Metallurgical and Chemical Engineering*, vol. 10, p. 248 (1912).

A. Lampen: Melting Points of Silica or Quartz and Other Refractory Materials. *Journal of the American Chemical Society*, vol. 28, pt. 2, pp. 846–853 (1906).

Day and Shepherd: Quartz Glass. *Science*, new ser., vol. 23, No. 591, p. 670 (1906).

G. W. C. Kaye: Expansion and Thermal Hysteresis of Fused Silica. *Philosophical Magazine*, ser. 6, vol. 20, p. 718 (1910).

P. A. Boeck: An Apparatus for the Determination of the Expansion Coefficient of Solid Bodies. *Transactions of the American Ceramic Society*, vol. 14, p. 470 (1912).

A. Blackie: On the Behavior of Fused Silica at High Temperatures. *Transactions of the Faraday Society*, vol. 7, pp. 158–165 (1911).

Rammelsberg: Solubility of Quartz in Alkaline Solutions. *Annalen der Physik und Chemie, Poggendorf*, vol. 112, p. 177 (1861).

Rieke and Endell: The Transformations of Silica at Higher Temperatures. *Zeitschrift für Anorganische Chemie*, vol. 79, pp. 239–259 (1913).

R. Schwarz: Chemical Relations of the Various Modifications of $SiO_2$. *Zeitschrift für Anorganische Chemie*, vol. 76, p. 422 (1912).

LANGE AND MILBERG: The Behavior of the Different Forms of Silicic Acid toward Caustic and Carbonated Alkaline Solutions. *Zeitschrift für Angewandte Chemie*, 1897, pp. 393 and 425.

RIEKE AND ENDELL: Devitrification of Quartz Glass. *Silikat Zeitschrift*, vol. 1, No. 1, p. 6 (Coburg, 1913).

Volume Change of Some Ceramic Raw Materials on Burning. *Silikat Zeitschrift*, vol. 1, No. 4, pp. 67, 85 (Coburg, 1913).

C. DOELTER: *Handbuch der Mineral Chemie*, vol. II, pp. 115–272. Dresden, Theodor Steinkopff, 1914.

H. L. LECHATELIER: *La Silice et les silicates*. Paris, A. Hermann et Fils, 1914. La Silice *Revue universelle des Mines*, ser. 5, vol. 1, p. 85 (1913).

## *Part II.—The Silica Refractories*

CRAMER: Properties of Quartzite. *Tonindustrie Zeitung*, vol. 25, p. 864 (1901).

WERNICKE AND WILDSCHREY: Investigations Concerning Quartzite. *Tonindustrie Zeitung*, vol. 34, No. 24, p. 262 (1910).

Quartzite and Its Application in the Refractories Industry. *Tonindustrie Zeitung*, vol. 34, pp. 688, 723, 768 (1910).

HOLMQUIST: Causes of the Expansion of Dinas Brick. *Tonindustrie Zeitung*, vol. 35, p. 1324 (1911).

GRUM-GRZIMAILO: The Refractory Qualities of Dinas Brick. *Stahl und Eisen*, vol. 31, p. 224 (1911).

K. ENDELL: The Constitution of Silica Brick. *Stahl und Eisen*, vol. 32, p. 392 (1912).

O. LANGE: Concerning Silica Brick for Open Hearth Furnaces. *Stahl und Eisen*, vol. 32, pp. 1729–1737 (1912).

K. ENDELL: Silica Quartzite. *Stahl und Eisen*, vol. 33, pp. 1770, 1855 (1913).

WERNICKE: Quartzite and Silica Brick. *Stahl und Eisen*, vol. 33, p. 235 (1913.)

E. HEYN: Thermal Conductivity of Refractory Building Materials. *Stahl und Eisen*, vol. 34, pp. 832–834 (1914); *Mitteilungen aus dem Königlichen Materialprüfungsamt*, vol. 32, pp. 89–198 (1914.)

STEGER: Specific Heats of Refractory Products. *Silikat Zeitschrift*, vol. 2, No. 11, p. 203 (Coburg, 1914).

GOERENS AND GILLES: The Thermal Conductivity of Refractory Materials. *Ferrum*, vol. 12, pp. 1–11; 17–21 (1914).

GREAVES-WALKER: Notes on the Manufacture of Silica Brick by the "Anaconda Method." *Transactions of the American Ceramic Society*, vol. 13, p. 152 (1911).

G. H. BROWN: Note on Load Tests Made on Magnesite, Chrome and Silica Brick. *Transactions of the American Ceramic Society*, vol. 14, p. 391 (1912).

G. H. BROWN: Relative Thermal Conductivities of Silica and Clay Refractories. *Transactions of the American Ceramic Society*, vol. 16, p. 382 (1914).

KENNETH SEAVER: Manufacture and Tests of Silica Brick for the By-product Coke Oven. *Transactions of the American Institute of Mining Engineers*, vol. 53, p. 125 (1916).

DOUGILL, HODSMAN AND COBB: Thermal Conductivity of Refractory Materials. *Journal of the Society of Chemical Industry*, vol. 34, pp. 465–471 (1915).

BOYD DUDLEY, JR.: The Thermal Conductivity of Refractories. *Transactions of the American Electrochemical Society*, vol. 27, p. 285 (1915).

WOLOGDINE AND QUENEAU: Conductivity, Porosity and Gas Permeabilityof Refractory Materials. *Electrochemical and Metallurgical Industry*, vol. 7, pp. 383, 433 (1909).

S. M. MARSHALL: Relative Thermal Conductivity of Silica and Clay Brick. *Metallurgical and Chemical Engineering*, vol. 12, p. 74 (1914).

C. W. KANOLT: Melting Point of Fire Bricks. *Technologic Paper* No. 10, U. S. Bureau of Standards, pp. 1–17 (1912).

ANONYMOUS: Silica Brick Plant of the Robinson Clay Products Co. at Akron, Ohio. *Iron Trade Review*, vol. 41, pp. 339–343 (Aug., 1907). New Silica Brick Plant of the Harbison-Walker Refractories Co. at East Chicago, Ind. *Iron Trade Review*, vol. 43, pp. 889–892 (Nov., 1908).

H. W. CROFT: Refractories in the Iron and Steel Industry. *Iron Trade Review*, vol. 53, pp. 1138–1140 (Dec., 1913).

F. T. HAVARD: Chapter on "Preparation of the Siliceous Refractories.," *Refractories and Furnaces*, McGraw-Hill, New York, 1912.

H. O. HOFMAN: Chapter on "Refractory Materials," *General Metallurgy*, McGraw-Hill, New York, 1913.

PAUL WERNER: *Die feuerfeste Industrie*, Hartleben, Leipzig, 1911.

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# Matte Granulation at Herculaneum, Mo.

BY S. PAUL LINDAU, B. S.,* HERCULANEUM, MO., AND HENRY B. SMITH,† RIVERMINES, MO.

(New York Meeting, February, 1917)

THREE years ago it was decided by the management to granulate the matte that is produced in the smelter of the St. Joseph Lead Co. at Herculaneum, Mo., thereby doing away with a large amount of labor in handling the matte and in subsequent crushing. On Mar. 1, 1915, the granulator was put in operation and up to Jan. 1, 1916, 18,735 tons had been granulated. The four blast furnaces have a daily output of about 280 tons of pig lead and 100 tons of matte assaying about 11.0 per cent. Pb. It is expected that projected improvements in roasting methods will eliminate more sulphur from the charge, and cut down the matte-fall to about one-half of the present figure.

The blast furnaces are tapped into movable forehearths or settlers, 6 ft. by 4 ft. by 22 in., the slag overflowing into 27-cu. ft. slag cars which are hauled to the dump by electric locomotives. Previous to the adoption of granulation, the matte was tapped from the forehearths into small slag pots which were dumped after cooling. The matte cakes were broken with sledges, and transported to the crushing plant for fine crushing and screening. Here the matte was reduced to ¼-in. size before being fed to the roaster bins. A Wedge and a Holthoff mechanical furnace are used for preroasting before sintering on the Dwight and Lloyd machines. The matte is returned to the furnaces as an iron flux until the gradually accumulating copper content reaches about 7 per cent., after which it is put aside to be concentrated in a small copper blast furnace, the resultant high-grade matte being sold.

When the matte does not settle readily from the slag, two settling devices are often used, the second one being a large catch-pot, which gathers any matte that may overflow from the forehearth, the slag then overflowing from the catch-pot into the slag cars. When full of matte, the catch-pot is removed, allowed to cool, and the contents broken and crushed.

With the present plan of granulating, the matte is tapped into 6-cu. ft. iron ladles. These are transported to the granulating plant by a 5-ton

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† Chemist, Doe Run Lead Co.

overhead electric traveling crane, the runway of which extends the full length of the blast-furnace building. A cylindrical container receives the matte from the ladles. This container is a specially designed Traylor

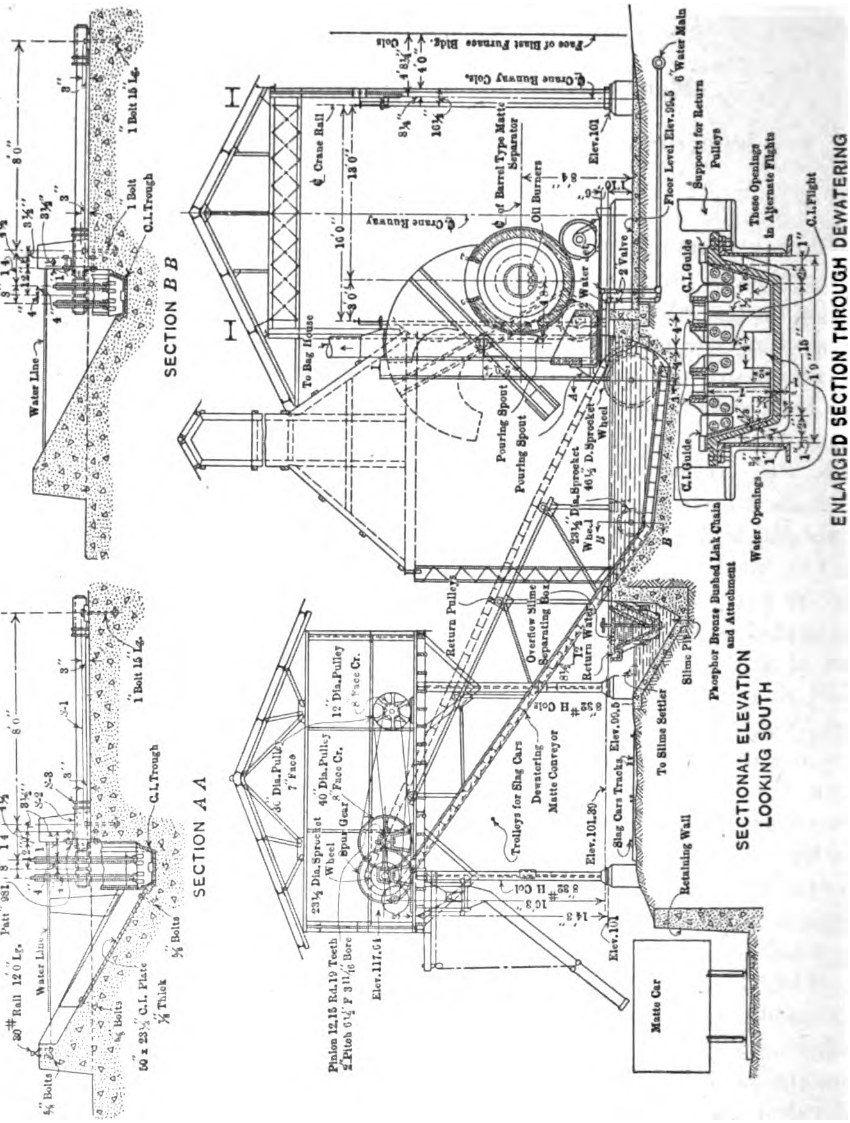


FIG. 1.—MATTE GRANULATING PLANT OF THE ST. JOSEPH LEAD CO., HERCULANEUM, MO.

copper converter minus the tuyères and with a 20-in. opening at one end for the flue and oil burners. It measures 77 by 120 in. inside, is lined with magnesite brick 9 in. thick and can be tilted by means of a 50-hp. motor. A steel flue 18 in. in diameter, leading from one end, conducts

any fume to the bag house. Just under this flue are placed two oil burners for heating. Generally one is found sufficient to keep the matte in a molten state. The consumption of 18° to 20° Bé. crude oil is 250 gal. per 24 hr., or 2 to 2½ gal. of oil per ton of matte granulated. Air at 30 lb. pressure is supplied to the burners for atomizing the oil.

The granulation is accomplished by pouring the molten matte through two superimposed flat jets of water shooting horizontally into a concrete tank lined with cast-iron plates (Fig. 1). The stream of matte is disintegrated into small shot-like particles before reaching the body of water. The stream of molten matte is accurately directed on to the jets of water by a so-called pouring box, the spout of which is 6 in. above and 12 in. in front of the upper nozzle. This box is lined with common brick and the matte is poured directly into it by revolving the container. The pour holes are slots 2 in. wide by 8 in. high.

A dewatering drag conveyor removes the granulated particles of matte from the tank. The floor of the tank slopes down to the trough in which the conveyor operates, at an angle of about 30°. It is placed to one side, out of the line of the streams from the nozzles. It elevates the matte over the slag track and discharges into standard-gage railroad cars, which are weighed, and their contents sampled and emptied into the roaster bins.

In the end of the concrete tank opposite the nozzles, a V-sh ped settling box takes the overflow water. This box prevents the loss of the coarser slime, while the finer material settles out in a series of settling tanks, one overflowing into the other. An excelsior filter finally clears the water before it enters the circulating pond. There are two series of settling tanks, one in use while the other is being cleaned out.

### *Operation and Construction*

The matte in the cylindrical container is kept fluid by one or both of the two oil burners before mentioned. The burners are placed so that the flame shoots slightly upward in the container. An oxidizing flame is used, though there would be some practical advantages if a reducing flame could be employed. It was found impossible, however, to maintain the required temperature with a reducing flame. If the matte remains in the container too long, an oxidized scum is formed which interferes with the pouring.

The matte is discharged from the barrel container through an opening in the side, 2 by 8 in., provided with a spout delivering into the pouring box which rests on the concrete wall of the tank directly over the water nozzles. This pouring box is made of sheet iron and is lined with common firebrick; inside dimensions are 18 in. by 18 in. by 3 ft. The matte tends to chill and build up in the box, but a narrow passage about 3 in.

wide is easily maintained along the path of the stream. This box in turn discharges through an opening 2 by 8 in., the matte being directed by a cast-iron spout so that it meets the horizontal jets of water at an angle of 70° to 80°. All the matte should be broken up and be prechilled before striking the main body of water. As a rule the stream of matte is broken up by striking the upper jet while the lower jet insures further cooling. It was found that if some of the matte missed the jets or if the rate of pouring was too rapid, thus preventing the thorough prechilling of the matte, some of the semimolten particles united again into large lumps, frequently causing explosions in the tank. Likewise, too rapid pouring will promote the formation of large granules up to 1 in. in diameter, which are detrimental to good roasting. Too slow pouring allows the matte to chill before leaving the pouring box, thereby causing the opening in the spout to gradually freeze. The right speed was obtained after a little experimentation. A stream of matte as large as 2½ in. in

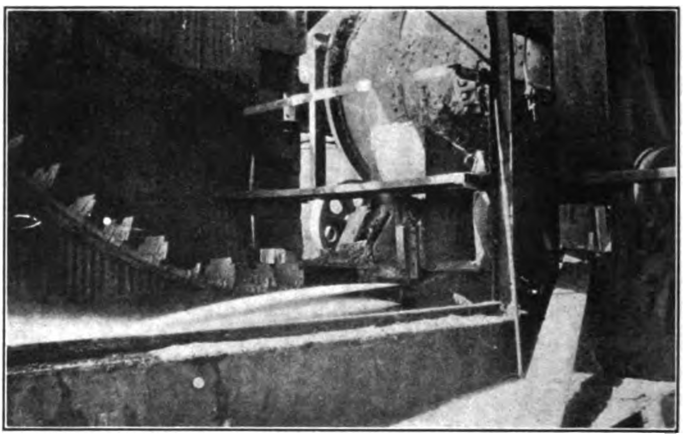

Fig. 2.—Granulation Tank, Showing Jets of Water, Pouring Box and Drag Conveyor.

diameter where it strikes the jets has been granulated satisfactorily. A hot matte makes a better product for roasting, because it is more uniform and finer. A good product is that of which 75 per cent. passes a 1/16-in. screen. The average rate of pouring is 3.2 cu. ft. of matte per minute.

The water jets are delivered through rectangular nozzles, the openings being ⅜ by 3¾ in. with the nozzles 5 in. apart. The supply of water is 100 gal. per minute under a head of 40 ft. The granulating tank is constructed of concrete, 17 ft. long and 7½ ft. wide, the floor being plated with 1-in. cast-iron plates, at a 30° slope into the conveyor trough. This degree of slope has been found sufficient to cause the granulated particles to run into the trough. The drag conveyor is driven

by a 10-hp. motor at a speed of 30 ft. per minute and runs in a sectional cast-iron trough 4 in. deep and 15 in. wide. A very small amount of water, about 5 per cent., passes over with the matte into the railroad cars, drainage being assisted by notching the conveyor flights alternately in the center and on the ends, thus allowing the entrained water to escape and flow back into the tank. The tail sprocket wheel and the idler sprocket wheel (Fig. 1) are both under water. The shafts of these wheels extend through stuffing boxes in the sides of the tank, to the bearings on the outside.

Some lead settles out of the matte while in the container, about 30 to 60 pigs of 65 lb. each, according to the condition of the furnaces, being poured out daily. In case lead goes over and is granulated with the matte, it manifests itself by a sputtering and popping on the surface of the water. After the matte has been poured down to the lead level, the operator turns the container backward and pours the lead into a ladle, whence it is molded into pigs.

To prevent metal losses, the fumes are caught by a swinging hood which fits over the charge opening and connects with the blast-furnace flue leading to the bag house. The hood may be swung back so as to uncover the charge opening, when the crane is ready to pour a ladle of matte into the barrel container.

This installation requires only two men per shift for operation, one to operate the container and one the crane. The forehearth tappers at the furnaces attach the crane hook to the ladles. Under the old method of handling the matte by hand-pots, the cost was 87 c. per ton, which included hand-breaking. Crushing and screening amounted to 56 c. per ton in addition, which brought the total cost to \$1.43 per ton. Granulation costs only 75 c. per ton, which makes the total saving of \$1.43 − 0.75 = \$0.68 per ton of matte granulated.

### *Summary*

| | |
|---|---|
| Matte granulated per 24 hr., tons | 100 |
| Rate of pouring, cubic feet per minute | 3.2 |
| Oil consumed per 24 hr., gallons | 250 |
| Oil per ton of matte granulated, gallons | 2.5 |
| Water used per minute (under head of 40 ft.), gallons | 100 |

### *Screen Analysis on Granulated Matte*

| Inch-Mesh | Per Cent. |
|---|---|
| On 1/4 | 5 |
| On 3/16 | 4 |
| On 1/8 | 14 |
| On 1/16 | 35 |
| Through 1/16 | 42 |
| | 100 |

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# Grain Growth Phenomena in Metals

BY ZAY JEFFRIES, B. S., MET. E.,* CLEVELAND, O.

(New York Meeting, February, 1917)

THE object of the present paper is to enlarge somewhat on the general principles advanced in my discussion[1] of Mathewson and Phillips' article on The Recrystallization of Cold-Worked Alpha Brass on Annealing.[2] It will also serve as an acknowledgment of Prof. Howe's[3] most important remarks on my contribution. In this paper the writer has adopted Prof. Howe's suggestion to substitute the term "germinative temperature" for "critical temperature for grain growth." Instead of being an exact or certain temperature it should be considered that the germinative temperature phenomenon exists throughout a small temperature range.

## *Factors Influencing Fast Growth Phenomena*

The development of grains of macroscopic size at the germinative temperature should be considered the extreme condition of a general rule. When considered in this light it is not strange that the development of these large grains should be the exception and not the rule. The principal factors influencing the operation of the laws of fast growth temperature are as follows:

1. Rate of heating.

In the original paper,[1] the germinative temperature was defined as "the minimum temperature at which two adjacent grains can coalesce to form one larger grain, provided that this larger grain will have sufficiently increased its power of attack, to enable it to absorb adjacent grains which cannot coalesce with each other. Time is always to be understood as a factor governing the first stage of grain growth." If the time of heating is short, the germinative temperature is raised. The formation of large grains in metals may occur at relatively high temperatures when the rate of heating is rapid. It is to be understood that in such cases, grain growth

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[1] *Bulletin* No. 113, p. 987 (May, 1916).

[2] *Bulletin* No. 109, p. 1 (January, 1916).

[3] Prof. Howe's paper will be printed in December *Bulletin*.

must be very rapid, the germinative grains having the power to absorb the inert grains before the latter can coalesce with one another. The rapidity of absorption of the inert grains by the germinative grains is at times almost beyond comprehension. In one instance, the author calculated that 37,500,000 individual grains were absorbed by one germinative grain in 10 sec. The grain growth took place at such a temperature that the inert grains would have coalesced freely with one another, had time been given.

A further increase in rate of heating will prevent the formation of coarse grains. Other influences of change of rate of heating were considered in the original paper.

Figs. 1 and 2 show visually the effect of rate on the formation of large grains. Fig. 1 represents a section of metal held at the fast growth temperature for 30 min. This is a sample of compressed metal heated with electric current in which the temperature gradient was the cause of the selective grain growth. The photograph is at a magnification of 5 diameters and the grains can readily be seen macroscopically. Fig. 2 represents a section of the same metal in which the temperature was

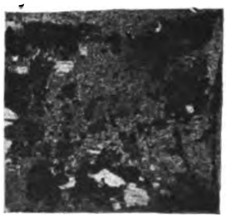

FIG. 1.

FIG. 2.

FIG. 1.—A SECTION OF METAL HELD AT THE GERMINATIVE TEMPERATURE FOR 30 MIN. × 5.

FIG. 2.—SECTION OF THE SAME METAL AS IN FIG. 1, IN WHICH THE TEMPERATURE WAS RAISED QUICKLY TO A POINT FAR ABOVE THE GERMINATIVE TEMPERATURE RANGE, AND HELD THERE FOR 30 MIN.

raised quickly to a point far above the germinative temperature range, and held at this temperature for 30 min. Some of the larger individual grains in Fig. 1 occupy an area equal to about 1,000 average grains in Fig. 2. (The lines in Fig. 2 are cracks which formed in the specimen during mounting.) The mean grain size of the fine-grained areas in Fig. 1 is very much less than the mean grain size in Fig. 2. The grain size at the center of Fig. 1 represents about the normal grain size just above the fast growth temperature. The grain size shown in Fig. 2 is approximately the equilibrium grain size for a temperature several hundred degrees above the fast growth temperature.

Fig. 3 (18 diameters) shows the large grains formed at the germinative temperature in the process of absorbing the equilibrium grains in the hotter portion of the sample.

Fig. 4 (18 diameters) represents the same phenomenon when nearly all of the small grains have been absorbed.

2. *Resistance to grain growth* might be divided into two parts:

(*a*) The resistance to grain growth inherent in the pure predominating constituent; for example, copper in the pure state at a certain temperature has a certain inherent resistance to grain growth.

(*b*) The resistance to grain growth due to obstruction by foreign bodies; for example, copper oxide in copper or pearlite in low-carbon steel.

While it might seem anomalous, high resistance to grain growth due to the second cause greatly favors the formation of very coarse grains at the germinative temperature. As the resistance to grain growth increases in a given material the germinative temperature increases, and the time necessary for the formation of the large grains decreases rapidly. As an example of the above, a sample of metal containing 2 per cent. of non-metallic material required about 1½ hr. to form the large grains at a comparatively low germinative temperature. The same metal containing 10 per cent. of non-metallic material required but 2 or 3 min. to form the coarse grains. The coarse grains in the latter case were larger than those of the former.

In the sample containing 10 per cent. of non-metallics (by volume) the germinative temperature was only 100°C. below the fusion temperature. The addition of 12 per cent. non-metallics was sufficient to raise the germinative temperature above the fusion temperature and hence make it non-existent.

If it is considered that high resistance to grain growth favors the formation of coarse grains, then, naturally, low resistance to grain growth would defeat their formation. The explanation of this is found in the fact that as the resistance to grain growth increases, corresponding to a rise in the germinative temperature, the existing grains at the germinative temperature are very much smaller than the equilibrium grains for that temperature. Since the grains are so far out of equilibrium as regards size, when the germinative grains reach a size considerably larger than the existing grains, the latter are absorbed rapidly; first, because they are so small and, second, because the temperature is so high.

The mechanism of the fast growth phenomena operates in the reverse order for low resistance to grain growth. In the first place the germinative temperature is low and the existing grains at the fast growth temperature are more nearly in equilibrium in regard to size. This, coupled with the actual lower temperature, will materially reduce the velocity of grain growth as well as the size of the maximum grains at the fast growth temperature. In order that selective grain growth may take place at all, it is essential that the existing grains at the fast growth temperature should be smaller than the equilibrium grains for that temperature. As a general rule it might be stated that, other conditions being

equal, the finer the existing grains at the germinative temperature, the greater will be the tendency toward selective grain growth.

3. *Influence of Grain Size Prior to Deformation.*—It is easily possible in pure metals either to help or defeat the selective grain growth at the germinative temperature. If very large grains have been formed by previous high temperature anneal, like normalizing of nearly pure iron such as Prof. Sauveur used in his experiments, it is probable that the grain fragments which formed in the cold-bent specimen after annealing were so large that the germinative temperature phenomena were largely masked. In addition to this, the germinative temperature would be so low that the velocity of grain growth would be decreased. Since the coalescence of adjacent grains in the grain growth region is a function of time, among other things, it is probable that grains not far removed from the centers of selective growth might coalesce with each other before the larger grains in their conquest would have time to absorb them.

While Prof. Sauveur observes that coarse grains did not form in nearly pure iron, it has been the writer's experience that there is nearly always a region in which the grains are larger than those on either side. Sometimes the difference is not great enough to be unmistakably observed in the ordinary microscopic examination, but grain size determinations usually show the difference. A good example of the grain-growth phenomena in a solid solution comparatively free from foreign bodies will be found in Mathewson and Phillips' micrographs.[4]

In a given pure metal with a given amount of cold deformation, the grain size before deformation will have a marked effect on the selective grain growth during annealing. If the initial grain size is large the grain fragments formed during annealing will be correspondingly large, and if the initial grain size is small the grain fragments will also be smaller than in the previous case. The latter condition is conducive to the formation of large grains at the germinative temperature. Instead of normalizing the pure iron, if Prof. Sauveur had given it an initial treatment so as to produce small grains he would have noticed a more marked recrystallization effect after cold deformation with subsequent annealing. On the other hand, if the grains are initially very large it is possible almost completely to mask the fast growth phenomenon.

4. *Temperature and Deformation Gradients.*—The formation of large grains at the germinative temperature is dependent upon a rather even grain size with a temperature gradient or an even temperature with a deformation gradient or any combination of the two. In compressed metals heated by electric current the grain size is usually quite uniform and the temperature gradient serves to start the formation of the large

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[4] C. H. Mathewson and Arthur Phillips: Recrystallization of Cold-Worked Alpha Brass on Annealing, *Bulletin* No. 109, p. 1 (January, 1916).

grains. The greater the temperature gradient, the greater will be the tendency toward selective grain growth.

If a metal is deformed in the cold in such a manner that the degree of deformation differs progressively from point to point—for example, in a bent metal bar (the neutral axis of the bar will have no deformation, but the edges perpendicular to the neutral axis will have a maximum deformation)—there will be a deformation gradient from the surface to the neutral axis. If a bar so bent be heated in a muffle it might be considered that there was no temperature gradient, but the deformation gradient, with temperature constant, would produce a recrystallization gradient in which the parts most severely deformed would recrystallize at the lower temperature. This difference in temperature of recrystallization furnishes the cause for the selective grain growth due to deformation gradients. Various combinations of temperature and deformation gradients might be encountered. For example, the condition might obtain in which the temperature gradient just counterbalances the effect of deformation gradient, and fast growth at the proper temperature would not occur. Local deformation gradients occur in any metal which has been cold plastically deformed, due to the variations in the directional properties and size of adjacent grains.[5]

If the total cold deformation of a piece of metal has been moderate such as would obtain after one or two passes through rolls, the local deformation gradients may give rise to a comparatively few germinative grains which during heating absorb the inert grains thus causing coarse crystallization. As the degree of cold deformation increases, the local deformation gradients will decrease. When the total average cold deformation exceeds about 30 or 40 per cent. reduction in area of the metal, the number of germinative grains formed during normal muffle heating may approximately equal the number of equilibrium grains for the maximum temperature reached during annealing, and thus the metal is said to be fine grained. The above adequately explains the cause of coarse crystallization during annealing of moderately cold deformed metals, and the formation of progressively finer grains with the same heat treatment, as the degree of cold deformation increases.

5. *Influence of Thickness of Sample.*—Stead and Carpenter[6] find that metal sheets which are less than about 0.012 in. in thickness are very much more susceptible to the formation of coarse grains than when the thickness is greater. Their experiments were conducted with electro-deposited iron and the coarse grains formed while cooling from gamma to non-gamma iron. The reasons for the formation of coarse grains are, first, a temperature gradient and, second, the thinness of the metal plates. These conditions will be recognized as ideal for the operation of the tempera-

[5] Mathewson and Phillips: *loc. cit.*

[6] J. E. Stead and H. C. H. Carpenter: The Crystallising Properties of Electro-Deposited Iron, *Journal of the Iron and Steel Institute*, vol. 88, p. 119 (1913).

ture-gradient law, since the progressive change of gamma to non-gamma iron would produce very small grain fragments and even a slight temperature gradient would be sufficient to cause selective growth.

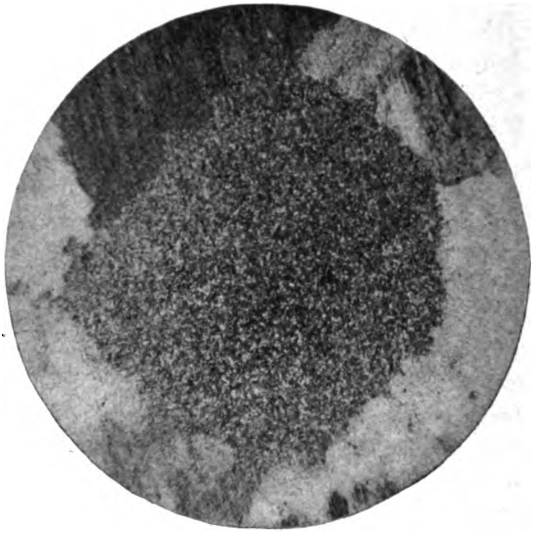

FIG. 3.—THE LARGE GRAINS FORMED AT THE GERMINATIVE TEMPERATURE IN THE PROCESS OF ABSORBING THE EQUILIBRIUM GRAINS IN THE HOTTER PORTION OF THE SAMPLE. × 18.

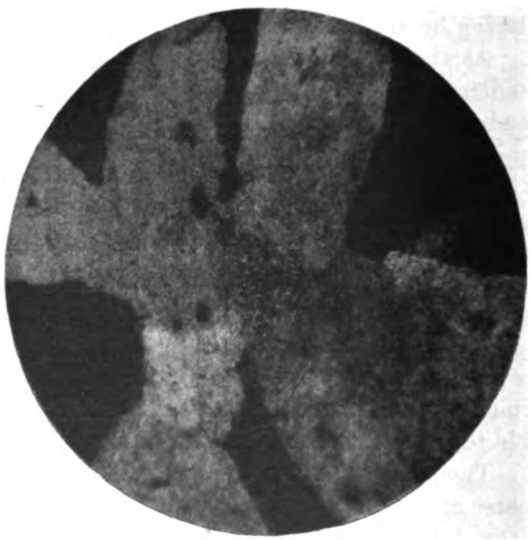

FIG. 4.—THE SAME AS IN FIG. 3 WHEN NEARLY ALL OF THE SMALL GRAINS HAVE BEEN ABSORBED. × 18.

The effect of thickness of section on grain growth is general in metals. The author has found that wires less than about 0.01 in. in diameter

have two regions in which fast grain growth takes place. One occurs at the ordinary germinative temperature. The other occurs at most any temperature above the germinative temperature, the time varying inversely as the temperature. In the second fast growth range the large grains almost invariably first form near the surface of the sample. It is the writer's opinion that in such cases, where the ratio of the surface grains to the interior grains is large, the former, being unbalanced by having only one side in contact with adjacent grains, are readily absorbed by these interior adjacent grains. When the size of the surface grains due to this cause is enough greater than the equilibrium interior grains the latter are absorbed. The formation of large grains caused by small diameters of wires can be materially checked by the introduction of foreign bodies such as non-metallics. The practical effect of the introduction of non-metallics is to decrease the diameter of wire above which coarsening does not take place. In wires having a diameter of less than about 0.002 in. it is doubtful whether the formation of grains extending across the diameter of the wire can be prevented by any means, provided the wire is exposed for a long time at a sufficiently high temperature.

### *Fast Growth Phenomena in Low-Carbon Steel*

What has preceded in this paper has considerable bearing on the formation of coarse grains in low-carbon steel. Referring to Prof. Sauveur's Fig. 27 and Chappell's Fig. 12,[7] there is a good reason why the coarse grains in the latter should be more columnar. In Sauveur's Fig. 27, as the large grains grow toward points of less cold deformation they encounter larger and larger grain fragments. This will be recognized as an adoption of Dr. Mathewson's ideas and nomenclature. The obstruction to grain growth offered by non-metallic globules is such that after the coalescence of the grains the globules do not change position but the grains grow around them. Assuming the same to be the case with pearlite as the obstructing medium (although the ferrite of the pearlite may be divorced, leaving only the cementite as the obstructing medium), the fragments which form in the interior of an original grain of ferrite would not have obstructing bodies at their interior boundaries. In the first stages of selective grain growth the grain fragments, being small, would be absorbed readily one at a time and when the obstructing masses of pearlite were encountered grain growth would easily take place around them. As the grain fragments increase in size their resistance to absorption becomes greater; hence, a point will be reached at which equilibrium exists. This is especially true when considering a definite time period.

On the other hand, as the large germinative grains grow at the expense

[7] C. Chappell: Recrystallization of Deformed Iron, *Journal of the Iron and Steel Institute*, vol. 89, pl. 48 (1914).

of the small inert grains, the latter will have had more time to coalesce with one another as the time of sojourn in the germinative temperature range increases. It is well known that two adjacent grains which are not able to coalesce with each other in five minutes, might do so in five hours. Another factor which should not be lost sight of is that when a metal has been but slightly cold deformed, some of the adjacent grain fragments resulting may be in the same orientation and only be separated by a slip plane of amorphous metal. This amorphous material may be absorbed during heating by the crystalline metal, thus producing what is analogous to coalescence.

In Chappell's Fig. 12, however, the grain growth took place at a temperature well above the Acl point (870°C.) In very low carbon steels in which the mass of ferrite is much greater than that of the pearlite, there is considerable evidence that the pearlite does not change into the equilibrium percentage of austenite when heating above Acl. Quenching experiments on samples which have been previously normalized, invariably show less martensite in the very low carbon steels than would be expected from the equilibrium diagram. When these same steels, according to Osmond[8] are heated to very high temperatures—1200° to 1300° C.—larger percentages of martensite are formed. It is likely, therefore, that there is less austenite formed during the heating of these steels than would be expected from the equilibrium diagram. This is borne out by heating curves and by the occurrence of globules of free cementite. In fact, Chappell's own micrographs would lead one to this conclusion.

It is further probable that germinative grains which had reached con-considerable size below the Acl point, would have greater power to divorce the ferrite from the pearlite than would smaller grains.* Consequently, while the pearlite or austenite might easily prevent the small grains from coalescing with one another, the large germinative grains might have sufficient power to absorb the inert grains. This is precisely what does happen in the case of a pure metal mixed with about 10 per cent. of its volume of a non-metal ic material.

Prof. Howe's suggestion, that about 13 per cent. pearlite might easily offer sufficient mechanical obstruction to grain growth to completely defeat the selective action, is highly acceptable. Before leaving this phase of the subject, it would be interesting to consider the effect of the critical transformations in steel. If a piece of steel were heated by electric current to such a temperature that the inner portion were above the

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*The writer has frequently measured the areas of pearlite in very low carbon steels, and almost invariably the percentage is less than would be expected from the combustion carbon determinations.

[8] *Trans.*, vol. 27, p. 879 (1897). A steel containing 10 per cent. carbon quenched from 1050° C. would consist of 60 per cent. free ferrite and 40 per cent. martensite.

$A3$ point and the outer portion below the $A3$ point, a fast growth temperature condition should exist at the intersection of these two regions. Similarly, during cooling from above the $A3$ point if the inner portion were kept above the transformation point and outer portion below, a fast growth temperature condition should exist at the boundary. By the proper regulation of the heating rate or the cooling rate either large austenite grains or large non-gamma iron grains should be formed.

The existence of gases in metals cannot be neglected as a factor in the resistance to grain growth. It would seem, however, that if gas globules were responsible for the obstruction of grain growth, their presence could be determined by the aid of the microscope using high magnifications. That gases in solution or mechanically entrapped in metals may greatly affect the annealing temperature after cold deformation is shown quite conclusively by Rose[9] and Phelps.[10]

### *Effect of Cold Deformation on Uniformity of Orientation of Original Grains*

Prof. Howe's statement that the uniformity of orientation after cold deformation is greater within the individual original grains than between adjacent grains, is no doubt generally true. It must be considered, however, that in cold-deformed metals there is sufficient difference in orientation within the original grains to be the chief cause of recrystallization on annealing. That we do not always see these differences in orientation may be due (1) to the fact that the diameter of individual grains in which we can study etching pits is usually not more than 0.0002 in. This distance is so small that variations in orientation might easily escape even a close observation. (2) To the fact that the sections (viz., longitudinal) which are usually examined in deformed metals are the most conducive to similar etching tints.

An experiment will demonstrate the points in question very clearly. Small wires 0.003 in. in diameter, were heat-treated in such a way that grains about ½ in. long were formed, each grain occupying the full section of the wire for that length. Pieces of this wire were bent sharply through an angle of about 90° in such a manner that the two ends of a single grain were not internally deformed. In other words, all of the deformation due to bending was confined to a small area near the center of an individual grain. It is obvious that the two ends of the grain in question now have decidedly different orientations. The exact angle of difference is known, namely, 90°. It is obvious that there is no sharp

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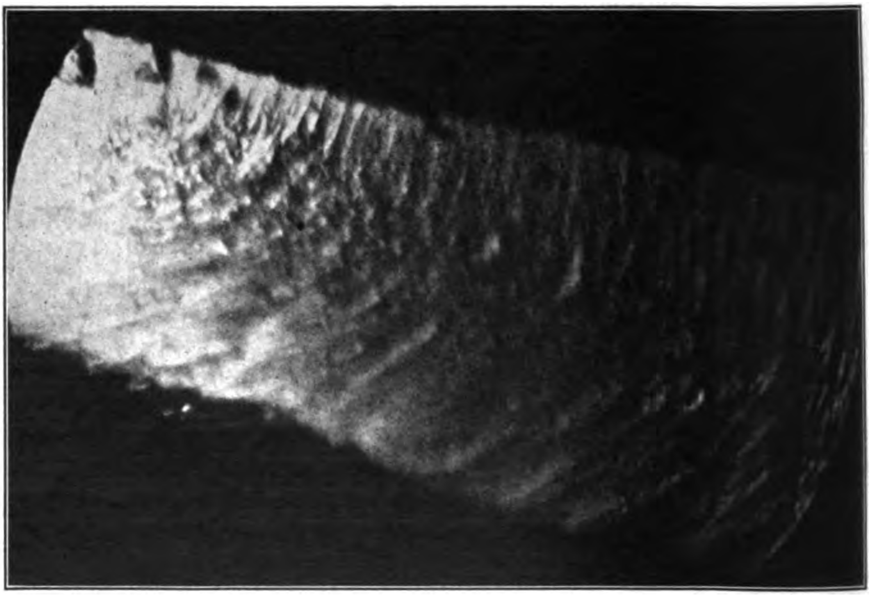
[9] Thomas Kirke Rose: On the Annealing of Gold, *Journal of the Institute of Metals*, vol. 10, p. 150 (No. 2, 1913).

[10] John Phelps: The Effect of Hydrogen on the Annealing of Gold, *Journal of the Institute of Metals*, vol. 12, p. 125 (No. 2, 1914).

line of demarcation at the bend. The change in orientation is progressive. When a sample like the above is examined under the microscope the etching tint is usually quite uniform. However, if the same sample be subjected to a twisting operation the orientation has been changed with respect to different axes, and the etching tint does not remain uniform. Fig. 5 (660 diameters) is a photomicrograph of a sample of wire so twisted. The differences in etching tint can be readily seen.

The recrystallization properties in the bent sample are approximately the same as those in the twisted sample. As further evidence of decided change in orientation within the initial grains, Stead and Carpenter



A SINGLE GRAIN, TWISTED COLD. SHOWS DIFFERENCES IN ETCHING TINT. × 660.

demonstrate this condition very nicely by means of different etching tints within the same initial grain and by drawings (The Crystallising Properties of Electro-Deposited Iron, *Journal of the Iron and Steel Institute*, vol. 88, Figs. 6*A* and 6*B*, Plate 12. Drawings on p. 147 (1913).

No explanation of the recrystallization of cold-deformed metals on annealing has been forthcoming which did not assume a change in orientation within the initial grains due to the deformation. Rosenhain[11] seeks to explain the formation of the new grains by assuming that the amorphous material forms nuclei for the new grains. The study of saturated liquid solutions in the presence of the solid solute teaches us

[11] *Internationale Zeitschrift für Metallographie*, vol. 5, p. 65 (1913).

that the solute in the crystalline state, and not the solute or solvent in the amorphous (dissolved) state, is the controlling factor governing saturation equilibrium. As an example, if a solution of sodium chloride in water is saturated at a certain temperature with no crystalline sodium chloride present, the dissolved salt contains a certain amount of potential energy. This potential energy is not changed in quantity by the addition of a sodium chloride crystal to the solution, but some of the dissolved salt crystallizes and the degree of concentration in the solution is lowered. The force necessary to change this degree of concentration must be looked for in the crystal of sodium chloride. In the presence of amorphous and crystalline metal, therefore, the crystalline fragments and not the amorphous layers will be the controlling factors in recrystallization.

Beilby[12] has demonstrated the above experimentally. This experiment is so basic in its general relationship to recrystallization that it should be repeated here in his own words. "Experiments have been made on the building up of an entirely amorphous mass of gold. Gold was precipitated from a solution of the chloride as a spongy brown powder. The washed and dried powder was compressed in a die after the gases it had contained had been pumped off at a temperature of about 150°. The pressure was applied by a small hydraulic press. This pressure was so great that the gold plug made a depression in the bottom plate of the die, which was of tool steel. The pressure must have been 60 to 80 tons per square inch. The gold squirted very slightly through a hole in the bottom plate, and also between the plunger and the sides of the die. The squirted gold was very hard and rather brittle, quite unlike pure gold prepared in any other way. The specific gravity of the compressed plug was 19.05, or rather lower than the specific gravity of gold. Its microstructure was granular and slightly spongy. This amorphous gold was certainly very much more rigid than even the hardest of cold-worked gold; it was almost devoid of plasticity. Another plug of compressed gold was made from the same powder, but a tiny spiral of very fine soft wire was dropped into the die before it was packed with powder. When a very moderate pressure was reached the soft gold squirted quite freely past the plunger, leaving the plug of hard gold behind.

"The first compressed plug was heated to 350° and showed no development of crystallization. In the absence of crystalline nuclei the amorphous form is stable even at this temperature. The squirted metal from the second plug which did contain nuclei crystallized freely at 350°."

[12] G. T. Beilby: The Hard and Soft States of Metals, *Journal of the Institute of Metals*, vol. 6, pp. 36–37 (No. 2, 1911).

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# Biographical Notice of Charles Kirchhoff

BY R. W. RAYMOND, NEW YORK, N. Y.

(New York Meeting, February, 1917)

CHARLES WILLIAM HENRY KIRCHHOFF was born March 28, 1853, at San Francisco, Cal., where his father, Charles Kirchhoff, was at that time employed in the consular service of his native country, Germany. A few years later, the family moved to Hoboken, N. J., in the public schools of which city the son received his preliminary education, proceeding later to Germany, where he was graduated in 1874 as mining engineer and metallurgist at the Prussian Royal Mining Academy of Clausthal, in the Harz. Upon his return to the United States, he became chemist of the Delaware Lead Refinery at Philadelphia, and retained that position for three years. But in 1877 he began what was to be an almost uninterrupted life-long association with David Williams, the publisher of *The Iron Age*, of New York, who established in that year, and continued for a brief period, a journal entitled the *Metallurgical Review*, on the editorial staff of which Mr. Kirchhoff received a place. The enterprise was doubtless an attempt to cover a wide metallurgical field outside of that which properly belonged to *The Iron Age*. But it was soon abandoned, and Mr. Williams wisely concentrated his energies upon the older journal, which, under his vigorous and skillful management, and the labors of the able editors and correspondents whom he selected, became one of the greatest institutions of its class in the world. From 1878 to 1881, Mr. Kirchhoff was assistant editor of *The Iron Age*. From 1881 to 1884, he was managing editor of the *Engineering and Mining Journal;* in 1884 he returned to *The Iron Age*, to become for five years its associate editor, and in 1889, on the retirement of James C. Bayles, editor-in-chief.

This list of dates and employments, without further comment, sufficiently indicates, at least to the eye of an expert, that Mr. Kirchhoff had found his congenial career in trade journalism. His qualifications for this profession were somewhat exceptional. He possessed the scientific training, the knowledge of foreign languages and literatures, and the power of making and keeping friends, which enabled him to get early notice of technical novelties; he had the taste for statistics which made him both industrious and intelligent in their collection and use; and to these traits he added a mastery of the meaning of such accumulated data, and a

sane, critical judgment of the situations which they represented, as well as of the sources and the figures themselves, which made his opinion weighty concerning them. His estimates and prognostications, expressed in quiet, clear, forcible but unsensational style, were quoted by the non-technical press, when editorial utterances, more brilliant in rhetoric or pleasantry, passed unrepeated. In short, his mind was equipped for the long-distance as well as the near-by view of things; he could see both science and business, and his lens was achromatic at either focal distance. Such men surely achieve reputation and influence, and never lose what they have thus patiently won.

It was in 1883, while he was managing editor of the *Engineering and Mining Journal*, that Mr. Kirchhoff was engaged by the Director of the U. S. Geological Survey to collect and arrange annually the statistics of the production of lead, copper and zinc in this country. Having had, at an earlier period, some personal experience in work of that sort, I feel myself qualified to judge of its difficulties and to recognize success in the overcoming of them. It is not a simple matter of commanding the reports of all producers and compiling the results. It means the personal winning of the confidence of individual producing concerns, one after another; the intelligent reduction of their data to a common standard; the ingenious filling of gaps in the returns; the checking of statistics of production by those of transportation, sales and stocks on hand—in short, the editing, in a critical as well as a mechanical sense, of a mass of material more or less incomplete and heterogeneous. Such a labor, performed by a competent hand through a series of years, doubtless grows easier and easier because it educates the contributors while it makes them increasingly willing to do their part toward a result so creditable and useful. Mr. Kirchhoff's reports, continued for 23 years, constitute a lasting memorial of his own ability and of his influence upon the managers of great national industries.

But a man might possess all the admirable qualities above enumerated, and yet lack the indefinable gift of administration. This gift Mr. Kirchhoff exhibited when in 1904, upon the death of John S. King, business manager of *The Iron Age*, he became Vice-President and General Manager of the great David Williams Co. This responsibility, as well as that of his editorial position, he carried successfully until his retirement from active business at the close of 1909, when *The Iron Age* was sold to other parties. This was the occasion of a luncheon in his honor, given by his friends and associates at the Engineers' Club, Jan. 16, 1910, and made notable by the attendance of many distinguished engineers and captains of industry, including the venerable John Fritz, then 87 years old, who made the journey from Bethlehem for this purpose. Numerous letters and speeches (including the presentation, by Mr. Kirchhoff's associates in the David Williams Co., of a bronze statue) bore witness to the

admiration and esteem with which he was universally regarded. There was general regret over his retirement from a position which he had so signally adorned—a regret from which, as I remember, I ventured to dissent, preferring to praise and congratulate him upon a rest well earned, and taken at a time when, with powers unimpaired, he could both enjoy and make fruitful of good to his fellows a life more wide and free. Alas! it was otherwise decreed.

Mr. Kirchhoff's activity and reputation were not confined to the United States. As editor of *The Iron Age,* he maintained relations with technical and commercial leaders abroad, and frequently visited Great Britain, Germany and France for the purpose of studying the developments of the iron and steel industries of those countries. A series of his articles in *The Iron Age,* republished in 1900 as a book, entitled "Notes on Some European Iron-Making Districts," was widely read. After his retirement, he reviewed in the columns of *The Iron Age* the proceedings of the International Metallurgical Congress, which he attended at Düsseldorf in the spring of 1910; and in the following October he read before the American Iron and Steel Institute in New York a notable paper on the recent progress of iron and steel metallurgy, as reflected in that Congress.

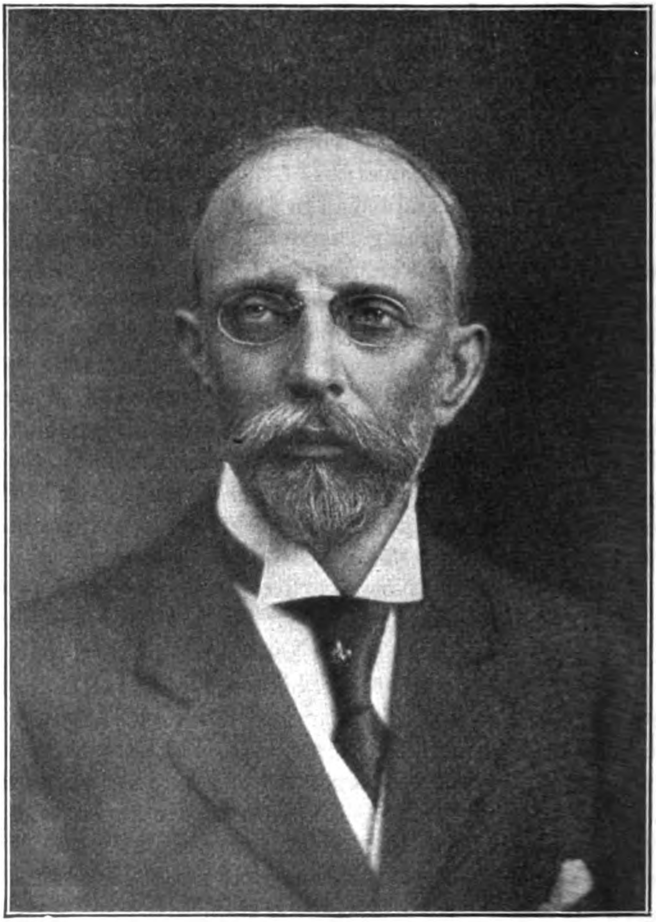
Mr. Kirchhoff was specially active and prominent in connection with the American Museum of Safety Devices and Industrial Hygiene, on the managing board of which he served for several years. His labors in this field were recognized by the government of France, which conferred upon him in 1908 a decoration as *Officier d'Académie.*

He was a member of the American Iron and Steel Institute, the Iron and Steel Institute of Great Britain, the Verein deutscher Eisenhüttenleute, the American Society of Mechanical Engineers, and honorary member of the Franklin Institute of Philadelphia.

He joined the American Institute of Mining Engineers in 1875, immediately after beginning his career as a metallurgical chemist. In 1884, he was strongly supported as a candidate for the office of Secretary, left vacant by the resignation of Dr. Brown. He and I were not only friends but associates at that time on the editorial staff of the *Engineering and Mining Journal;* and these relations were not at all disturbed by our amicable contest for the Institute secretaryship. No one who was present at the banquet held in connection with the Cincinnati meeting of February, 1884, has forgotten the graceful and humorous speech made by Mr. Kirchhoff from the standpoint of the defeated candidate. Thenceforward to the end of his life, he was my strong supporter, wise adviser, and loyal and beloved friend.

In 1887 and 1888, and again in 1892 and 1893, he served as a member of the Council; and in 1896 and 1897, as Vice-President; in 1898 and again in 1911 as President; from 1907 to 1912 as Director, and in 1913 as Past-President. His presidential address, at the New York meeting of Febru-

ary, 1899, on "A Decade of Progress in Reducing Costs," is a classic, showing precisely what such a paper should be, and exhibiting those characteristics of careful and tireless inquiry, comprehensive grasp of details, wisdom in generalization and clearness of statement which I have already noted as the elements of his power and success.



CHARLES KIRCHHOFF.

But not less important—and far more laborious was his work in the memorable year 1890, when the Institute, with the aid of American engineers and ironmasters, entertained some 500 guests, belonging to the Iron and Steel Institute of Great Britain and the Association of German Ironmasters. Mr. Kirchhoff was Secretary of the American Reception Committee, of which Mr. Andrew Carnegie was Chairman. But his most arduous labors were performed in connection with Mr. W. P.

Shinn of the Transportation Committee, for whom he personally conducted one of the two Pullman trains which were kept running through the month of October with the foreign guests as passengers.

For some years before his death, Mr. Kirchhoff fought a brave battle against a disease usually regarded as fatal. With patient and cheerful courage, he held his own for a while, and even seemed to be gaining ground. But it was trench-fighting, and gains were small and slow. An attack of the grip, reinforcing the older enemy, brought the end at last; and he died, July 22, 1916, at his summer home near Asbury Park, N. J. He leaves a widow, two sisters and a brother.

In closing this imperfect sketch of my dear friend, I can present no better summary of his character than that which is embodied in the Minute adopted by the Directors of the American Institute of Mining Engineers upon the tidings of his death, and I therefore repeat as my own this paragraph:

"Mr. Kirchhoff was one who secured, because he practiced, loyalty in friendship, justice and kindness in personal intercourse, openness and equity in business. Modest, but not timid; prudent, but progressive; gentle, but firm; full of chivalry and of common sense; indefatigably industrious and inexhaustibly patient, he won victories without making enemies, and crowned with a heroic death the record of a life unstained."

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# The Function of Alumina in Slags

BY CARL HENRICH, LINCOLNTON, GA.

(New York Meeting, February, 1917)

I HAVE read with particular interest that portion of the discussion by Anton Eilers referring to the high-lime (and also high-alumina) slags made by August Raht in 1881, while smelting the Horn Silver ores at Franklin, Utah. The two analyses of such slags, furnished by Mr. Raht to Mr. Eilers are:

*Horn Silver Slags*

| | Slag I | Oxygen(*b*) percentage | Slag II | Oxygen(*b*) percentage | Average of I and II | |
|---|---|---|---|---|---|---|
| | | | | | | Oxygen |
| $SiO_2$ | 33.9 | 18.1 | 35.0 | 18.7 | 34.50 | 18.40 |
| FeO | 26.8 | 6.0 | 25.4 | 5.7 | 26.10 | 5.80 |
| CaO | 26.1 | 7.5 | 24.9 | 7.1 | 25.50 | 7.30 |
| BaO | 3.2(*a*) | 0.3 | 3.2 | 0.3 | 3.20 | 0.30 |
| $Al_2O_3$ | 10.8 | 5.1 | 9.3 | 4.4 | 10.05 | 4.75 |
| | 100.8 | 37.0 | 97.8 | 36.2 | | |

(*a*) Not determined in this analysis.
(*b*) Not given in Mr. Eilers' paper.

Oxygen ratio, Al as base.

I. O in acid: O in base: : 18.1 : 18.9 : : 0.96 : 1
II. O in acid: O in base: : 18.7 : 17.5 : : 1.06 : 1
Average: 18.4 : 18.15 = 1.01 : 1

Oxygen ratio, Al as acid.

I. O in acid: O in base: : 23.2 : 13.8 : : 1.68 : 1
II. O in acid: O in base: : 23.1 : 13.5 : : 1.76 : 1
Average: 23.15 : 13.4 = 1.73 : 1

By assigning to alumina the rôle of a base, Mr. Eilers deduces the formula $3FeO.2SiO_2 + 4CaO.2SiO_2$.

The nearest approach to a simple formula expressing the average composition of these slags, which I can figure, is

$$9RO.Al_2O_3.6SiO_2.$$

Allowing a ratio of 4FeO : 5CaO, we have for this formula:

$SiO_2$ = 35.0 per cent.; $Al_2O_3$ = 10.0 per cent.; CaO = 27.0 per cent.; FeO = 28.0 per cent.; in round numbers. This would be a singulo silicate of CaO and FeO with about 34 per cent. or one-third of the RO bases replaced by $Al_2O_3$.

However, it has always appeared to me to be an absurdity to force a sesquioxide to replace FeO or CaO or any other RO, simply to make the analysis of some slag or mineral fit into some orthodox formula. That $Al_2O_3$ can be substituted for $Fe_2O_3$ or $Cr_2O_3$, just as MgO, CaO, BaO, FeO, and MnO can be substituted for each other in various chemical combinations, is well established. But I do not know of any natural silicate in which it has been shown conclusively that $Al_2O_3$ would take the place of RO, and I do not believe that alumina will do it in any artificially produced slag silicate.

If in the above slags we consider alumina as an acid, *i.e.*, the slags as mixtures of silicates and aluminates, we find that they satisfy, or closely approach the formula:

$$(9RO.5Al_2O_3) + 6(6RO.5SiO_2).$$

In short, assuming alumina as an acid, $2Al_2O_3$ would be the equivalent of $3SiO_2$, or 102.8 mass-units of alumina would be the equivalent of 90 mass-units of silica, and the oxygen ratio of the correct slag of the above type would be 5 : 3. The slag would be essentially a lime-iron 5 : 3 "silicaluminate" (slightly acid "sesqui-silicaluminate").

Slag No. I corresponds very nearly to this degree with an oxygen ratio of 23.2 : 13.8. Slag No. II does not correspond so well, and it is also metallurgically slightly inferior to No. I as shown by the lead and silver contents. On the supposition of Al as basic, the increase in the silica of this slag should make it better metallurgically than No. I, which, however, is evidently not the case.

I am aware that many metallurgists are unwilling to assign to alumina the rôle of an acid or to consider it as a substitute for silica in the calculation of their slags. Except in iron blast-furnace practice, it is of no great importance whether we consider it as a base or an acid or do not consider it at all. As the insoluble residue of ores or limestone is frequently calculated as silica, it is automatically assigned to the rôle of an acid. But practical metallurgists have, as a rule, avoided the issue by following Kerl's advice and have kept the alumina of their slags as low as possible, so low, in fact, that usually it has been immaterial which way the alumina has been regarded. The metallurgist in a modern centrally situated blast-furnace plant can do this, but occasionally the alumina problem becomes a serious one to the metallurgist at an isolated plant, smelting, perhaps, the ore from only one mine, so that the question has its interest from a practical as well as from a scientific side.

In any slag the chief points of interest are:

The formation and melting points, since these determine coke consumption and furnace temperature; its viscosity, which must be small, both to allow it to flow readily from the furnace and to permit a quick mechanical separation of it from the matte, speiss or metal; the chemical composition, since it must not dissolve large quantities of the valuable constituents of the ore, and must (as in iron smelting) occasionally remove detrimental ones.

In general, an increase of alumina or silica will raise the formation temperature of a slag, and ordinarily it will be made thereby more viscid at or near its formation temperature; but these qualities are likewise affected by the various bases entering the slag and their relative proportions.

As practical experience has ordinarily shown that scaffolds over the tuyères, and hard slag and metal taps resulted from high alumina, among the bases in unisilicate and sesquisilicate slags, alumina has acquired a bad reputation among lead and copper smelters.

I might suggest at this point that it is a well-known fact that suddenly cooled slags ("chilled") are more quickly attacked by hydrochloric acid than if slowly cooled. This suggests that the crystallized and crystalline minerals occurring in slowly cooled slags are not contained as such in the molten magma, but are only formed from the original magma during the cooling process.

Among the cases in which high alumina in the smelting charge became a practical problem were the Horn Silver ores smelted by Mr. Raht in 1881, already mentioned, the oxidized ores of the early days of the Copper Queen at Bisbee and the Detroit Copper Mining Co. at Morenci and my own experiences in smelting some ores of the Champion and United Copper Mines near Nelson, N. Z.

In those times, I suppose, we all considered alumina as a base when we considered it at all. I know that I did when confronted with an unusual amount of kaolinized porphyry in some of the Detroit Copper Mining Co.'s ores. I know, too, that the behavior of the slags calculated in this way was often a sore puzzle to me. For instance, I give below two slags made at Morenci about 1884. No. I was a good slag, keeping the furnace open and running freely, the other was a bad slag, inclined to form noses and scaffolds above the tuyères, thick, sticky and sure to bring the smelting campaign to an abrupt end if not promptly changed.

The oxygen ratio, considering alumina as a base, does not throw any light on the radically different nature and behavior of these slags. The only thing which at that time seemed feasible to me, was to follow Kerl's rules as far as possible, and keep the alumina low.

But when I went from Morenci to Nelson, N. Z., and there faced, unexpectedly, the problem of smelting the copper ores of one mine, with no available flux but magnesian limestone, I found myself "up against

*Detroit Copper Mining Co.'s Slags*

| | Slag I | Oxygen percentage | Slag II | Oxygen percentage |
|---|---|---|---|---|
| $SiO_2$ | 34.3 | 18.3 | 38.1 | 20.3 |
| $Al_2O_3$ | 11.8 | 6.5 | 16.9 | 7.9 |
| FeO | 38.5 | 8.6 | 33.1 | 7.4 |
| CaO | 10.1 | 2.9 | 8.0 | 2.3 |
| MgO | 2.3 | 0.9 | 2.2 | 0.9 |
| | 97.0 | | 98.3 | |

Oxygen ratio,

| | | |
|---|---|---|
| $Al_2O_3$ base: | 18.3 : 17.9 | 20.3 : 18.5 |
| $Al_2O_3$ acid: | 23.8 : 12.4 | 28.2 : 10.6 |

it." The ore was mainly chalcopyrite, with some pyrite, in a gangue locally known as serpentine. The average composition of the country serpentine on both sides of the lode serpentine appeared to have the approximate formula $3MgO.Al_2O_3.3SiO_2$ or about 30 per cent. MgO, 25 per cent. $Al_2O_3$ and 45 per cent. $SiO_2$.

The lode serpentine, forming the gangue of the ore, approached more nearly the composition of $4MgO + 4Al_2O_3 + 5SiO_2$, or about 17.5 per cent. MgO, 43.5 per cent. $Al_2O_3$ and 39 per cent. $SiO_2$. One actual analysis of the gangue of roasted ore figures 36 per cent. $SiO_2$, 46 per cent. $Al_2O_3$ and 18 per cent. $MgO_2$. That silica-alumina ratio in the available ore of the smelting charge was a decided and puzzling novelty. Kerl's advice was evidently not applicable in this case. As I published, in 1886 or 1887, in the *Engineering and Mining Journal*, an account of my experiences in smelting that ore, it would be superfluous to repeat that account here. The following slag analyses and appended remarks will suffice:

| | Slag 1 | Oxygen percentage | 2 | O | 3 | O | 4 | O | 5 | O |
|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 36.4 | 19.4 | 34.6 | 18.4 | 39.1 | 21.3 | 21.8 | 11.6 | 18.4 | 9.8 |
| $Al_2O_3$ | 27.8 | 13.1 | 26.4 | 12.4 | 22.9 | 10.8 | 31.8 | 14.7 | 20.8 | 9.8 |
| FeO | 14.2 | 3.1 | 32.2 | 7.2 | 28.6 | 6.4 | 30.6 | 6.8 | 35.9 | 7.9 |
| CaO | 1.5 | 0.4 | .... | .... | .... | .... | .... | ..... | 10.0 | 2.9 |
| MgO | 18.4 | 7.4 | 6.8 | 2.7 | 8.4 | 3.3 | 15.5 | 6.2 | 14.3 | 5.7 |

Oxygen ratio,

| | | | | | |
|---|---|---|---|---|---|
| Al basic: | 19.4:20.4; | 18.4:22.3; | 21.3:20.5; | 11.6:27.7; | 9.8:26.3; |
| | 0.95 | 0.83 | 1.05 | 0.42 | 0.37 |
| Al acid: | 32.5:10.9; | 30.6:9.9; | 92.1:9.7; | 26.8:13.0; | 19.6:16.5; |
| | 2.98 | 3.09 | 3.32 | 2.02 | 1.19 |

| | 6 | O | 7 | O | 8 | O | 9 | O |
|---|---|---|---|---|---|---|---|---|
| $SiO_2$............ | 19.7 | 10.5 | 22.6 | 12.0 | 24.2 | 12.9 | 23.9 | 12.7 |
| $Al_2O_3$............ | 18.5 | 8.7 | 22.5 | 10.6 | 26.2 | 12.3 | 26.8 | 12.6 |
| FeO............ | 28.5 | 6.3 | 32.7 | 7.3 | 32.1 | 7.1 | 33.5 | 7.4 |
| CaO............ | 15.0 | 4.3 | 5.1 | 1.4 | 8.0 | 2.3 | 6.4 | 1.8 |
| MgO............ | 18.3 | 7.3 | 18.4 | 7.4 | 9.2 | 3.7 | 9.4 | 3.8 |

**Oxygen ratio,**

**Al basic:** 10.5:26.6; 12.0:27.8; 12.9:25.4; 12.7:26.6;
0.40 0.43 0.51 0.50

**Al acid:** 19.2:17.9; 22.6:15.2; 25.2:13.1; 25.3:13.0;
1.07 1.49 1.92 1.95

1. Slag of first trial run—very short run. Scaffolded and froze up. Furnace: 36-in. round water-jacketed (1884) Arizona copper furnace.

2. Slag from same run, when furnace began to scaffold.

3. Same run. From chilled scaffold above tuyères.

4. From second equally short run. Somewhat better roasted ore, more coke and somewhat higher blast pressure. Thick, gluey slag from slag-tap. Scaffolding.

5. From third, successful run, this time considering alumina as an acid, and counting 35 weight-units of alumina the equivalent of 30 units of silica. Blast pressure: 9 to 10 oz. First slag fall, changed to:

6. By addition of more limestone flux to the charge. This is a very free-running light slag, separating easily and clean from the 45 per cent. copper-matte. Thirty-six-inch furnace running at the rate of 45 tons smelting charge in 24 hr.

7. Slag from last 12 hr. of run, finishing roasted ore on hand, and with shortage of limestone flux, which could not be replenished in time. Furnace not running as free as on No. 6 slag, only at the rate of 27 tons in 24 hr. As available limestone flux diminished, Slag Nos. 8 and 9 ensued, which allowed to finish the campaign with increased fuel and

| | Slag | Oxygen percentage |
|---|---|---|
| $SiO_2$.................................. | 26.6 | 14.2 |
| $Al_2O_3$.................................. | 15.4 | 7.2 |
| FeO.................................. | 42.6 | 9.5 |
| CaO.................................. | 9.5 | 2.7 |
| MgO.................................. | 0.2 | 0.1 |

**$Al_2O_3$ as a base; Oxygen ratio:** 14.2:19.5.
0.73

**$Al_2O_3$ as an acid; Oxygen ratio:** 21.4:12.3.
1.74

blast, using up all roasted ore on hand. If continued on this slag, however, the furnace would surely have scaffolded and frozen up.

An instructive slag, high in alumina, is the preceding typical Copper Queen slag of about 1884:

Considering the composition and behavior of all the slags given above it seems to me evident:

1. In slags containing larger quantities of alumina, the alumina should be considered as an acid, replacing silica, and not as a base.

2. The higher the percentage of alumina, the nearer the slag should approach a "singulo-silicaluminate," *i.e.*, the nearer the oxygen ratio of the bases should come to the combined oxygen of the silica and alumina.

3. An increase in magnesia calls for a higher percentage of bases, while absences of magnesia, and a pure limestone as a flux, will permit an approach to a "bi-silicaluminate" slag.

4. The safe way will be to start with a "sesqui-silicaluminate" as the type of slag to be produced:

$$x(4RO.3SiO_2) + y(2RO.Al_2O_3),$$

which is approximately the slag actually, successfully and involuntarily, I suppose, made in 1881 by August Raht, while smelting very aluminous and very limey ores from the Horn Silver Mine.

There has always been a bias among metallurgists in favor of assigning to alumina the rôle of base in the composition of the slags. This has been natural, as the textbooks on mineralogy assign to alumina in the composition of the silicate minerals the rôle of a base, with very few exceptions. Hence, not questioning the correctness of this authoritatively assigned basic function of alumina, expressed as such in all the orthodox formulas of these minerals, it seemed only proper and correct to consider $Al_2O_3$ as a base in the artificial silicates, the slags. When this view played the mischief with the running of the furnace, this was ascribed to the innate depravity of alumina. You could support a little of it, but you must not allow too much of it, or it would get the better of you. That was about the practical conception in lead and copper smelting, using low furnaces, cold blast, low pressure, and under the necessity of keeping the fuel cost down.

At least, this accurately describes my own view of the case in 1884, when confronted with kaolinized porphyry of the Detroit Copper Mining Co. But having had forced upon me the conviction of the acid character of $Al_2O_3$ in the artificial slag silicates, or at least in some of them, it has engendered some doubt whether alumina always has that function of a base in the natural silicate minerals assigned to it in the orthodox formula of silicates, a question with which I purpose dealing at some future time.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Countercurrent Decantation*

BY LUTHER B. EAMES,† E. M., TIMMINS, ONT., CANADA

(New York Meeting, February, 1917)

THE recovery of dissolved gold from slime pulp in the cyanide process was first accomplished by intermittent decantation. This simple process consists in mixing with the pulp containing the values in solution, a solution of lower gold content, settling the mixture in a tank and decanting the clear supernatant fluid. The thick pulp remaining in the tank is pumped to a second tank together with more barren solution and again settled and decanted. After several repetitions of this operation, values are so far reduced that further washing is not profitable. The gold recovery of this process is high, but the plant required is bulky, labor cost is high and the amount of solution to be precipitated is excessive.

## HISTORICAL DATA

As early as 1901, a plant was built in the Black Hills of South Dakota by John Randall, employing the same principles but attempting to make the process continuous by substituting for flat-bottomed tanks, cones which operated continuously, receiving a constant feed and discharging a steady stream of thickened pulp. These cones were operated in series, the thick underflow of the first one forming, with a stream of diluting solution, the feed to the second cone of the series. Barren solution was added to the tank immediately preceding the discharge tank and, after being slightly enriched by the low-grade pulp in this tank, overflowed to form a diluting solution again for the richer feed entering the third tank from the end of the series, and so on back to the richest tank of the series. Clear water was used for the wash in the final tank. This is the principle on which all successful countercurrent decantation plants operate at the present time, but Randall's plant was not successful because of mechanical difficulties in getting a continuous thick discharge from his cone tanks. A similar plant was built in South Africa although there the washes were not repeatedly used, as in Randall's case, but were precipitated after each contact with the ore. This also was abandoned

* Also published at this time in the *Bulletin* of the Canadian Mining Institute.

† Mill Superintendent, Hollinger Consolidated Gold Mines, Ltd

because of mechanical difficulties and the cost of precipitating the large quantities of solution that had to be used. For a number of years the process was not used, and it was not until the introduction of the Dorr thickener that the minds of metallurgists began to turn again to the continuous decantation principle.

In 1910, two decantation plants were built making use of flow sheets similar to that used by Randall nine years before, but substituting Dorr thickeners for the cones. One of these was at Mocorito in Sinaloa, Mexico, and was installed under the direction of C. Dupre Smith, while the other was designed by J. V. N. Dorr, assisted by the writer, for the Vulture Mines Co. of Wickenburg, Ariz. While perhaps not perfect at first, both of these pioneer plants were so successful as to encourage further installations, few and scattering at first but in considerable numbers during the past three years.

## Theoretical Considerations

In view of this increasing importance, the following discussion of the principles and characteristics of the process is offered. For the purpose

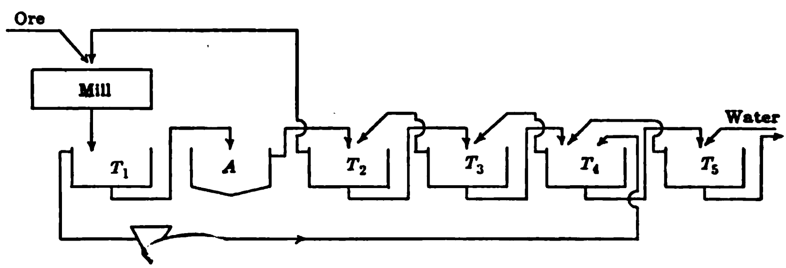


Fig. 1.—Typical Flow Sheet of Countercurrent System.

of investigation a simple yet typical flow sheet has been selected. This is shown in Fig. 1.

This flow sheet assumes that crushing is done in solution, the overflow from the tank $T_2$ being used for the crushing solution. This crushing solution leaves the grinding circuit with the ground pulp and enters $T_1$, and that part which does not pass to the agitators with the pulp overflows $T_1$ and goes to precipitation. After depositing its gold contents, it is used to dilute the underflow of $T_3$ as it enters $T_4$. The overflow of $T_5$ is also mixed into the feed to $T_4$. The overflow of $T_4$ mixes with the underflow of $T_2$ to form the feed to $T_3$, and so forth, as indicated in the flow sheet. At each succeeding mixture the solution meets a pulp of higher dissolved content than itself and is enriched while the pulp is correspondingly impoverished. The pulp at each step approaches the discharge end of the mill while the solution goes to the feed end—hence countercurrent decantation.

*Variables Affecting Decantation Process*

The principal variables that may affect the efficiency of the process are:

1. Grade of ore.
2. Ratio of solution precipitated to ore treated.
3. Thickness at which pulp can be discharged.
4. Cost of chemicals.
5. Rapidity of dissolving, and the place in the circuit where it takes place.
6. Efficiency of precipitation.

Since the decantation process is one involving volumes and dilutions, it is possible to calculate accurately what distribution of values should take place under any given set of conditions. As far as possible, each one of the above variables has been mathematically considered independently of the rest and the results have been plotted.

The effect of variations in the grade of ore is shown in Fig. 2 and scarcely needs comment. In practice, of course, no such increase as is shown in the gold-solution curve would ever be allowed to take place on account of the difficulty of precipitating such high-grade solution and the danger of leakage, but the graph shows conclusively that all solutions increase in value in direct proportion to the increase in the grade of the ore and that the higher-grade solutions increase at a much more rapid rate than the final washes.

The important part played by the ratio of solution precipitated to ore milled is shown in Fig. 3. For the particular grade of ore considered, which in this case was $10 recoverable per ton, with precipitation to 3 c., the economic ratio may be roughly determined by inspection of the lowest curve, which represents the value of the solution leaving the last tank of the series with the tailing. It will be noted that the loss in gold increases very fast as the amount of solution precipitated is decreased, while after a certain point the increased recovery due to increasing the volume precipitated is very slight.

The final choice of the exact ratio to be used must be influenced by the cost of precipitation, which is mainly the cost of zinc. To show clearly the effect of precipitation cost on the economic precipitation ratio, Fig. 4 was plotted. I have considered it safe to assume that the cost of increasing the amount precipitated is due to the additional zinc used. The loss in dissolved gold and the cost of the zinc used in precipitation are both plotted to the same scale in cents per ton of ore. By adding the ordinates of these two curves a third is formed which represents the total loss in gold and zinc. The lowest part of this curve represents the economical range and indicates that for a $10 ore, under the conditions assumed with zinc at 26 c., from 200 to 250 tons should be precipitated

per 100 tons of ore treated. In dotted lines are shown the corresponding curves for a \$5 ore under similar conditions. In this case the range is from 150 to 200 tons. While there is a considerable range within which practically identical results may be expected, it is apparent that each operator should figure out for his own conditions just what his range is.

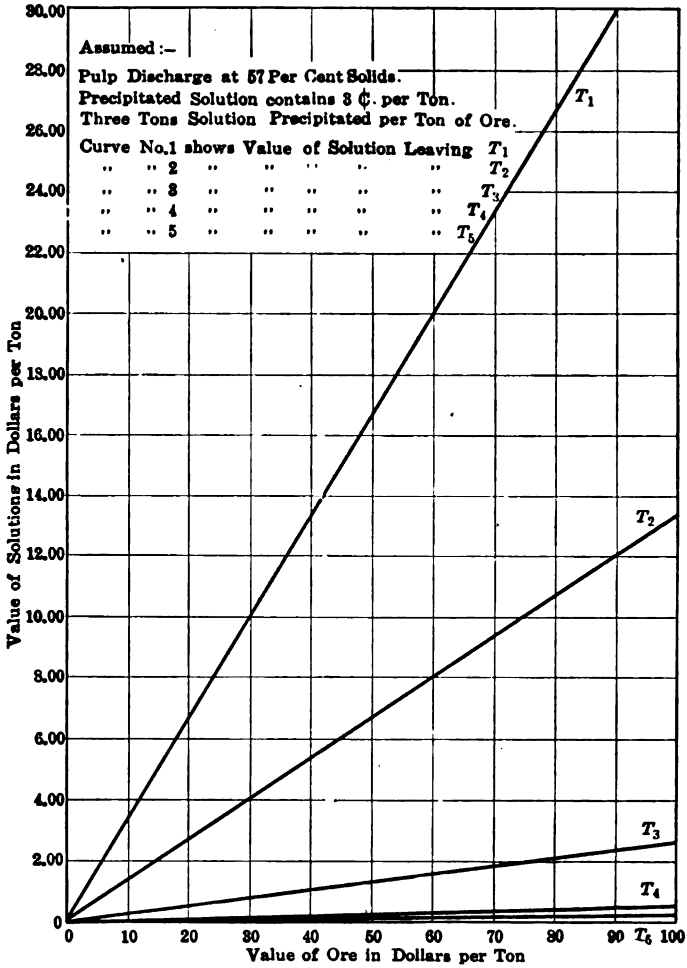


FIG. 2.—EFFECT OF VARIATIONS IN GRADE OF ORE TREATED.

The next, and one of the most important variables to be considered, is the thickness, or percentage of solids, to which pulp can be settled. In Fig. 5 the values of the various overflows have been plotted so as to show the effect of variations in the moisture in the underflows. The full-line curves represent the values of the solutions—that is, they are shown as per ton of solution—but in calculating losses per ton of ore, the ratio

of the solution to the ore present must be considered. The loss in dissolved gold per ton of dry ore for the last tank has, therefore, been plotted in a separate curve, and it is this curve which should be given the most serious attention in determining the suitability of an ore for decantation. The moisture in the pulp also has a direct bearing on the cyanide loss,

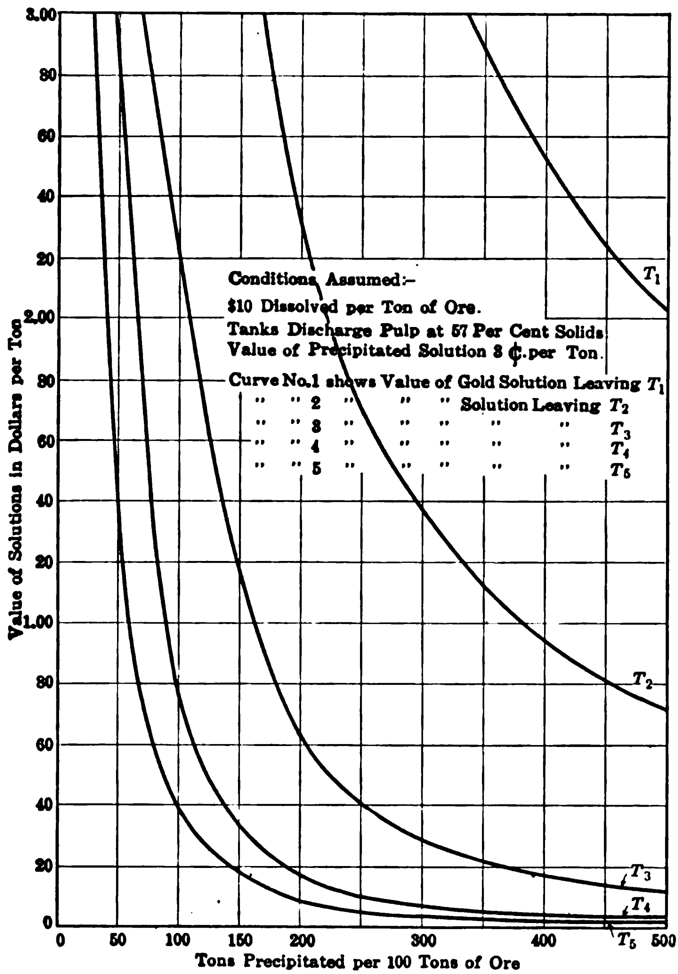


FIG. 3.—EFFECT OF VARIATION IN THE RATIO OF SOLUTION PRECIPITATED TO ORE TREATED.

which is also shown in Fig. 5. This has been shown in pounds rather than in cents because solution strength and the price of cyanide both affect its value. Enough is shown, however, to make it plain that there is a very decided limit to the density of the pulp that can be handled economically, and one is forcibly reminded that the cyanide strength

should be kept as low as possible. Operators as a rule seem to be inclined to "play the game safe" as regards solution strength and it is probable that in many cases cyanide could be saved without any considerable loss in gold by using solutions of a lower strength.

In making the calculations upon which the foregoing curves are based, it was assumed that 75 per cent. of the gold was dissolved in the grinding department and the remaining 25 per cent. in the agitators. This is, of course, an approximation which cannot be accurate, and even with the cleanest ores some gold is dissolved in passing through the tanks following the agitators. This is a condition that has always been met, no matter

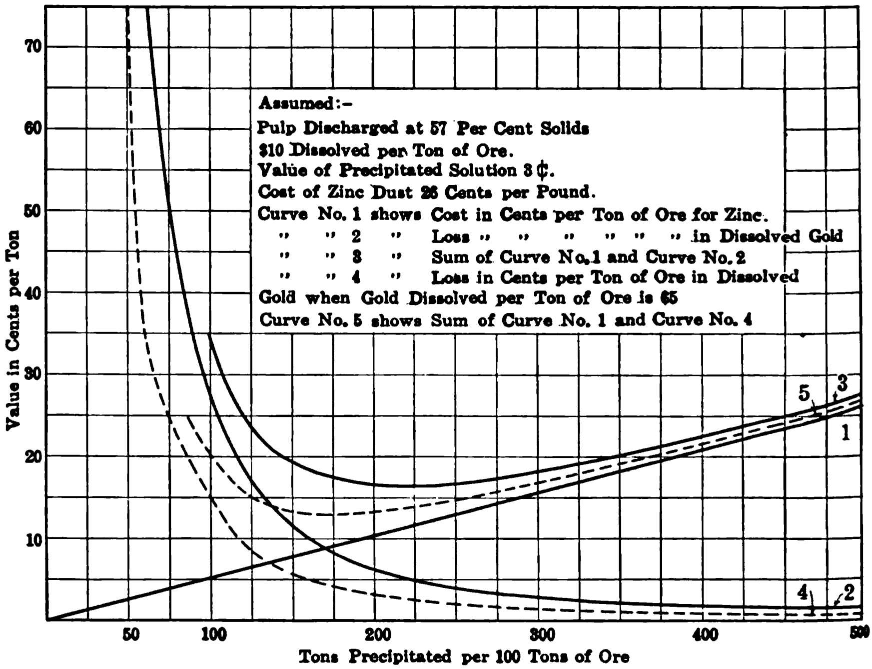


Fig. 4.—Method of Determining the Proper Ratio of Solution Precipitated to Ore Treated.

what the method of recovering the dissolved value. Changing solution during agitation has been practiced for years in the treatment of silver ores, and in the treatment of concentrates at Goldfield with a view to reducing this lag of dissolving. I have also found that on Goldfield ore, pulp from the final agitators, reagitated without change or addition of solution, would give up no more gold, while reagitated filter tails from the same ore would show a distinct reduction. The same condition exists in the pulp fed to decantation plants and there is little doubt that some dissolving takes place even in the last tanks of the series. Some of this value is lost, particularly that freed near the final discharge,

but much is recovered that would probably not be won by any other method. The superintendent of one of the decantation plants treating Tonopah ore informed me that his sampling indicated that the additional dissolving taking place in his plant was sufficient to offset his entire dissolved loss and 3 c. per ton additional.

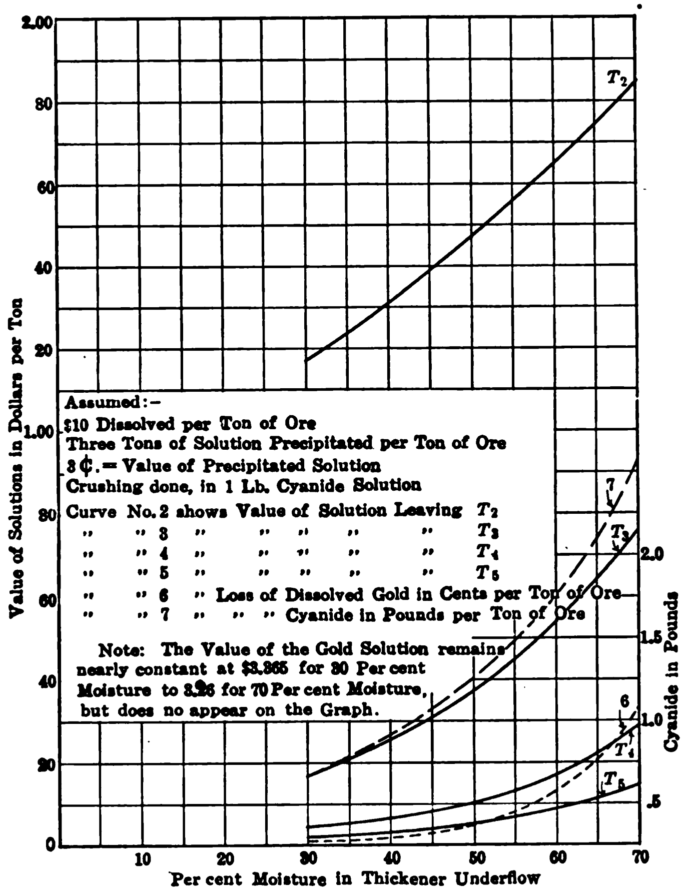


FIG. 5.—EFFECT OF VARIATIONS IN THE PERCENTAGE OF MOISTURE IN THE THICKENER UNDERFLOW.

Any considerable dissolving during decantation will be indicated by a difference in the assay value of the solution in the underflow of the tanks as compared with the overflowing solution. In practice there is always more gold per ton in the underflow solution than in the overflow of any given tank, but in the ores of Porcupine district this difference is very small. Other causes may, and no doubt do, tend to produce this difference between the overflow and the underflowing solution. Adsorption is probably the most important and perhaps the least understood of these. In the case of the ores of the Porcupine district this phenomenon

is of small importance, as the ore is composed of crystalline schists and quartz and there is little tendency for the ore to flocculate under the influence of the solutions used. The gold and silver ores of the Western States are in many cases in eruptive rocks; these ores usually flocculate in solution and in doing so seem to entrap a portion of the value in the solution. At any rate there is a much more noticeable difference in the assays of tank effluents in the treatment of these ores. Faulty mixing of the products fed to the tank has in some cases been blamed for this. In most plants riffles and other devices are put in launders to insure thoroughness in mixing, but our experience in Porcupine would indicate that the ordinary launder makes a perfect mixture. Adsorption, then, would seem to be the more important cause of these differences, and should be a profitable field for further investigation.

Proper precipitation is essential in decantation, as it has always been in every other process, and, as shown by Fig. 6, dissolved gold is lost in proportion to the amount of value in the barren solution used. It will be observed, however, that the loss does not increase as fast as the value of the barren solution does.

It will be observed that the value of the solution leaving $T_4$ increases at nearly the same rate as the barren solution and that all the preceding overflows increase by exactly the same amount as $T_4$. The water dilution cuts the final loss down to a slower increase than the barren solution itself.

## Mechanical Features

### *Dorr Thickener*

The mechanics of the decantation system is of the simplest, but is worthy of study none the less. The Dorr thickener is universally used for the separation and is so well known and so standardized that it need only be mentioned in passing. It has been fully described by Mr. Dorr himself in a paper read before the Institute.[1]

The arrangement of the tanks in plan may be largely governed by space available and other local conditions. In elevation it is very convenient to have the last tank the highest and the preceding ones successively lower, so that the overflowing solutions may be transferred by gravity with a minimum of attention. Where such an arrangement is not possible, the tanks may all be placed on the same level using automatically regulated air lifts to transfer the solutions from tank to tank.

### *Diaphragm Pumps*

As a means of transferring pulp, the diaphragm pump has distanced all competitors. The main reasons for this are that it not only pumps

[1] The Dorr Hydrometallurgical Apparatus, *Trans.*, vol. 49, pp. 211–237 (1914).

but at the same time measures the volume of pulp transferred. The flow of pulp is not easily obstructed by foreign matter, such as chips, waste and

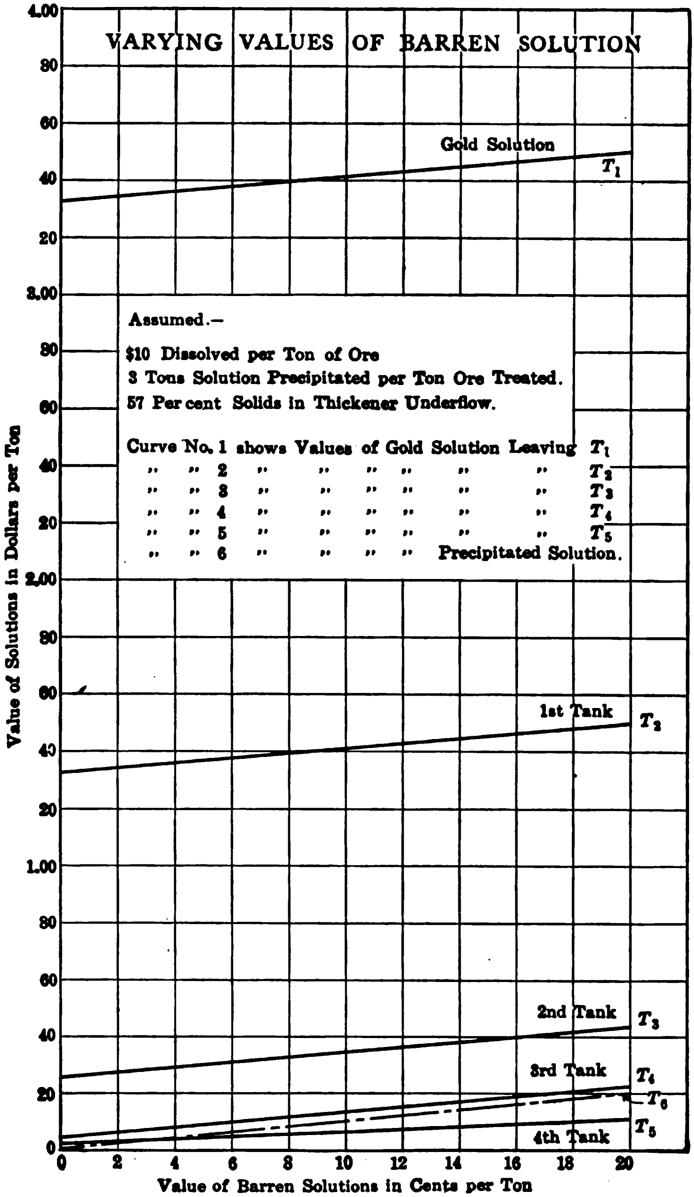


FIG. 6.—EFFECT OF VARIATIONS IN THE VALUE OF BARREN SOLUTION.

the like, since the openings are full pipe size straight through the pump. The capacity of the pump can be controlled with certainty by means of

cone pulleys. The attendant's duties are all on one floor, as it is operated from the overflow level instead of at the bottom of the tank. Operating cost is very low on a properly designed pump.

Air lifts were used for a time, owing to their cheapness and simplicity, but they are hard to regulate and are wasteful of power. The most careful watching will not prevent them from "running away," that is, transferring too fast, consequently thinning the pulp and sending large quantities of rich underflow solution toward the discharge. On the other hand, if they are set too slow the gradual thickening of the discharge slows up the flow and finally stops it.

Spigot discharge to a bucket elevator or centrifugal-pump sump has also been tried, pulp and solution being mixed and elevated together. There are several objections to this arrangement, the principal one being high power and maintenance cost for the pumping or elevating machinery. Both the solution and the pulp must be lifted more than the full height of the tank for each decantation. The small high-velocity stream of pulp passing through a spigot at relatively high pressure is subject to frequent stoppages due to foreign matter, and even when this does not cause a complete plugging it interferes with uniformity in operation. There is also the objection that work is done on two floors, one below and one above or near the top of the tanks.

There has been a certain amount of prejudice against diaphragm pumps due to the fact that some of the earlier pumps were poorly designed for the work they had to do. Faulty valves were responsible for much of this. Poor methods of regulation also had their effect.

The practice in most plants in the Porcupine district now is to use cone pulleys for the regulation of capacity, although some of the operators favor regulation by varying the length of stroke.

The valves should be of the floating type, as any hinge device will catch the wood chips that are present in the best-screened pulp. The chips lodged in the hinge of the valve cause leaks which, though small in amount at first, cause cutting of the seat and consequently permanent leakage. With the floating valve there is no place for chips to lodge and the whole circumference of the seat is washed by pulp at every stroke. This type of valve also has the advantage that the lower valve may be placed directly below the upper one and made small enough to be lifted out through the upper-valve seat when the upper valve has been removed. No tools are required for the removal of these valves and it is a simple matter to inspect them.

The best results have been obtained when the working surface of both the valve and seat were of high-grade rubber. Belting was used at first, but it was found that minute leaks were almost sure to start, due to the fact that belting is not yielding enough to close over any chip that may lodge on the valve seat, and that a leak once started would ruin both

valve and seat in the course of a few days. On the other hand, valves of rubber seating on rubber have operated 6 months without the slightest decrease in efficiency and with scarcely perceptible wear.

Diaphragm pumps have been operated at speeds varying from 15 to 100, the higher speeds usually in conjunction with a short stroke.

The practice at the Hollinger mill has been to use a low speed and a stroke as long as the diaphragm could safely stand. Measurements taken on the Hollinger pumps equipped with standard No. 4 Gould diaphragm at 3-in. stroke gave results as shown in the following table:

| Number of Strokes | Volume per Stroke, Cubic Feet | Specific Gravity | Per Cent. Solids | Tons Solids Pumped per day | Per Cent. Increase in Speed | Per Cent. Increase in Volume per Stroke | Per Cent. Increase in Tonnage |
|---|---|---|---|---|---|---|---|
| 14.5 | 0.139 | 1.54 | 54.5 | 76.3 | | | |
| 23.0 | 0.148 | 1.48 | 50.5 | 114.5 | 58.5 | 6.5 | 50 |

From the above figures, which are typical of numbers of tests made, it may be inferred that the volume pumped is roughly proportional to the speed of the pump but that leakage is slightly greater on the lower speeds.

The low speeds and the placing of the discharge lips high enough above the discharge valve to leave 3 or 4 in. of pulp over the valve at the end of the upstroke have rendered the pumps of the Porcupine district practically free from the splash and dirt that have been one of the chief objections to diaphragm pumps in the past.

Where tanks have a settling capacity of over 125 tons of solids per day, it may be found advisable to use two diaphragms in parallel, making a duplex or even a triplex pump. This arrangement has several advantages; with the lowered speed, strains of the pump are lessened and distributed, and repairs can be made on one unit without complete stoppage of the tank discharge. Diaphragm life appears to be roughly proportioned to the number of strokes, so that an increase in the number of diaphragms employed, with a corresponding decrease in the strokes per diaphragm does not result in an increased cost for diaphragms.

As a safeguard against waste and other foreign matter it is advisable to screen the pulp before it goes to the decantation tanks, and where small wood chips are to be expected a fairly fine screen, usually of punched plate, will be of great service in protecting thickeners and pumps.

### *Measurement of Solution Precipitated*

Every cyanide plant has some method of determining the amount of solution precipitated, and in decantation plants having only one series

of tanks the entire tonnage of precipitated solution is used at one place. If, however, more than one series of tanks is used it becomes important to split this precipitated solution into parts proportional to the tonnage of ore being washed in each series. This can not be done by regulating the valve at the outlet of the barren line, as trial has shown that under some conditions an increase of as much as 50 per cent. can be made in the amount flowing from a pipe line without any visible change in the stream.

This difficulty has been overcome by the use of V-notch weir boxes, one for each series of tanks. In each weir box is a float compartment and a float operating an indicator on a scale which reads in tons per 24 hr. While the weir box, which can be readily made at any mine, is not a recording instrument but gives only an instantaneous rate reading, its use greatly simplifies the problem of proper distribution of barren solution. Water also should be added in proper amounts and uniformly distributed. A smaller weir box has been found convenient for this purpose.

It is usual to determine the specific gravities of various pump discharges about a decantation plant at least once a shift. In a small plant this consumes little time but if there is a large number of measurements to be made the distance to be covered by the operator is considerable, especially if he has to return to a central point for each weighing. In such plants, a spring balance with a dial and revolving hand, that can be carried about the plant, is a good type. I have used a milk scale having an adjustable tare-indicating hand, which makes one complete revolution for 10 lb. I use a narrow-necked can which holds just 10 lb. of water when level full. This makes it possible to add a paper dial which can easily be divided so as to read directly specific gravities. The large sample makes possible accurate readings and enables the operator to determine gravity at the place where the sample is taken.

### *Tray Thickeners*

In his paper on the Dorr metallurgical apparatus, Mr. Dorr touched on the future of tray thickeners, and spoke of the possibility of a complete decantation plant being installed in one tank. While this has not been done as yet, a four-step decantation followed by a continuous filter is in operation, the four steps of the decantation being completed in two tanks equipped with single trays. While no figures have been made public the results are said to be creditable.

Another use of the tray in decantation, which is shortly to be tried at the Hollinger mill, is to increase the capacity of existing plant without increasing the number of tanks installed. Since the same grade of solution is handled in both tank and tray there is no danger of any

mixing of solutions of different grades. This plan should decrease the cost of buildings, tank foundations and building site in almost direct proportion to the increase in capacity gained, while tank cost, power and labor should all be decreased to a marked degree. The outcome of this experiment should have a direct bearing on the future of the decantation process. It must, however, be borne in mind in this connection that some slimes require time for their final thickening and consequently necessitate a tank of some depth, while other slimes find their capacity limit in the thin-pulp settling rate. For this latter class, as explained by Messrs. Coe and Clevenger in their paper on slime settling,[2] a shallow tank or tray is sufficient while the former class requires a tank of carefully calculated depth to give the required time for thickening.

## The Decantation Plant of The Hollinger Co.

The Hollinger decantation plant consists at present of five rows of 40-ft. tanks, four tanks to a row, forming a plant of five units. The tanks are arranged with a difference in elevation of 2 ft. 6 in. between steps with the final tanks of the series the highest, so that all solutions gravitate through and out of the plant to precipitation. The Barrett specification roof is supported on flat trusses, the lower chords of which pass just above the tank rims. These trusses also serve to support the thickener mechanisms and the walks between the tanks.

The diaphragm pumps used were designed by the company's staff, and have been very reliable and economical. They are all three-throw or triplex pumps so that in spite of the large tonnage handled the duty on each diaphragm is light. It is not uncommon for diaphragms to last 300 days while the life of the present type of valves and seats has yet to be determined.

The pumps are used not only for pulp transferral, but also for the final discharge. This makes regulation of the final discharge for moisture much easier, more reliable, keeps the work of the operator all on the upper floor and allows the tailing to be discharged at a considerably greater elevation than would otherwise be the case.

The barren solution and water wash added to each row are measured by separate float-reading weir boxes assuring uniform results from the various units.

### *Labor*

The plant is operated by one man per shift who oils all machinery, watches and adjusts the pumps and records their performance. The

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[2] Laboratory Method for Determining the Capacities of Slime-Settling Tanks, *Bulletin* No. 111, p. 597 (March, 1916).

solution man makes titrations and regulates the addition of water solution but has no other duties in the decantation plant. A repair man on day shift makes all repairs and has time for other work.

### *Power*

The power for each tank including motor and line-shaft losses is under 1 hp., while each three-throw pump consumes about the same amount.[3]

### *Cost*

The costs for the 12 weeks from Jan. 28 to Apr. 21, 1916, have been taken as typical of what is done by this plant at its present capacity. During this time 85,854 tons were decanted at a cost of $599 for supplies, including power, and $1,194 for labor, or $0.007 per ton for power and supplies, and $0.0139 for labor, making a total of $0.0209 per ton for decantation. Labor is no doubt higher here than it will be in the future, as a greatly increased tonnage is to be treated while supplies and power should remain nearly the same. The cost as it stands is about 40 per cent. of the cost of filtering on leaf filters at about the same daily tonnage.

### *Extraction and Recovery*

In the ores of the Porcupine district the recovery by dilution seems to be almost the theoretical maximum. Adsorption does not seem to have any appreciable effect. There is a slight dissolving during decantation which, while it adds to the recovery, makes the soluble loss somewhat greater than it would otherwise be.

The figures quoted below on chemical consumption and recovery refer to only two units of the Hollinger plant. The figures on these units are given because the other units of the mill share their feed with the original Moore filter plant, and likewise their barren solution, while for commercial reasons the two units in question have been given a separate solution system and separate precipitation presses. These two units are therefore the only ones upon which all the figures are available.

In comparing the results quoted, however, it should be borne in mind that the flow sheet has been modified in this plant somewhat, because of limitations of space, so that the overflow of $T_2$ instead of that of $T_1$ goes to precipitation. The effect of this is to raise the theoretical value of the overflow of the last tank 3 c. at 3 to 1 precipitation.

A statement of results follows:

Period covered, same as that for which costs were given—from Jan. 28, 1916, to Apr. 21, 1916.

[3] More recent measurements show that 20 tanks, of which four contain trays, consume 9.2 hp. motor input; while 24 three-throw pumps take 11.1 hp., or a total of 20.3 for the plant.

| | |
|---|---|
| Tons of ore treated | 38,885 |
| Value per ton of ore treated | $8.92 |
| Ratio of ore to solution precipitated | 100 to 285 |
| Tons solution precipitated | 110,604 |
| Strength of cyanide used | 0.9 lb. per ton, or 0.0045 per cent. |
| Cyanide added per ton of ore | 0.46 lb. |
| Difference between pulp feed and pulp discharge for first tank after agitators | 25 c. |
| Average moisture in tails | 45 per cent. |
| Average value of barren solution | 3.2 c. |
| Dissolved gold per ton of solution discharged | 11.71 c. |
| Dissolved gold per ton of ore discharged | 9.57 c. |

It is theoretically possible, taking into consideration the flow sheet, the grade of ore treated, the barren solution used and the thickness of pulp attained, to have reduced the overflow of the last tank to 7.6 c. leaving a difference of 4.1 c. to be accounted for by continued dissolving, adsorption, etc.

Viewed in one way it may be said that actual losses are 54 per cent. higher than theoretical, but where one is dealing with samples so easily affected by faulty manipulation and where any error except losses in assaying tends to raise the results, a check to 4 c. does not seem bad. The average loss would have been somewhat less if the occasional high results had been omitted, but this was not done.

From the foregoing, I believe one is warranted in concluding that a reasonably accurate forecast can be made of the results to be expected from a decantation plant and that these results may compare very favorably with the results obtained from filter plants.

In conclusion I would say that I am indebted to P. A. Robbins, Managing Director of the Hollinger company, for permission to quote results from the Hollinger plant.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1. 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# A Method for Distinguishing Sulphides from Oxides in the Metallography of Steel

BY GEORGE F. COMSTOCK, A. B., MET. E., NIAGARA FALLS, N. Y.

(New York Meeting, February, 1917.)

It seems a common opinion among metallographists that all light-gray inclusions seen with the microscope in polished sections of steel are manganese sulphide. Examples of this belief are continually appearing, as for instance in the paper by Dr. Henry Fay on manganese sulphide as a source of danger in steel rails,[1] and in Lieutenant-Commander Cook's paper on Metallography of Steel for U. S. Naval Ordnance.[2] Slate-

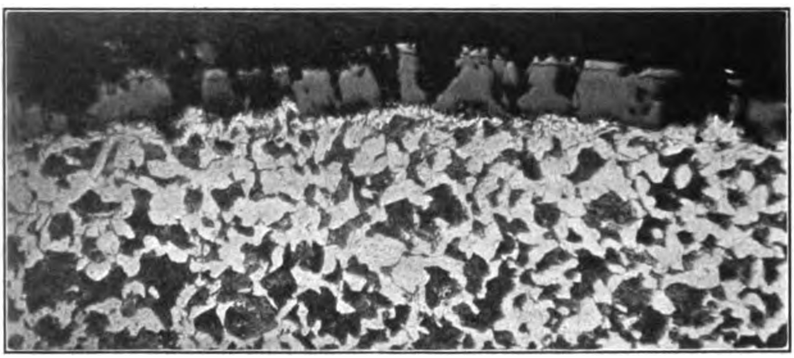


Fig. 1.—Transverse View of the Edge of a Small Hot-rolled Steel Rod. Etched with Picric Acid. × 200.

colored inclusions are considered to be silicates, and dove-gray inclusions, manganese sulphide. To show the danger in the latter unqualified assumption, it should be sufficient to examine the edge of any piece of steel that is covered with a fairly thick scale, and that has been polished in such a way as to prevent the scale from breaking away entirely below the polished surface. This can be done by protecting the scaled edge of

* Metallographist, Physical Testing Laboratory, Titanium Alloy Manufacturing Co.

[1] A Microscopic Investigation of Broken Steel Rails: Manganese Sulphide as a Source of Danger. *Proceedings of the American Society for Testing Materials*, vol. 8, p. 74 (1908).

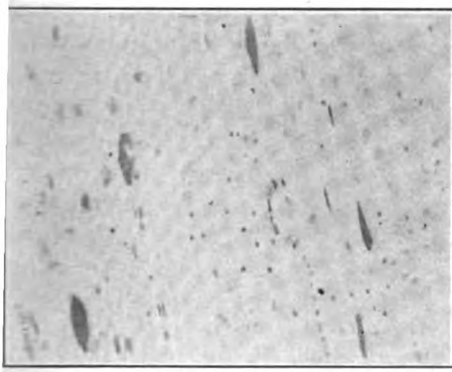
[2] *Trans.*, vol. 53, p. 238 (1915).

the specimen with some soft metal, or fusible metal applied molten, or ordinary red fiber such as electricians use for insulating. An examination of this scale, if the polishing has been done well, and the sample is preferably unetched, will show it to be light gray in appearance, and apparently "manganese sulphide." As a matter of fact, however, we know that scale is chiefly iron oxide, and contains sulphur only as an impurity. Its similarity in appearance to manganese sulphide is shown in Fig. 1, which is a transverse view of the edge of a small hot-rolled steel rod, magnified 200 diameters. The steel has been etched with picric acid, but the gray scale can be plainly seen at the edge of the metal. The very dark material beyond the scale is the red fiber in which this specimen was clamped to prevent the scale from falling away during polishing.

In the course of several years' work with the microscope on rail steel and other commercial steels the writer has not infrequently seen light-gray inclusions that, from evidence given by sulphur prints, or for other reasons, did not seem likely to be sulphides, and in some cases these were known without any doubt to be oxides, like the scale mentioned above. Finally, a case of some importance arose, where it was desired to know definitely whether certain light-gray inclusions were sulphides or not, and it was not possible to decide the question from their color alone. Reference to the standard textbooks on metallography did not help much, for the only real test described for distinguishing oxides from sulphides involved heating the polished sample in a current of carefully purified hydrogen, and this method was considered as too complicated and necessitating too much preparation and practice. Etching with weak organic acids was not found satisfactory, and neither was the method of using a drop of sulphuric acid and watching for bubbles of hydrogen sulphide gas. Tried with known sulphide inclusions, both these methods attacked the steel itself too strongly, or else, if the acids were weakened, had no effect either on steel or sulphides. It was finally decided to get together a few specimens containing sulphides, and others in which oxides were known to be present, and to try various solutions on them until one was found that would attack one type of inclusion and not the other.

This work had not progressed very far when it was remembered that boiling alkaline sodium picrate, which is used to darken cementite, having no effect on ferrite and very little on pearlite, was always found to attack the sulphide inclusions in a sample, leaving black pits instead of the light-gray spots seen before etching. Fig. 2, for instance, shows some sulphide inclusions in a polished section, unetched, cut from a segregated streak in a steel rail, where sulphur prints indicated the presence of high sulphur. This same spot etched with boiling alkaline sodium picrate is shown in Fig. 3, where the sulphides are seen to have been blackened by this reagent. The darkening seems to be due to an actual solution of the manganese sulphide in the etching liquid, so that instead of a smooth

polished surface, we now have a surface full of pits or hollows, where the sulphides were, and as these hollows do not reflect any light back into the microscope with vertical illumination, they appear black in the field of view. Figs. 4 and 5 show some other sulphide inclusions in a shell-steel billet, before and after etching, respectively. In this lower-carbon steel



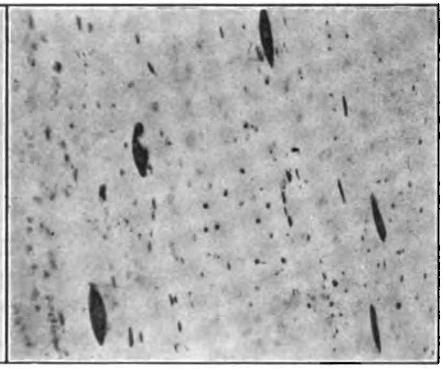


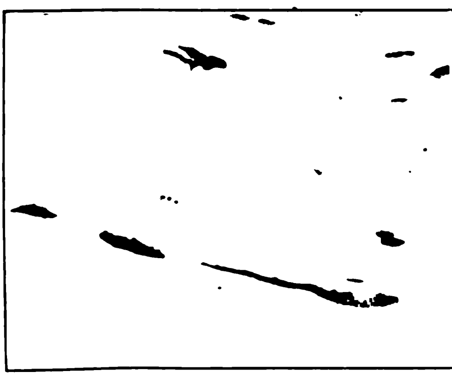
Fig. 2.—Unetched.

Fig. 3.—Etched with Boiling Alkaline Sodium Picrate.

Sulphide Inclusions in a Polished Section Cut from a Segregated Streak in a Steel Rod. × 130.

the ferrite-pearlite structure is brought out faintly by the etching, but not enough to interfere with the distinct blackening of the sulphides.

When this etching with boiling alkaline sodium picrate was tried on steel known to contain oxides, the problem of distinguishing oxides from



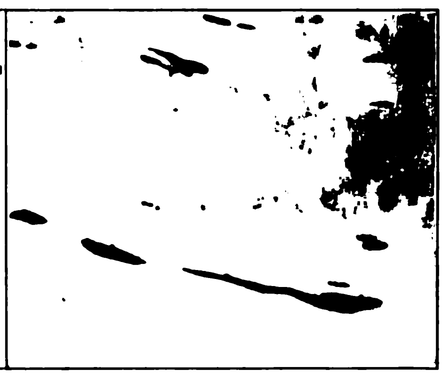


Fig. 4.—Unetched.

Fig. 5.—Etched.

Sulphide Inclusions in a Shell-Steel Billet. × 130.

sulphides was solved at once, for the oxide inclusions were found to be absolutely unattacked. Fig. 6 shows some oxides in a polished unetched sample of clean rail steel treated while molten in a crucible with nickel oxide. The steel undoubtedly reduced the nickel, forming iron oxide.

Fig. 7 shows this same spot after etching, and looks exactly like the unetched view. Fig. 8 shows a cross-section, unetched, of a seam in the base of a steel rail. These surface seams are always filled with scale, or oxide, although the light-gray color has sometimes led observers to believe that manganese sulphide was present. The fact that such seams give

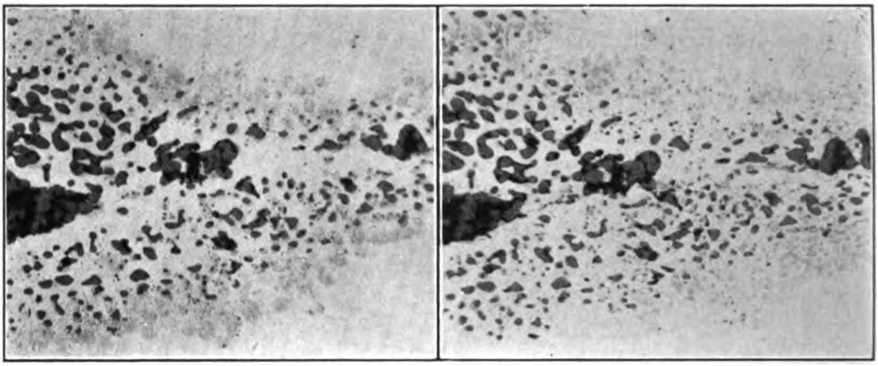


Fig. 6.—Unetched. Fig. 7.—Etched.
Iron Oxides in Rail Steel. × 130.

light streaks instead of black spots on sulphur prints and etched sections, however, shows that oxide and not sulphide is really present in them. This seam, etched with boiling alkaline sodium picrate, is shown in Fig. 9. The oxide is seen to be entirely unattacked.

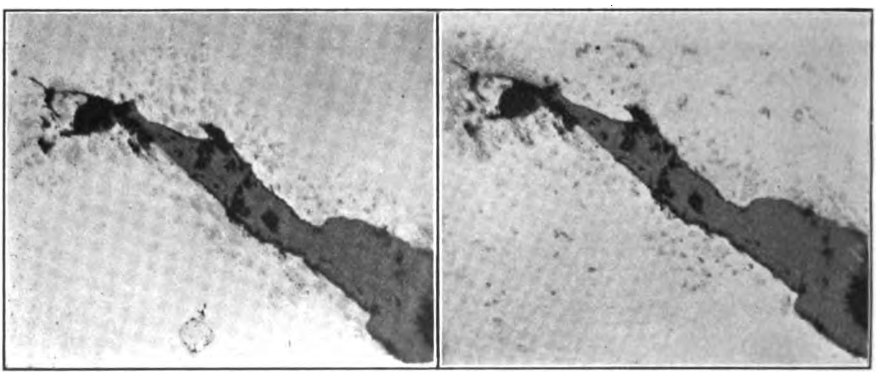
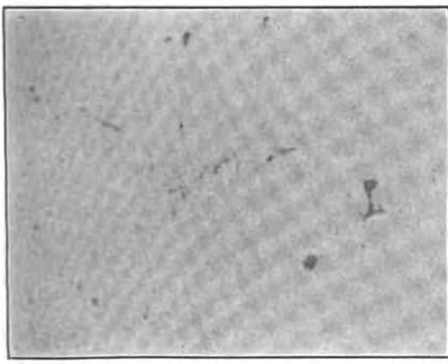


Fig. 8.—Unetched. Fig. 9.—Etched.
Cross-section of a Seam in the Base of a Steel Rail. × 130.

The application of this solution to etching all grades of steel for distinguishing between oxides and sulphides has not been described before, so far as the writer is aware. In ease of application and certainty of results it is much superior to other methods that have been suggested for this purpose. One advantage is that the solution used is one that every

metallographist should have on hand for etching cementite in high-carbon steel. Thus this same solution can be made to serve two purposes. It was first discovered by Kourbatoff, who recommended it for darkening cementite because it does not attack ferrite or pearlite. The writer makes up the solution according to the formula given in Sauveur's *Metallography and Heat Treatment of Iron and Steel*, the procedure being about as follows: 25 g. NaOH are dissolved in 60 to 70 c.c. of water, 2 g. of picric acid are added, and the solution is heated until the picric acid is dissolved, when the volume is brought up to 100 c.c. by adding more water. In using this solution to etch polished-steel specimens, it is brought to boiling in a beaker on a hot plate, the specimen is immersed in the solution, and boiling is continued for 10 min. Then the specimen is removed, washed, and dried. The polished and etched surface may be wiped dry with chamois skin without affecting the results. If the speci-



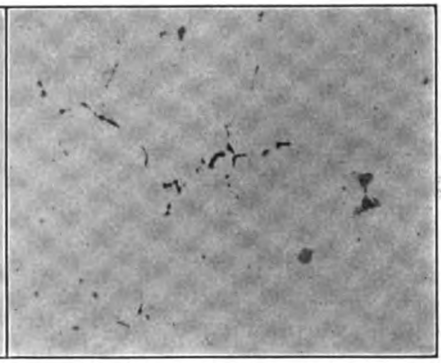


FIG. 10.—UNETCHED. FIG. 11.—ETCHED.
Polished Section of a Steel Casting which Cracked while Hot. × 130.

men contains cracks, this wiping is necessary, and repeated washing and drying may also be necessary, to remove the last traces of the sodium picrate from the surface to be examined. The solution may be used many times, but as the boiling makes it more concentrated water should be added occasionally to keep the volume about 100 c.c.

Before closing, some illustrations will be given of the practical use of this method for distinguishing oxides from sulphides. Fig. 10 shows a polished section of a steel casting which cracked while hot and in which the presence of oxides might therefore be expected. An experienced eye could perhaps tell which of these inclusions are sulphides and which are oxides, but the distinction at best would not be convincing. After etching by the method described above, however, the difference between the two kinds of inclusions becomes evident, as shown in Fig. 11. The arrangement of these sulphides is noteworthy, as a network is strongly suggested, and by this method of etching it can easily be shown that the

small inclusions often found in a network arrangement in the ferrite of well-deoxidized steel castings are merely finely divided sulphides. An instance of such inclusions was described by Dr. Henry Fay in a paper on Some Causes of Failures in Metals,[8] and well illustrated by his Figs. 16 and 17 on page 450. These are said to be "slag" by Dr. Fay, but their appearance is exactly similar to numerous occurrences of sulphides which have come to the writer's attention.

A large complex slag inclusion, also in a cracked steel casting, is shown unetched in Fig. 12. The light-colored rough-looking part of this inclusion at the center of the field of view was bright yellow, and hence considered to be sulphide of iron. The other light-gray portions, including the spots in the dark-gray part, were supposed at first to be manganese sulphide, and the darker part, manganese silicate. But when etched with boiling alkaline sodium picrate, only the yellow constituent was

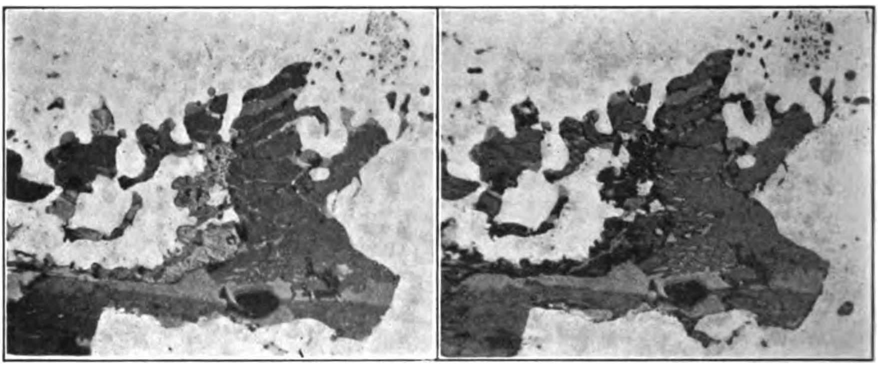


Fig. 12.—Unetched. Fig. 13.—Etched.
Large Complex Slag Inclusion in a Cracked Steel Casting. × 130.

attacked, showing that it alone was sulphide, and the light-gray constituent was oxide instead of manganese sulphide. The appearance after etching is shown in Fig. 13.

A section cut from a bar of "ingot iron" that was red-short and cracked in rolling is shown in Fig. 14, taken before etching. Most of the inclusions found here were light gray in color, but many of them had yellow parts attached to them at one or both ends. The yellow substance, which is very rarely seen in steel, was taken to be sulphide of iron, and the gray inclusions were at first supposed to be manganese sulphide. But the manganese content of this ingot iron was found to be very low, and the sulphur also was not high enough to allow the presence of so many sulphides. Consequently, it was doubtful whether these gray inclusions were sulphides or oxides, until the sample was etched with the boiling alkaline sodium picrate solution, when it was apparent at once

[8] *Proceedings of the American Society for Testing Materials*, vol. 11, p. 439 (1911).

that they were oxides, and only the yellow substance was sulphide. The inclusions of Fig. 14 are shown after etching in Fig. 15, where the distinction between oxide and sulphide is very plain. One of the largest particles of the yellow constituent was noticed, before etching, to have

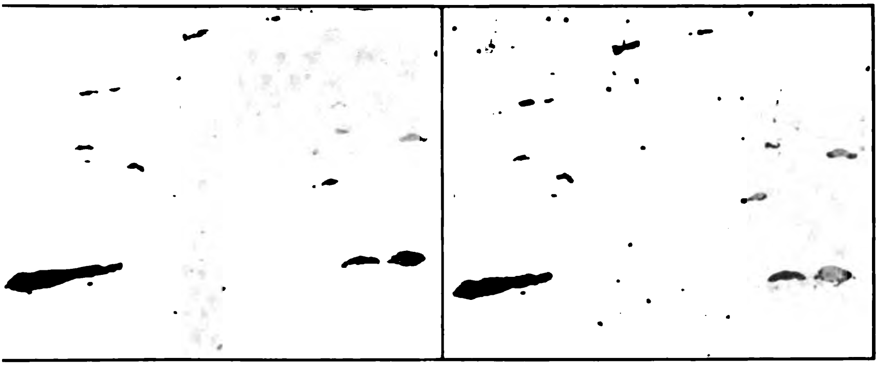


FIG. 14.—UNETCHED. FIG. 15.—ETCHED.
Section Cut from a Bar of "Ingot Iron" that was Red-short and Cracked in Rolling. × 130.

a spotted appearance, and when examined under high magnification it was seen to be full of very fine gray spots. This is evidently a eutectic of iron oxide and iron sulphide. It is shown in Fig. 16, magnified 570

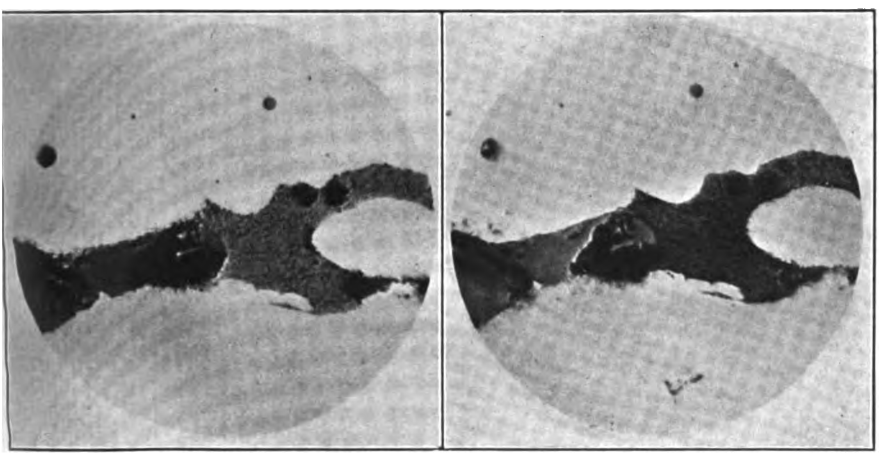


FIG. 16.—UNETCHED. FIG. 17.—ETCHED.
One of the Large Inclusions in Fig. 14. × 570.

diameters, the large unspotted dark-gray areas, with the black pits due to fracture in polishing, being oxide, and the spotted area being the eutectic. The oxide is here seen to be darker than the sulphide. Fig. 17 shows this same spot after a short etching with boiling alkaline sodium

picrate, and it is apparent that the eutectic, but not the oxide, has been attacked, for the former is now darker than the latter, and its structure is blurred and indistinct.

These notes are presented in the hope that the simple method here described will appeal to metallographists in general as being worthy of trial in cases where the identity of light-gray inclusions in steel is in question, so that in future sulphides need not be called "slag" and the error of calling oxide or scale inclusions "manganese sulphide" merely because their color is light gray may also be avoided.

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# On Grain Growth

BY HENRY M. HOWE, LL.D.,* BEDFORD HILLS, N. Y.

(New York Meeting, February, 1917)

THE brilliant and very original matter in Professor Jeffries' discussion† should rank not only as an independent paper, but as a most important one. In particular, the explanation which it gives of the remarkable Sauveur phenomenon, over which so many of us have puzzled long and hard, is so clear, complete, and cogent as to go far toward establishing Professor Jeffries' hypothesis. The very features of Professor Sauveur's photographs which fit this hypothesis so exactly are reproduced in Chappell's photographs of steel coarsened under somewhat different conditions.[1]

The ideas developed by these discoveries are so novel that convenience of language seems to call for new conventional terms for expressing them. In what follows here I propose certain terms for this end, for criticism and improvement, and I attempt to generalize from Professor Jeffries' hypotheses.

Generally speaking, a given set of grains which undergo no growth in the cold will, when the temperature is raised progressively, start to grow on reaching a certain temperature, and will continue growing at all higher temperatures.

The "germinative" temperature is that at which growth begins, quite as the conditions which cause a seed to grow are "germinative." We have then an *inert temperature range* below the germinative temperature, and a *growth range* above it, the germinative temperature being the limit between these two ranges. So, too, a germinant grain is one which is at its germinative temperature, and hence beginning to grow, while inert grains are those below their own germinative temperature, and the term "growing grains" includes those which are germinant and those which are above their germinative temperature. These last could be called "supergerminant" if the distinction should be needed.

"Germinative temperature" seems better than "critical temperature

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* Emeritus Professor of Metallurgy, Columbia University.

† Of the paper of C. H. Mathewson and Arthur Phillips, *Bulletin* No. 113, p. 987 (May, 1916).

[1] *Journal of the Iron and Steel Institute*, 1914, Pt. I, Plates 46 and 47, following p. 496.

for grain growth" and "equiaxing temperature," first because "critical" has been so much overworked and suggests neither critically fast nor critically slow growth, and second because a germinative temperature is what the hypothesis postulates, and because the further hypothesis that this coincides with the equiaxing temperature is only an inference and for the present purpose an unnecessary one. Indeed, Professor Jeffries[2] informs me that he has found cases in which the germinative temperature is far above that of equiaxing.

When only a limited part of the metal is at its germinative temperature, this becomes a "fast-growth" temperature, because the germinant grains feed unopposed on the inert ones against which they abut.

The late results of White and Wood[3] tend to complicate our ideas, for they indicate that germinance depends not alone on temperature but also on time, so that germination may occur on a long exposure to a temperature at which none can be detected after a shorter exposure. This indeed agrees better with our general knowledge of "aging." In this view inertness is not absolute but only relative.

*Grain growth*, that is the absorption of certain grains by certain others, is favored, and the *germinative temperature* is consequently lowered, (*A*) by grain fineness, (*B*) by grain-size contrast, and (*C*) by prior plastic deformation.

(*A*) *Grain fineness* favors grain growth because it implies extensive grain boundary surface at which alone can absorption occur. To vary the angle slightly, a growth which doubles the size of extremely fine grains means only that each alternate very fine grain absorbs its adjoining still smaller neighbor, a much faster process than the absorption of one enormous grain by its even greater neighbor, faster whether we hold that the whole of a given grain is absorbed simultaneously, or, as seems more probable, that the swallowing is gradual, as for instance by some kind of boundary migration.

(*B*) *Grain-size contrast*, that is the difference in size between adjoining grains, favors grain growth because the absorptive and the resistant powers of a grain increase with its size, so that, when there is great grain-size contrast, growth is favored jointly by the great absorptive power of the larger grains and by the weak resistance of the smaller ones.

(*C*) *Plastic deformation* or *overstrain* favors grain growth, perhaps by leading to fineness of grain and to grain-size contrast. The fact that the germinative temperature is the lower the greater the deformation has been tallies with the former explanation, because increasing deformation would naturally lead to finer grain size. It may also increase the grain-size contrast. But the fact that grain growth will not occur at all unless

[2] Private communication, June 24, 1916.

[3] Recrystallization as a Factor in the Failure of Boiler Tubes, *Proceedings of the American Society for Testing Materials*, xvi, 1916, to appear.

there has been plastic deformation suggests that some additional cause applies here.

(*D*) Grain growth occurs the more readily the higher the temperature, and the greater consequently the mobility.

Grain growth is opposed, and consequently the germinative temperature is raised, by:

(*E*) Obstructions, such as cementite, slag, and other sonims in iron, and thorium oxide in tungsten; and by

(*F*) Grain-size equalization, or the decrease of grain size contrast.

*The rate of growth* increases:

(*G*) With the difference in absorptive power between adjoining grains, whether this is due to a difference in their:

(*H*) Temperature; or in their

(*J*) Size, that is to grain-size contrast; or in their

(*K*) Prior plastic deformation. It further increases:

(*L*) With the temperature;

(*M*) With the initial grain fineness; and

(*N*) With the prior plastic deformation.

(*P*) The rate of growth decreases as the grain size increases, because this increase implies a corresponding decrease in the extent of grain boundary surface.

The very fastest growth seems, from Professor Jeffries' results, to be due to (*H*), which is thus an extremely strong influence. From the fact that low temperature, if it does not arrest growth, at least retards it very greatly, we must admit that (*L*), too, has great influence. Further light on the strength of (*J*), grain-size contrast, is needed. Whether exact identity of grain size would arrest growth completely if the temperature and the degree of plastic deformation were identical, seems hardly capable of proof, because that identity can never be attained.

The reported cases of grain-size equilibrium, in which growth is reported to cease, seem to me more easily explained as resulting from (*P*) the progressive decrease in the extent of grain boundary, and from (*J*) a progressive lessening of the grain-size contrast as the smaller grains are eliminated successively, and a progressively smaller number of larger ones remains. From this point of view the apparent arrests of grain growth and the cases of so-called "grain-size equilibrium" are to be regarded as only retardations, however marked these may be.

Grain growth may be divided into the *normal* and the *exaggerated*. In most industrial heatings there is normal grain growth, that is to say a coarsening which is leisurely unless the temperature approaches the melting point. In strong contrast to this is the exaggerated growth noted by Stead, Carpenter, Sauveur, Ruder, Chappell, and Jeffries, which may be extremely rapid and may lead to extreme coarseness.

If we leave out of consideration temperatures approaching the melting

point, exaggerated grain growth seems to occur only at the contact of germinant and inert grains. Thus in Jeffries' experiments accelerated coarsening occurs only and always in case the mass is held under a current of such a strength as to bring only part of it to the germinative temperature, the outer and colder parts being inert because below this temperature. Here growth occurs at the germinant internal surface, the germinant grains feeding on the colder inert ones next outside them, thus increasing in size and thus creating a grain-size contrast, which according to Principle (*A*), enables them to feed on the still more inert grains still nearer the outside, and also to begin feeding on the neighboring grains nearer the axis, which, though growing because they are in the growth range of temperature, are growing more slowly because the grains on which they are feeding are themselves active, and thus offer more resistance to absorption than the inert grains against which the grains in the germinant layer abut.

Turning from this case, in which the contact of germinant and inert grains is brought about by the existence of a progressive change in temperature from shell to axis, we have the Sauveur cases in which it is due to a variation in the degree of plastic deformation which the different layers have undergone. When a piece which has been bent or has received a Brinell impression is next heated to a temperature which is at or above the germinative temperature of the most deformed layers, the germinative temperature of some one layer, or in case of a bent bar of some two layers about equidistant from the neutral axis, will equal this existing temperature. This layer and all the more deformed ones will begin growing, but the germinant layer will grow faster than the others, because it feeds on the inert because less deformed layers beside it.

A probably more accurate statement is that exaggerated growth occurs where there is a sharp increase of inertness from layer to layer. Here the grains of a certain layer are so much more active than the more inert ones in the adjoining layer against which they abut that they have a marked advantage as regards germinative power. The lead which this advantage gives them increases progressively as growth proceeds.

The experience of the wire-drawers, that though annealing often causes marked coarsening after the slight deformation of the wire-straightening process, it does not after the extreme deformation of industrial wire-drawing, conforms exactly with Professor Jeffries' hypothesis.

Here we may suppose that, after wire-drawing, the temperature is so low that in the early stages of the heating up it is passed through too rapidly to permit appreciable grain growth at this relatively low temperature, whereas after the slight deformation of straightening the germinative temperature may well lie at the relatively high temperature of annealing, at which first the sojourn is long, and second the rapidity of growth is relatively great. The reason why extremely slight deforma-

tions do not lead to coarsening on annealing is that the position of the germinative temperature which they determine is so high that it is not reached in annealing, if indeed it does not approach closely the melting point of the metal.

When a principle explains so much, the existence of certain things which it does not explain argues, not against its truth, but for the existence of additional principles not yet discovered. The two difficulties which I now raise should be taken in this sense.

My first difficulty concerns the Sauveur specimen of Professor Jeffries' Fig. 2, Professor Sauveur's Fig. 30. In this long stay at the germinative temperature of the outer parts of the coarsened bands, the grains in these bands should have fed to an important degree on the inert fine-grained core, and their resultant growth, proceeding as it should at right angles to the contact between the fast-growing band and the central core, should have developed columnar grains. Yet in fact the grains look equiaxed. On the other hand, such a columnar arrangement is prominent in the Chappell specimen just referred to.

My second difficulty is that this five-banded arrangement, with two outer bands of intermediate coarseness and a central fine one, should be the rule and not the exception, and should not have passed undetected the vigilant scrutiny of so many observers, for the coarsening is easily visible to the naked eye, and should not have had to wait till 1912 for its discovery. Thus the Sauveur coarsening is lacking in so large a proportion of cases as to suggest that it is held in check by one or more additional principles. I suggest the obstructing effect of foreign bodies as a possible explanation of one set of cases in which coarsening is lacking, a set illustrated by Professor Sauveur's finding that marked coarsening does not occur unless the carbon content is between 0.04 and 0.12, and that it occurs most readily when the carbon content is between 0.05 and 0.07 per cent.[4]

The presence of 0.12 per cent. of carbon, implying that of 13 per cent. of pearlite, might well offer enough mechanical obstruction to prevent coarsening, for each pearlite mass would act as a wholly foreign body to bar the union of the two grains between which it lies. Even as little as 0.07 per cent. of carbon, implying the presence of about 8 per cent. of pearlite, might have an important obstructive effect. On the other hand, when the carbon content is less than 0.04 or 0.05 per cent., there may well be enough iron oxide or minute gas bubbles to offer serious obstruction. Here we should remember Benedicks'[5] discovery that though the specific volume of his steels was strictly proportional to the carbon content between the limits of 0.45 and 1.20 per cent., that of his iron with 0.08 per

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[4] H. M. Howe: *The Metallography of Steel and Cast Iron*, p. 363 (1916).

[5] Carl Benedicks: Recherches physiques et physico-chimiques sur l'acier au carbone, pp. 2, 30 and 35, Paris, 1904.

cent. of carbon and 0.03 per cent. of silicon was abnormally great. It is true that this last was wrought iron, and hence that its specific volume would be made abnormally great by the presence of slag. But with only 0.03 per cent. of silicon the slag content should be extremely small, probably not more than 0.20 per cent., and hence insufficient to explain the surprisingly small density, which I refer therefore to the presence of minute gas cavities so often present when the carbon content is so small.

This brings us to a hitherto unsuspected element of fitness for man's needs which iron has, that the coarsening of its ferrite is limited to temperatures so far below the melting point as to give great viscosity, and hence to restrict the coarsening tendency. On rising past *Ac*1, the 8 per cent. of pearlite which we have just considered turns into 8 per cent. of austenite, and this increases very rapidly in quantity, till at *Ac*3 it forms the whole. As it increases in quantity it forms a more and more complete obstacle to the coarsening of the ferrite with which it is now associated, so that in fact the coarsening of ferrite, even in lower-carbon steels, is probably confined to temperatures below 800°, and thus many hundred degrees below the solidus, and hence corresponding to great viscosity.

Moreover, we may suspect that at *Ac*2 the change from alpha to beta ferrite, which Burgess and Scott[6] seem now to have established clearly, will break up any coarsening which has occurred in the alpha ferrite below this temperature. This, if true, would confine the coarsening range to below 768°.

These considerations lead us to expect to find iron less subject to coarsening than most metals.

It is true that Professor Jeffries' hypothesis explains satisfactorily and without needing this obstruction principle certain other cases of unexpected failure to coarsen, for instance that of my tapered bar of steel of about 0.01 per cent. of carbon from the American Rolling Mill Co. This, after straining with tensile stresses varying from 40,640 to 49,000 lb. per square inch, failed to coarsen noticeably in a 22-hr. exposure to 680°. Because the yield point of this steel is only about 20,000 lb. per square inch, even the least of my stresses may well have caused a degree of deformation so great that the corresponding germinative temperature was far below 680°, so that, in heating up to 680°, the germinative temperature of each of the various parts was passed through too rapidly to lead to material coarsening.

Yet this explanation hardly applies to Professor Sauveur's failure to coarsen his bent specimen in a 7-hr. exposure to 650° in case the carbon content is less than 0.04 per cent., for here there should be layers near the neutral axis in which the plastic deformation should vary progressively from nil to far above that for which the germinative temperature is 650°. Somewhere in this progressive series there should be a layer with

[6] *Comptes Rendus*, vol. 163, No. 2, p. 30 (July 10, 1916).

the deformation for which 650° is the germinative temperature, and this layer should have coarsened at the expense of its less strained neighbors nearer the neutral axis. But here the failure to coarsen may be explained by the suspected presence of gas bubbles, to which such low-carbon steel is so subject.

The important discovery that, though long heating at a low temperature might cause great coarsening, yet a quick passage up to a high temperature might prevent it, was communicated to me by Professor Jeffries on July 1, 1915, and is recorded, together with most of the information on which his present remarks are based, in a report of his which I have read, dated May 14, 1915.

Turning to Professor Jeffries' contention[7] that uniformity of etching tint is not valid evidence of uniformity of orientation within any given grain, though he would be right if he confined himself to saying that the retention of a uniform etching tint by a single grain is not proof positive of uniformity of orientation throughout, yet when each of many grains in a single field retains a uniform etching tint which differs from that of its neighbors, the inference is irresistible that there is a far greater approach to uniformity of orientation within each grain than between adjoining grains. The fact that certain specific rotations need not alter the brightness of the reflection is beside the mark. What is important is that the great majority of random rotations will alter it.

The irregularity of the slip bands and X bands, in the absence of corresponding irregularity of etching tint, is explained more readily as representing the effort of slip to integrate into a line parallel to the major stress, by stepping back and forth from one to another of two or more sets of conjugate slip planes, than as representing irregularity of orientation. The fact that this irregularity of the slip bands, from being negligible in the first stage of deformation, increases as deformation proceeds, accords with my interpretation, for whereas increasing deformation does not cause any corresponding irregularity of etching tint, it ought to facilitate this change from set to set of planes, by increasing both the frequency and the thickness of the amorphous sheaths surrounding the residual crystalline blocklets, of which the crystallographic slip planes are the paths of that slip. For whatever tendency slip has to persist along a given plane as long as it is passing through crystalline matter is clearly lessened when that slip comes to the amorphous sheath which parts each crystalline grain-fragment from the next.[8] The regularity of the etching pits in each grain even after extreme plastic deformation and the sharp change in their direction from grain to grain are very cogent evidence of the retention of substantial uniformity of orientation during deformation.[9]

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[7] *Bulletin* No. 113, p. 987 (May, 1916).
[8] Howe: *The Metallography of Steel and Cast Iron*, pp. 316, 483.
[9] *Op. cit.*, pp. 287, 485.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

**DISCUSSION OF THIS PAPER IS INVITED.** It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Geology of the Bawdwin* Mines, Burma, Asia

BY M. H. LOVEMAN, BAWDWIN, BURMA, ASIA

(New York Meeting, February, 1917)

THE orebody described below has been rediscovered and developed within the last 3 years. It has, however, been known and worked by the Chinese for hundreds of years. When assay values and size are considered, it contains what is probably one of the largest single bodies of zinc and lead sulphides yet found. The tonnages developed at other places, as at Broken Hill, are much greater but the average zinc and lead sulphide contents are considerably lower than at Bawdwin.

Bawdwin is situated in the semi-independent State of Tawng Peng, one of the numerous units which make up the Northern Shan States. These States are under British rule and are generally considered as a portion of Burma. They are governed as a separate administrative unit, however, under the Lieutenant Governor of Burma. Each State is under its own chief, or Sawbwa, who has considerable power over his own subjects.

Bawdwin is approximately in latitude N 23°6′, longitude E 97°20′, about 50 miles south of the nearest point on the Chinese border and 450 miles north of Rangoon (Fig. 1). The period at which mining was first begun at Bawdwin is unknown. The most reliable records have been obtained by deciphering local Chinese inscriptions and from these it is concluded that the work dates back to at least the beginning of the 15th century. The first undoubted reference to the mines by a European is by Symes in 1795. Crawford in 1827 estimated the output of silver from the mines at $600,000 annually. At about the middle of the 19th century the rebellion of the Chinese Mohammedans in the neighboring Chinese province of Yunnan so weakened the power of the Chinese that the incursions of hostile tribes (Kachins) made work extremely dangerous. The mines were finally abandoned by the Chinese about 1868. The closing down of the mines was probably not entirely due to outside causes but may have been influenced somewhat by the increasing difficulty of handling the water, as the workings gradually descended below the water level of the district. Several attempts were made by the Burmese kings, Mindon Min and Thebaw, to resume operations, but because of a

* Bawdwin—from Burmese: baw = silver and dwin = well or mine.

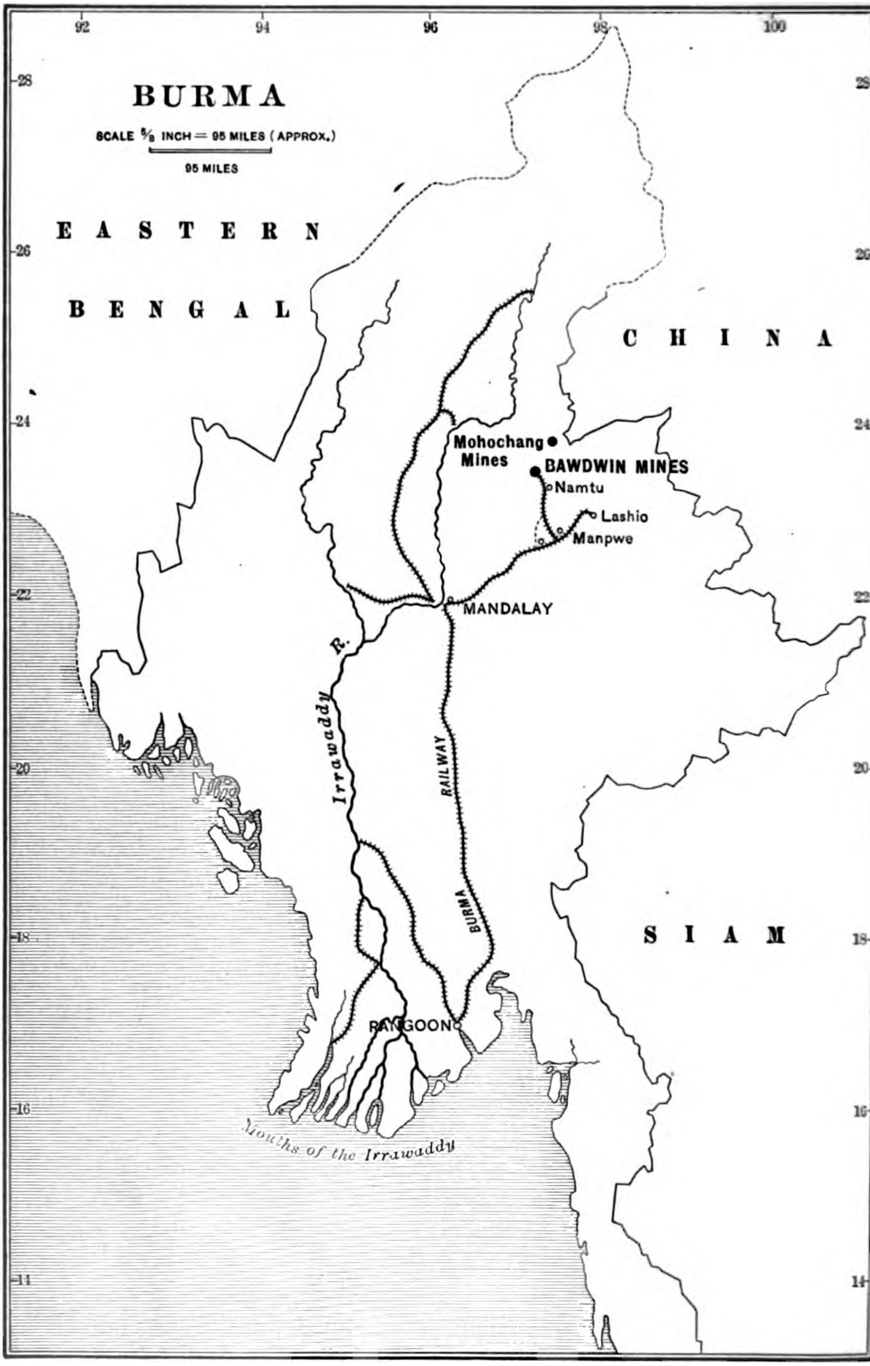



FIG. 1.—MAP OF BURMA SHOWING THE LOCATION OF BAWDWIN MINES.

combination of diseases and lack of mining skill such efforts proved unsuccessful.

During the course of the long occupation by the Chinese an immense amount of work was done and the amount of rock removed would certainly total well over 1,000,000 tons. The attention of Europeans was first attracted in 1901 by the extensive slag dumps which lined the valley for a considerable distance north and south of the mine workings. During the removal of the slag, prospecting work was started on the orebodies themselves. The multitude of Chinese workings proved a drawback rather than a help as they were the cause of impounding great bodies of standing water which made the reopening of the mine a dangerous undertaking. It is only within the last 3 years that this difficulty has been overcome and the orebodies have been developed.

## *Topography*

The State of Tawng Peng is extremely rugged and covered with a fairly dense jungle over its entire area. The present drainage has now produced the maximum of relief and further erosion without elevation of the land will tend toward a reduction in the inequalities. The ruggedness is intensified by the folding and possibly to some extent by faulting. The only cleared spots are the paddy fields along some of the stream bottoms and occasional small hillside farms. The inhabitants, Shans, Palaungs and Kachins, live in small villages of from 10 or 20 people to a few hundred. There is very little commingling of the different races. The Shan settlements are in the river bottoms; the Palaungs and the Kachins generally live well up the hillsides or along the high ridges. All communication is by mule tracks and footpaths which generally run along the ridges and descend to the valleys only for short stretches. All travel is done either on foot or by mules. Small Shan ponies are occasionally used.

The seasons are divided into a rainy and a dry period. The rains extend from the end of May to the beginning of November; the dry season embraces the balance of the year. The amount of rain is not excessive. It averages approximately 65 in. The months just preceding the beginning of the rains are hot and sultry; but in January and February the temperature drops to a point where fires become almost a necessity.

## *General Geology*

The rocks of Tawng Peng are entirely of the pre-Cambrian and Paleozoic eras. The youngest strata belong to the Carboniferous period. The western and central portions of the State are mainly composed of mica schists and of severely folded, unfossiliferous shales and quartzites. There are also some fairly extensive granite intrusions of an undetermined age. These rocks are followed unconformably by Ordovi-

cian sediments and are themselves therefore either Cambrian or pre-Cambrian in age. This series of rocks formed an ancient land surface along the eastern shore of which Paleozoic sediments were laid down. The sequence from the unfossiliferous basement to the Ordovician series is not an invariable one. Silurian sandstones are often found directly upon the unfossiliferous series. Their presence in this relation is accounted for either by faulting or overlap. At a few points along the borders of the pre-Ordovician rocks are found small exposures of rhyolite, tuffs, breccias and flows. The most important of these rhyolite bodies is that at Bawdwin. Overlying the rhyolites at Bawdwin, and probably conformably, is a great thickness of unfossiliferous sandstones and shales. For the present their position must be placed after the pre-Ordovician shales and quartzites and before the conformably overlying fossiliferous Ordovician marls and sandstones.

### *Topography and Surface Geology of Bawdwin*

The topography of the immediate vicinity of the mine is similar to the general description given above for the State of Tawng Peng; extremely rugged, narrow valleys with precipitous inclosing hills. The hills rise in instances to more than 2,000 ft. above the valley bottoms. One marked distinction, however, between the immediate vicinity of Bawdwin and the remainder of the surrounding country is the fact that the Bawdwin hills are absolutely devoid of the jungle covering so characteristic of all the rest of the country (Figs. 2 and 3). The hills are, however, covered by long grass, 3 to 6 ft. high, which is largely burnt off toward the end of the dry season and springs up again during the rains. The explanation of the absence of jungle is the obvious one of removal by the Chinese for fuel for smelting and domestic purposes. The demand for fuel was evidently so great that the hills were stripped clean of all trees and 50 years' desertion has been able to effect only slight reforestation. The greatest extension of the cleared area is to the northwest, evidently due to the fact that the carry from that direction to the site of smelting operations was a downhill one. The hills surrounding Bawdwin to the west, north and northeast are crowned by an elaborate network of earth fortifications. These fortifications were undoubtedly constructed by the Chinese for the protection of the mine but at what period and against whom is not known.

The rocks at Bawdwin fall into two divisions: first, the volcanic rocks, rhyolite tuffs, breccias and flows; and, second, the overlying and underlying unfossiliferous sediments, consisting of sandstone, shale, and occasional conglomerate beds, the last only in the overlying sediments. No limestone has been found in these sediments.

On the west section of the 653-ft. level, the deepest point in the mine,

sediments are present. These are presumably the underlying sediments, and appear to be unconformably overlain by the rhyolite tuff, although

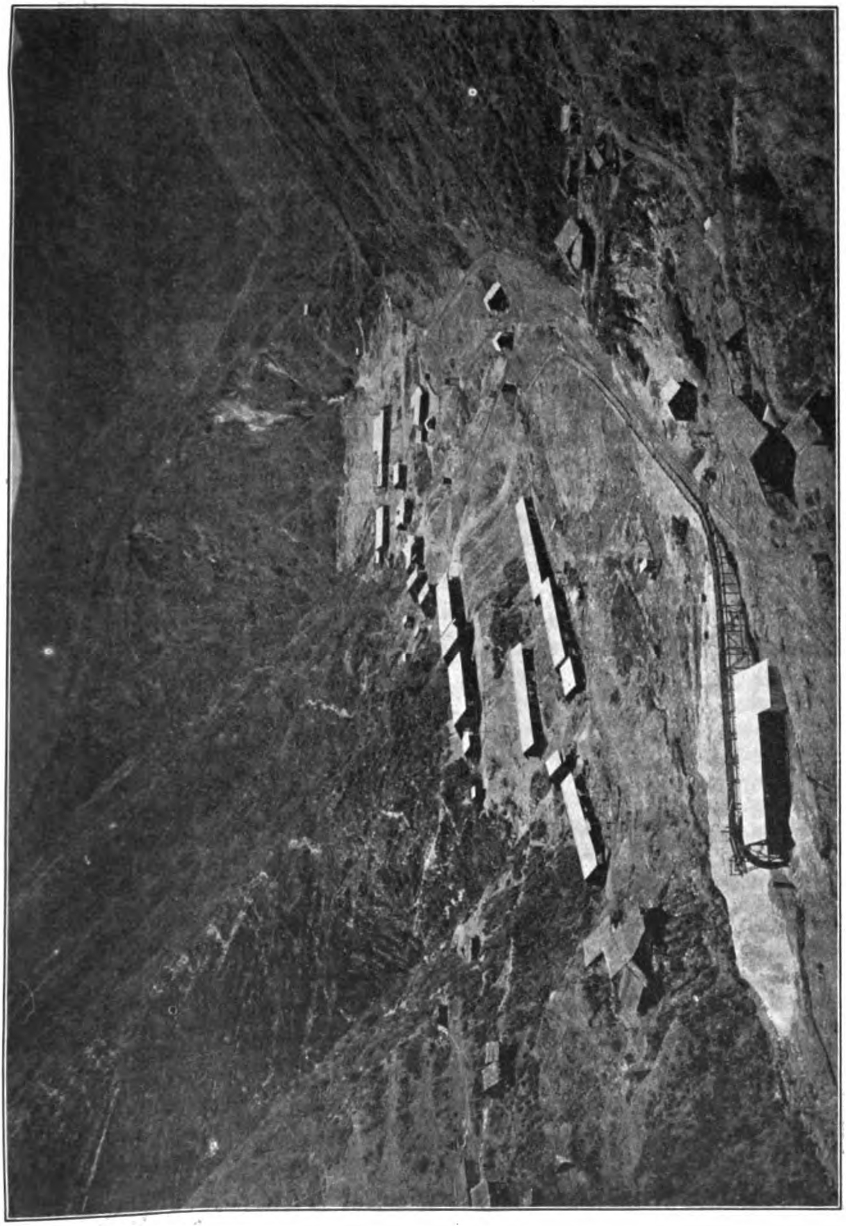


FIG. 2.—VIEW OF BAWDWIN SHOWING RUGGED CHARACTER OF THE TOPOGRAPHY.

this cannot be definitely stated, as the only point where the contact has been observed is a fault contact. The rocks are sandstones (practi-

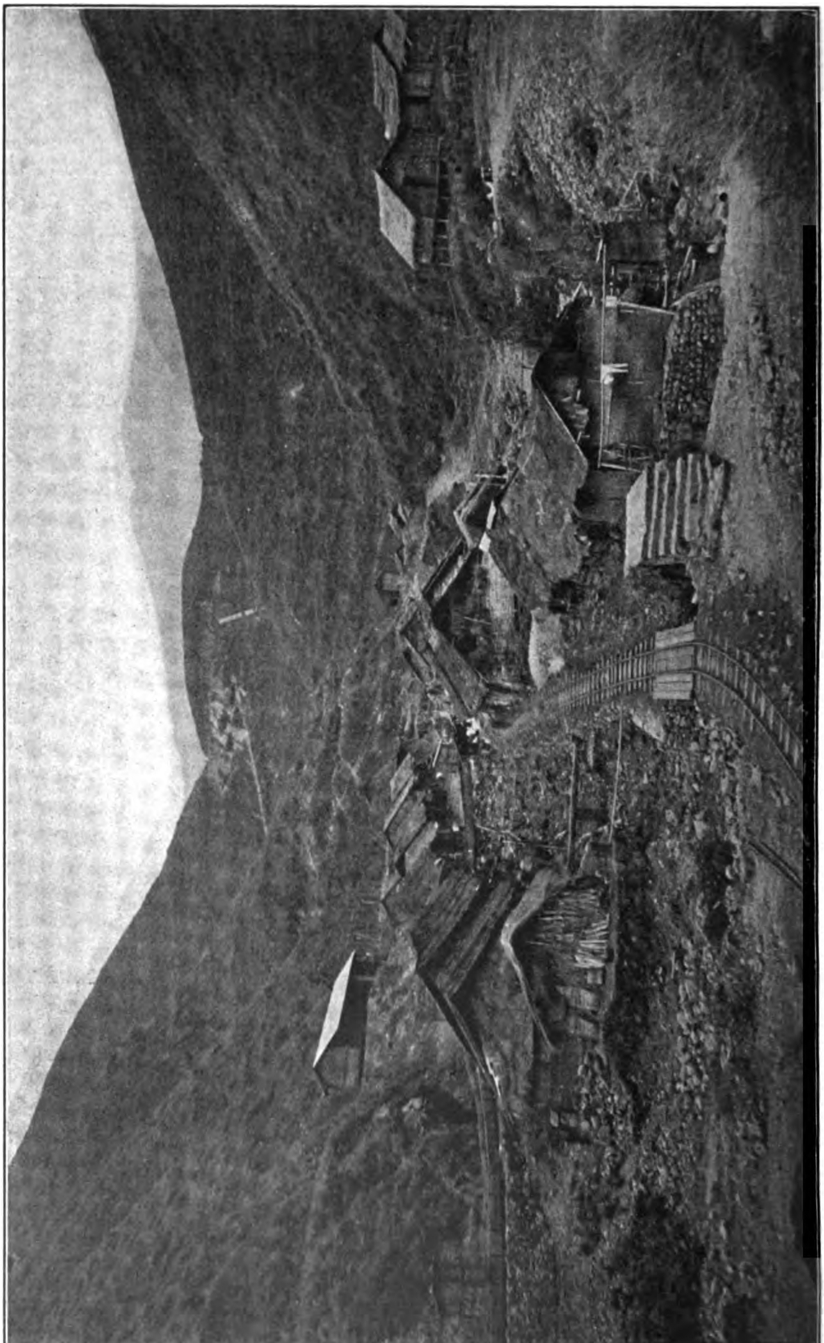


FIG. 3.—CHINESE MINERS' VILLAGE WITH AN ANCIENT CHINESE STONE ARCH BRIDGE IN THE FOREGROUND.

cally quartzites) and shales. The sandstones are red to gray with small irregular quartz grains cemented together by sericite. The shales are light gray to purplish in color, compact with the original lamination largely obscured, although the bedding on a large scale is still observable in both the sandstone and shale.

The rhyolite in its various phases forms a rough band running northwest and southeast, but of irregular outline with numerous offshots which

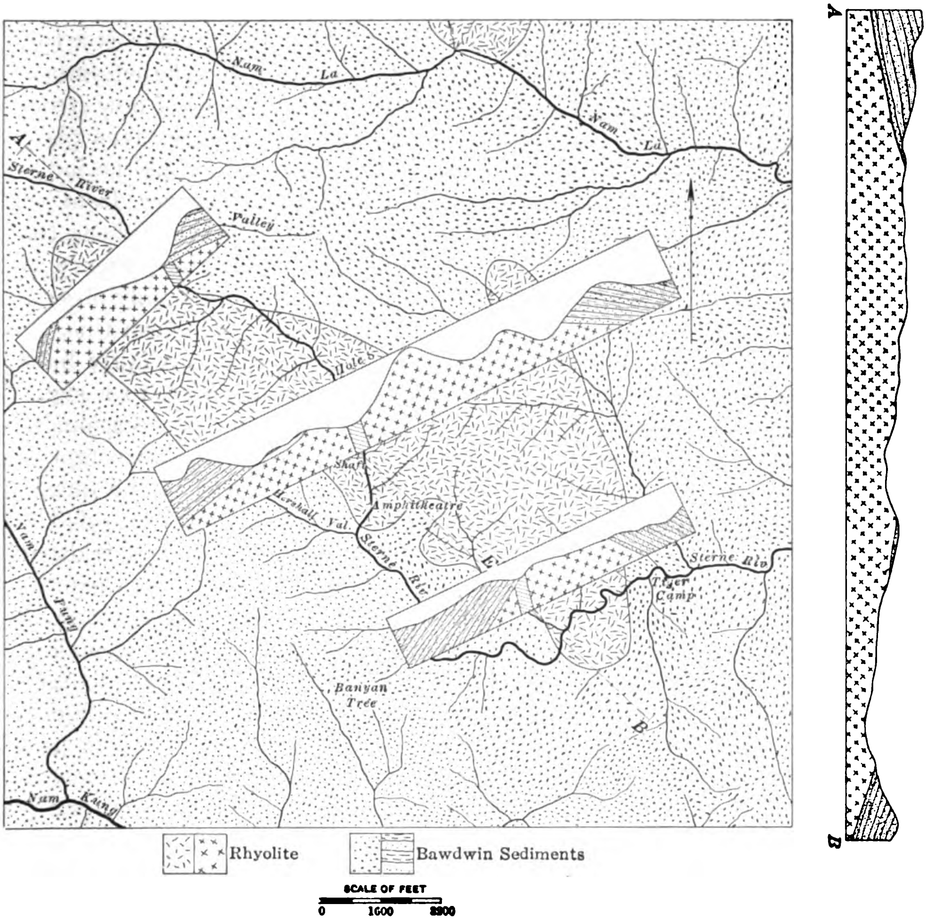



FIG. 4.—GEOLOGICAL SURFACE MAP WITH CROSS-SECTIONS. BURMA MINES, LTD.

are sometimes directly connected with the main mass and sometimes separated in outcrop by intervening, overlying sediments. The exposure of rhyolite does not represent its actual shape or extent but simply shows that portion from which the overlying sediments have been eroded. The general direction of the rhyolite marks the axis of an anticline, from the crest and flanks of which the sediments have been removed, exposing

the underlying eruptive. The anticline plunges both to the north and south with a resultant dome formation, having the main axis of the dome, however, running northwest-southeast (Fig. 4). This anticline is the principal one of a series of folds which parallel the rhyolite band on the west. The folding is further complicated by the great amount of faulting which accompanied it. Almost the entire exposure of rhyolite is a tuff with local areas where the included fragments attain sufficient size to justify the term rhyolite breccia for the rock at that point. Over a great portion of the area the pronouncement that the rhyolite is a tuff depends entirely on microscopic examination and even then the determination is not an absolute one. A rhyolite flow with all signs of flow structure obliterated by later changes, such as could easily have taken place in a rock of such great age as the one under discussion, would be indistinguishable from the tuff. The gradual gradation, however, from an undoubted coarse-grained tuff to the fine-grained type strengthens the belief in the tuff theory. Other observers have stated the presence of thin flows of rhyolite among the tuffs with flow structure plainly observable. The writer has up to the present been unable to confirm this.

The rhyolite away from the zone of mineralization is a fairly hard dense rock. Its color varies from light gray through pink to a slight purplish color. The groundmass is in excess as a rule but occasionally the phenocrysts of quartz and feldspar become so abundant as to comprise the major portion of the rock. The relative proportion of the quartz and feldspars vary considerably in different specimens, but as a rule the feldspars are in excess. The phenocrysts of both quartz and feldspar attain at times large dimensions, especially the feldspar, which reaches an inch or more, measured parallel to the prism. The feldspars are never found unaltered but are generally entirely changed to some secondary mineral, in most cases sericite. The change of the feldspars to sericite is characteristic of the rhyolite outside of the ore channel, while in the ore channel the change has been largely to kaolin. The feldspars are largely prismatic in form, occasionally tabular. Owing to the change to sericite, remarked above, only the outlines of the feldspars are left and these in a good many cases are largely obliterated. For the same reason the type of feldspar present cannot be definitely determined, but it is believed to be almost universally orthoclase. The feldspars occasionally alter partially to calcite with some quartz and sericite which may indicate the presence of some lime-bearing feldspars.

The quartz phenocrysts are idiomorphic to hypidiomorphic. Embayments due to the eating away of portions of the crystal by the liquid groundmass are common. The crystals are often cracked and broken and the parts separated from each other. There are occasional traces of what was possibly a ferromagnesian mineral, probably an amphibole.

The groundmass shows no glass but is entirely devitrified and ap-

pears to consist of submicroscopic feldspars and quartz. Small amounts of sericite in the groundmass probably resulted from the alteration of the feldspars. Zircon and apatite are present in small amounts as well as occasional grains of tourmaline.

The following rock analysis is typical of the rhyolite away from the ore channel. The alkalies were obviously about 4 to 5 per cent.

| $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | FeO | CaO | MgO | $K_2O$ | $Na_2O$ | Loss | Total |
|---|---|---|---|---|---|---|---|---|---|
| 70.20 | 14.20 | 4.85 | .... | 1.70 | 0.70 | not determined | | 4.00 | 95.65 |

The variety of rhyolite just described is typical of the major portion of the exposure. There is, however, a small section north of the upper end of Hershall Valley in which the rock is a coarse breccia. Again, other portions, as that comprising the mass of Mt. Teddy, consist of a multitude of small rock fragments in a groundmass of unresolvable volcanic dust. The nature of the rock comprising the ore channel has been so entirely transformed from its original character by crushing, silicification and other alterations consequent on the mineralizing solutions that it must be discussed separately. It should be noted, however, that the southwestern portion of the rhyolite tuff form the head of Hershall Valley to where it disappears under the sediments to the south of the Sterne River is a rock composed largely of small rock fragments, of broken crystals of quartz and feldspars and of great amounts of unresolvable volcanic dust, and even before the changes due to the faulting and mineralization was different in its character from the tuff characteristic of the rest of the rhyolite exposure.

The passage from the rhyolite tuffs to the overlying sandstone and shale is a gradual one and the exact boundary between the tuffs and sediments cannot be generally drawn. This is especially true along the southwest contact where the greatly altered rock of the mineralized zone abuts on the sediments. It appears that the basal members of the overlying sediments (coarse impure sandstones with large amounts of feldspars and rock fragments) were formed immediately after the deposition of the coarse tuffs and possibly before their consolidation. Thus the lower beds of sediments are simply the worked-over and somewhat sorted tuffs. Above these coarse impure sandstones, is a great thickness of well-bedded red and white sandstones, purplish and micaceous shales and occasional conglomerate bands. No fossils have been found in these beds. Conformably overlying these beds is a great thickness of Ordovician sandstones and marls with abundant characteristic Ordovician fossils. The geological sequence in the vicinity of the mine would thus appear to be as follows: A basement series of Cambrian or pre-Cambrian shales and quartzites followed by rhyolite tuffs and breccias. Conformably above the rhyolite is a great thickness of unfossiliferous sandstone and shale which is in turn conformably overlain by Ordovi-

cian beds. The unconformity (as observed at other points in the Northern Shan States) between the Ordovician beds and the basal series is a very marked one, the lower beds having been severely folded and greatly eroded before the deposition of the Ordovician strata. The sequence from the rhyolite through the unfossiliferous sandstones and shales to the Ordovician beds is, however, a conformable one. It does not seem probable that the Bawdwin rocks correspond in time of formation to the unconformity between the basal beds and the Ordovician, especially as a portion of the interval was one of erosion and not of deposition. Consequently the Bawdwin rhyolites are most probably of early Ordovician or late Cambrian age while the basal beds should be assigned to early Cambrian or pre-Cambrian. The rhyolite probably lies unconformably on the basal beds.

### *Ore Zone*

The orebodies occur in a wide zone of displacement (300 to 1,000 ft. across) in rhyolite tuff in which the faulting and crushing have been intense (Fig. 5). The zone has no well-defined boundaries but dies away gradually into hard undisturbed rock. As has been mentioned previously, the southwestern portion of the rhyolite area differs considerably in its lithological character from the remainder of the rhyolite. In this portion the tuff-like nature of the rock is much more pronounced, the constituents are considerably more numerous and variations in the type of rock are frequent. It is through this portion of the rhyolite that the zone of faulting passes, and, therefore, superposed on the original mixed character of the rock, are the great secondary changes due to the introduction of foreign materials and to the rearrangement of the minerals already present. The faulting extends through this type of tuff into the fine-grained tuff which comprises the major portion of the rhyolite area, but its effect there has been much less, due partially to the more homogeneous and compact nature of the rock and probably partially to a dying away of the severity of the faulting in that direction.

The rock of the ore channel varies from a fine-grained greatly silicified tuff consisting largely of introduced quartz to a rock largely composed of included fragments of other rocks, often dark blue or black angular fragments of shale. A type very characteristic of the neighborhood of the large orebodies is one consisting largely of a great number of large kaolinized feldspars set in a fine-grained groundmass. The alterations of the rock in the ore channel are so great that the original character is often almost entirely destroyed. The greatest single factor is the silicification which has penetrated all through the rock, replacing the groundmass, feldspars and included fragments. The feldspars are, however, generally altered to kaolin, the alteration being complete, the whole mass of the feldspars being changed. The change sometimes takes place

to a mixture of calcite, quartz and sericite instead of to kaolin. Considerable sericite also forms in the groundmass and occasionally from included fragments. Large amounts of a light-green non-pleochroic alteration product, probably a form of chlorite, are found. At a few points in the ore channnel small portions which escaped the intense crush-

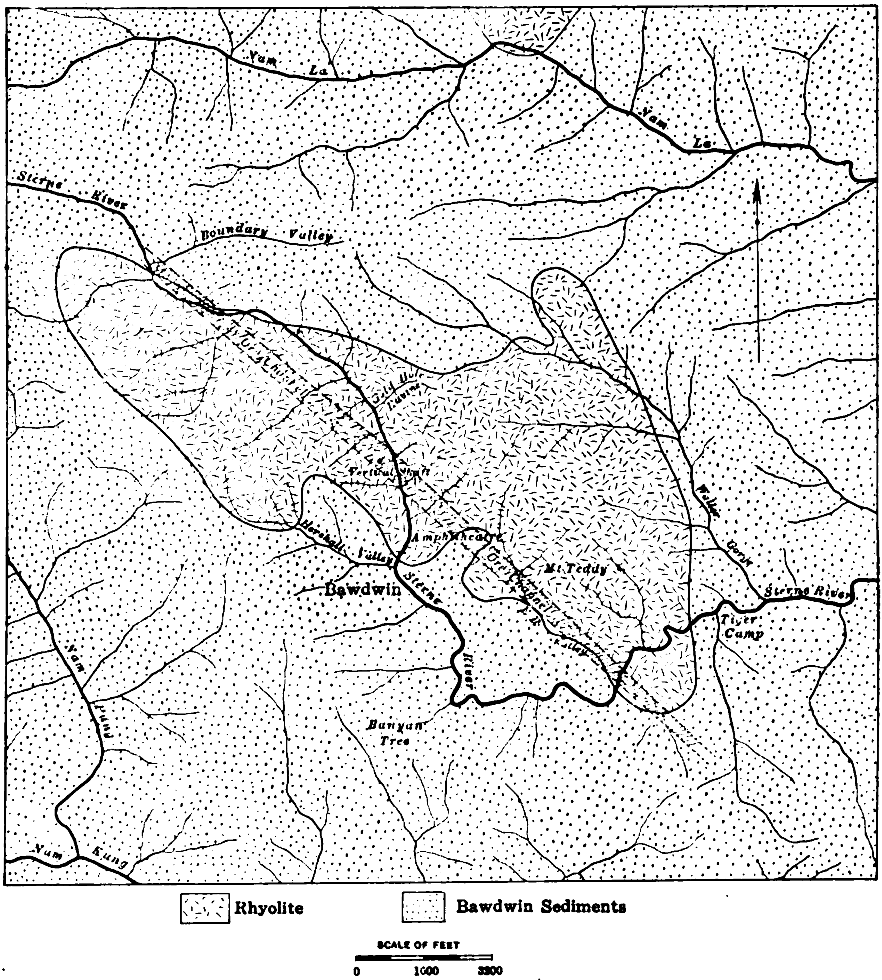



FIG. 5.—GEOLOGICAL SURFACE MAP. BURMA MINES, LTD.

ing and silicification and which retain the characteristics of the fine-grained tuff are preserved.

In a general way the following changes take place in the rock on approaching the ore channel. At a distance of 500 or 600 ft. the rock is a normal rhyolite containing considerable sericite and calcite due to ordinary secondary changes. On approaching the ore channel, the silica

increases and the alumina decreases. The amount of sericite in the groundmass increases and microscopic cubes of pyrite become numerous. Still closer to the ore channel, chlorite becomes abundant and the alteration of the feldspars takes place partly to kaolin instead of to sericite. Finally, in the ore channel, itself, the rock is a mixture of quartz, sericite, chlorite and kaolin with some calcite.

The following are rock analyses from various points in the ore channel. Owing to the extremely variable character of the rock itself, a tuff with included fragments, it is unsafe to draw any conclusions by a comparison of these analyses with the one given as typical of the unaltered tuff.

| | $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | FeO | CaO | MgO | ($K_2O$ $Na_2O$) | Loss |
|---|---|---|---|---|---|---|---|---|
| (1) | 77.20 | 8.28 | 0.71 | 1.09 | 1.07 | 0.48 | 2.55 | |
| (2) | 71.60 | 8.74 | 2.67 | 4.35 | 1.30 | 1.19 | 2.69 | |
| (3) | 80.40 | 10.25 | 0.46 | 1.28 | 1.40 | 0.37 | 2.91 | |
| (4) | 74.65 | 9.05 | 5.35 | ........ | 3.05 | 1.70 | determined | 4.48 |
| (5) | 83.65 | 8.00 | 2.85 | ........ | 1.45 | 0.45 | determined | 0.94 |

(1) Close to footwall of Chinaman Lode.
(2) Several hundred feet from hanging wall of Chinaman Lode.
(3) In Chinaman Lode—a block of country rock with large kaolinized feldspars and included in the lode.
(4) 300-ft. level—south of Shan Copper Lode.
(5) Close to surface on hanging-wall side of lode.

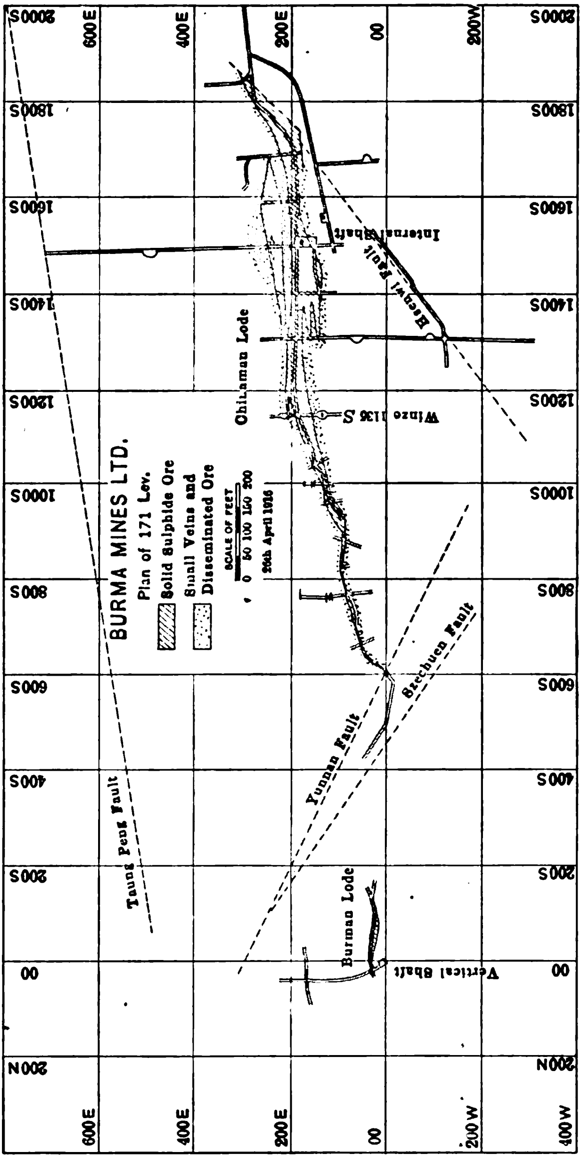
Lead, zinc and sulphur bring the totals of the above analyses up to approximately 100 per cent.

### *Orebodies*

The metals of commercial importance found at Bawdwin are zinc as sphalerite (ZnS), lead as galena (PbS), copper as chalcopyrite ($CuFeS_2$) and silver whose manner of occurrence is not definitely known but which is probably present largely as argentite in submicroscopic mixture with the galena.

The ore occurs as solid masses of sulphides with practically no admixture of gangue; as veins; as interlacing seams forming small stockworks; and as an impregnation of the country rock. The gangue when present is quartz, calcite or country rock (rhyolite tuff). The principal orebody, known as the Chinaman Lode, is at present developed for about 1,200 ft. in length and varies in width from a few feet to over 100 ft., maintaining on some levels an average width of 50 ft. for over 1,000 ft. along the strike (Fig. 6). It is primarily a zinc-lead-silver orebody with small amounts of copper along the edges. Some of the diverging seams and faulted portions of the Chinaman Lode are, however, variable in their metal content, showing rapid alternations from zinc-lead to copper

ore and *vice versa.* These changes take place both horizontally and vertically, sometimes as a gradual transition, sometimes marked by a fault plane. The largest single body of copper occurs on the 300-ft. level,




north of the Chinaman Lode. It is about 130 ft. long with an average width of 25 ft. running 14 per cent. copper, 7 oz. silver, and with only small percentages of zinc and lead on this level. It grades, however,

about 75 ft. above into zinc-lead ore and does the same at 30 ft. below. This lode is probably the faulted northern extension of the Chinaman.

The orebodies thus far opened up occur entirely in the altered silicified rhyolite tuff. The mineralization extends to a slight degree into the overlying sediments. It is possible, however, that this mineralization in the overlying sediments belongs to a later period than that in the tuff and is more of the nature of a slight retaking into solution of the minerals in the tuff and redeposition in the sediments by ground-water circulation. The nature of the orebodies in the underlying sediments is as yet unknown. It is believed that they will probably be more restricted as to size and be more in the nature of fault deposits, with replacement of the country rock assuming a subordinate rôle, the opposite of the conditions in the tuff.

A cross-section through the Chinaman Lode shows a central core of solid zinc-lead ore, with the zinc generally but not invariably in excess of the lead. On both sides of this central core are alternating bands of solid ore and heavily mineralized tuff. These bands parallel the main body in strike and dip but are not persistent themselves, coalescing and pinching out and in reality forming a sort of stockwork. The bands are generally high in lead and comparatively low in zinc. A slight percentage of copper is generally found on their edges. From both sides of these bands the mineralization extends far out into the tuff, gradually merging into barren rock. Occasional seams and patches of ore are found at considerable distances. There is no sharp boundary between mineralized and unmineralized country rock as a general thing, although this condition is approximated in a few places by fault planes. Pyrite is found to a limited extent in the main orebody, but attains a much greater abundance along the edges where it is disseminated all through the rock as a multitude of very small (microscopic) cubes. The central core of solid ore in the Chinaman Lode attains at points a thickness of more than 80 ft., and on some horizons maintains an average width of 55 ft. for over 800 ft. in length and, as stated above, an average of 50 ft. for 200 ft. additional. In this core the gangue is generally very finely disseminated quartz grains. The extreme richness in metal content of the orebody is best shown by the fact that a block roughly 800 ft. long by 600 ft. deep by 30 ft. wide contains about 1,750,000 long tons with an average value of approximately: Ag, 30 oz.; Pb, 31 per cent.; and Zn, 29 per cent. A theoretical block of the same size of solid galena and sphalerite with equal amounts of Pb and Zn would contain approximately 2,300,000 long tons. Thus the block in the mine is over 75 per cent. solid lead and zinc sulphides. The accompanying cross-section (Fig. 7) through the orebody shows that this solid sulphide core has a high dip to the west in the upper levels and turns gradually over to a high dip to the east in the lower levels. A gradual thinning of the orebody takes place on approaching the underlying sediments. This is probably due to the greater effect of the fault-

ing on a heterogeneous rock like the tuff than on the sandstone and shale. This wider zone of shattered rock in the overlying tuff and, in addition, the much more easily replaceable character of the rock, would result in a considerably wider area of mineralization.

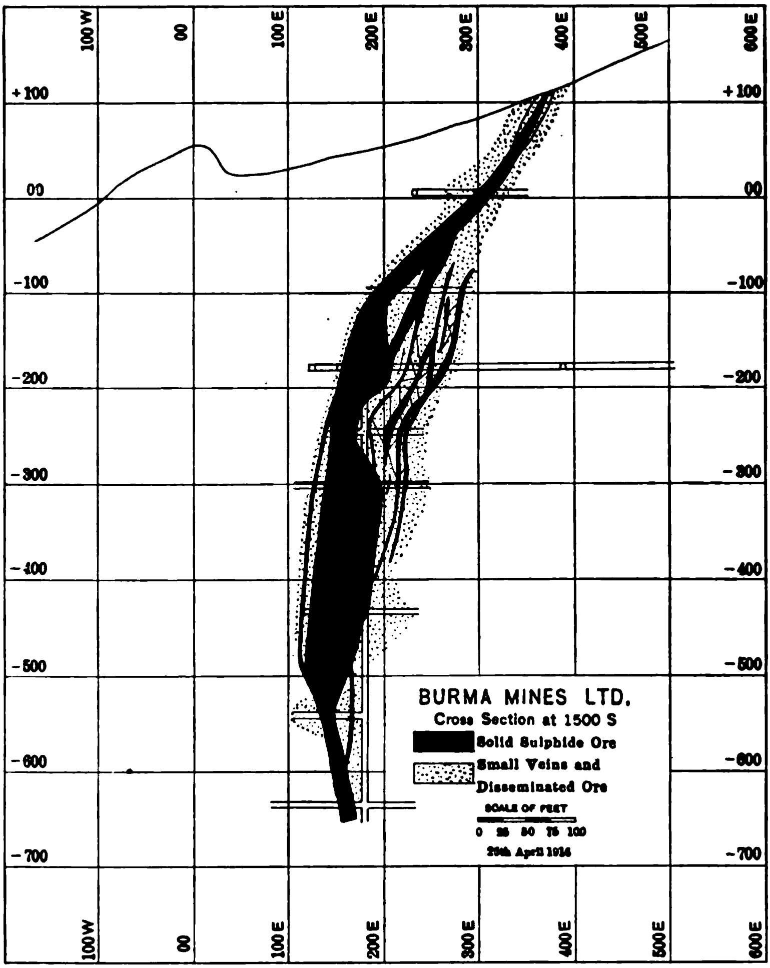



FIG. 7.

The original condition of the upper portion of the orebody is largely concealed by the work of the Chinese, but there appears to be a decided thinning of the orebody on approaching the surface. The outcrop, in fact, when considering the size of the orebody below, is scant, but this is probably partially accounted for by the oxidizing and subsequent leaching of the surface ores. At several points sphalerite and galena outcrop on the surface, but as a rule the surface ores have been largely

leached and the outcrops are marked by a wide zone of soft decomposed rock colored by iron oxides, copper carbonates and, to a lesser extent, by oxidized lead ores. This gossan for considerable stretches runs 3 to 4 oz. in silver and about 5 per cent. in lead. It is quarried at one point for a siliceous flux for smelting purposes. A characteristic assay follows: Ag, 3 oz.; Pb, 5.8 per cent.; Zn, 0.7 per cent.; $SiO_2$, 58.6 per cent.; Fe, 17.7 per cent. The depth of the gossan is extremely variable but very rarely is more than 50 ft. At approximately this depth it is succeeded by a zone of secondary copper sulphides, principally chalcocite with some bornite. The chalcocite occurs largely as a replacement of sphalerite. The secondary copper ores cannot be considered as occurring as a well-marked zone over the whole orebody, but simply at a few favorable places, at some points stretching well up into the oxidized ores; at others, well down into the normal sulphides. This secondary copper is

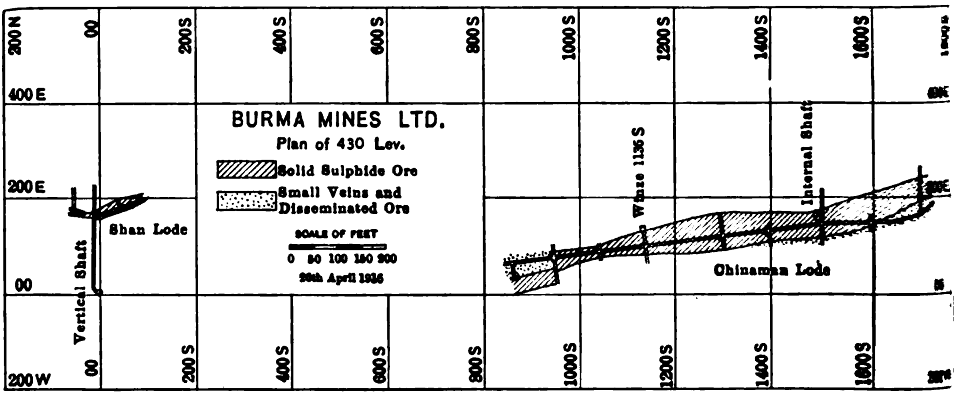



FIG. 8.

not present in large enough amounts to be commercially important. The points at which the chalcocite occurs are about 100 ft. above the present water level, but as erosion has been rapid it probably marks the water-table level at a not very distant period. The zinc and lead sulphides which, as remarked previously, at some points outcrop on the surface, extend down to the greatest depth yet reached, about 725 ft., with practically no change in their character or relationship to each other. The normal zinc-lead ore of the mine is an extremely intimate mixture of sphalerite and galena; the sphalerite, which is the older, being of a very fine granular structure, the grains varying from about 0.015 to 0.16 mm. in diameter. The galena forms around the separate grains of sphalerite, and as thin filaments running through the zinc. The ore grades off in both directions from this fine mixture, toward the zinc end to solid masses of soft, earthy sphalerite with occasional stretches of hard, dark sphalerite probably rather high in its iron content, and toward the lead end to masses of pure, coarsely cubical galena.

The silver content rises and falls consistently with the lead and independently of the zinc, indicating that the silver is largely contained in the lead. The presence of copper even in small amounts destroys the silver-lead ratio, as the copper present in the small veins adjoining the main orebody carries a high silver content (Fig. 8). As a general statement, it can be said that 1 per cent. of lead carries 1 oz. of silver. Silver minerals have not been detected in any form but, as stated above, it seems most probable that the silver is present as argentite intimately associated with the galena.

The accompanying assays show the relations between the silver and the lead and also represent typical assays of high-lead, high-zinc, zinc-lead and lower-grade ores.

| | Ag, Oz. | Pb, Per Cent. | Zn, Per Cent. |
|---|---|---|---|
| *Silver-lead Ore* | 47.6 | 50.4 | 19.2 |
| | 54.4 | 50.0 | 24.1 |
| *High-zinc Ore* | 11.0 | 11.2 | 38.8 |
| | 15.5 | 13.6 | 40.7 |
| *Zinc-lead Ore* | 38.5 | 33.9 | 36.3 |
| *Second-grade Ore* | 11.8 | 22.5 | 7.0 |
| | 18.0 | 20.0 | 12.5 |

## *Faulting*

The faulting along which the ore occurs was not concentrated in one fault plane, but occupies a zone reaching in places well over 1,000 ft. in width. Along this zone a multitude of faults of varying strike and dip—some normal, some reverse—were developed. The faulting is extremely complicated and of such varying strike and dip that a division into rigid systems cannot be made. The majority of the faults can be traced only for short distances and are either displaced by later faulting or die out. Roughly, however, two main fault systems can be determined, one with strikes about north and south and with high dips to either the east or west; and a second series with strikes varying from N35W to N75W and with high dips generally to the southwest. The displacement due to most of the faulting has been very slight, as is evidenced where faults have cut the orebodies. The displacement in these intersections is generally so small as to be barely noticeable. A few dominant faults, however, which can be traced for long distances, have caused severe dislocations. One such fault cuts off the Chinaman orebody on the southern end. So extended was the movement that the ore south of the fault has not yet been located. There are, in addition to the above faults, a great multitude of smaller ones of all strikes and dips but of subordinate importance. The faults as a rule are marked by varying thickness of soft gouge and rounded rock fragments. The gouge is often colored black by the minute sulphides contained in it. Faults when passing through the solid bodies of sulphides have produced smooth

slickensided surfaces, often with a brilliant polish which may be modified by etching. This is especially true of fault planes in the high-grade galena ore.

The faults just described appear to be later than the mineralization, although it is possible that some of the faults that strike north and south along the edges of the orebody are pre-mineral and acted as bounding walls for the main ore deposition. It is also most probable that some of the small veins branching off from the main Chinaman orebody represent ore deposits along fault planes. That these smaller veins are contemporaneous with the Chinaman Lode is proved in some cases by direct observation of the points where they merge into it. In the large replacement orebody, the Chinaman, the practically complete replacement of country rock by sulphides has entirely obliterated the original character of the ground. This orebody represents, however, a replacement in a wide faulted and crushed zone rather than a replacement that has spread out into the walls from a central fault plane. Many of the faults that are later than the main orebodies contain ore due to drag and, in some cases, ore probably formed by solution from the original orebodies and by redeposition along the fault planes which have naturally served as water channels.

The plan of the 171-ft. level (Fig. 6) shows some of the more prominent faults. The large amphitheater, directly above the Chinaman Lode, is bounded to a great extent by faults, and partly owes its existence to land slips along these planes. Its precipitous sides have probably been slightly accentuated by Chinese work but the amphitheater taken as a whole is probably almost entirely due to natural causes.

### *Underground Waters*

As is to be expected in a tropical region of fairly heavy rainfall, the ground-water level is close to the surface. The ore channel strikes diagonally across the trend of the ridges and the ground-water level was tapped by the company's drainage tunnels only a few feet below the valley bottoms (Fig. 9). There did not, however, appear to be a completely saturated zone below the water level. Diamond-drill holes at a greater depth than any of the workings occasionally go completely dry, and water pumped into them entirely disappears. This experience was met even in cases where the holes were cemented practically all the way down. The great number of old workings and the extremely shattered condition of the ground with numerous fault planes along which water circulates freely, tend to obscure the natural underground circulation. A heavy rainfall has an almost immediate effect on the amount of water in the upper levels, and a portion of the surface water probably drains down along faults to the lowest levels. Two analyses of water are given here-

with: No. 1 from a shaft sunk in the Chinaman Lode; No. 2 from a drainage tunnel which at the time the sample was taken was about 6,500 ft. from the orebody and probably represents with fair accuracy the underground water of the region away from the influence of the orebody.

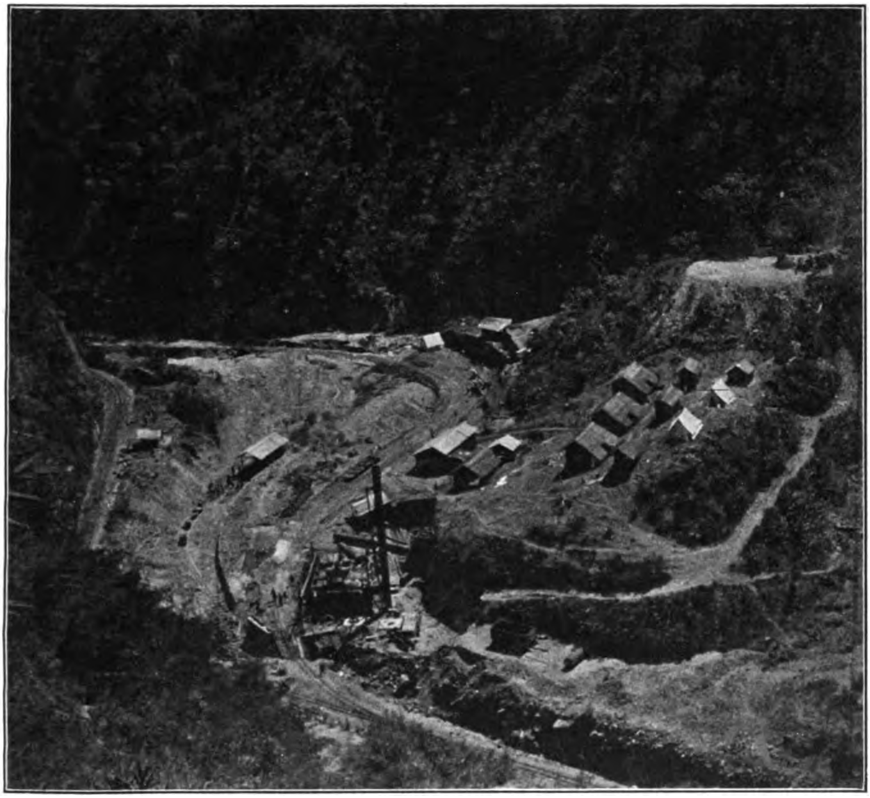


FIG. 9.—SETTLEMENT AT THE PORTAL OF THE TIGER TUNNEL. A DRAINAGE AND WORKING TUNNEL 7,000 FT. LONG.

*Parts per* 1,000,000

No. 1

| | |
|---|---|
| $SO_3$ | 2,160 |
| CaO | 227 |
| $Fe_2O_3$ and $Al_2O_3$ | 273 |
| $SiO_2$ | 35 |
| MgO | 227 |
| Pb | 30 |
| ZnO | 1,179 |
| NiO and CoO | 250 |
| Organic and volatile matter | 30 |
| Cl | Tr |
| Undetermined | 21 |
| Total | 4,434 |

No. 2

| | |
|---|---|
| $CaCO_3$ | 138 |
| $MgCO_3$ | 78 |
| $Na_2SO_4$ | 19 |
| $Fe_2O_3$ and $Al_2O_3$ | 13 |
| $SiO_2$ | 136 |
| Undetermined | 9 |
| Total | 393 |

No. 1 is a strong sulphate water with an extremely high content of zinc. The large amount of nickel and cobalt in solution is remarkable inasmuch as the sulphides of these metals have not been definitely recognized in the orebody. The cobalt arsenate, erythrite, however, forms rapidly along some exposed faces. This sample was taken at a point about 400 ft. below the surface, the lowest point in the mine at the time, and it does not appear probable that the large amount of zinc is due to any extent to solution from exposed faces on the upper levels, because, if this were true, copper should also be present, as the descending water of the upper levels has a high copper content. The entire absence of copper is noteworthy and appears to indicate either that the body of underground water is virtually stagnant and its mineral content is derived from the immediate vicinity, or that the contained copper in the water above is redeposited before reaching the lower levels. A large number of the faults have impervious gouges and probably break the ground up into blocks, some saturated, some practically dry.

### *Minerals*

The list of minerals found at Bawdwin is not extensive. The principal ones follow:

*Quartz* as phenocrysts in the rhyolite; as a large portion of the cryptocrystalline groundmass; as a secondary product from the alteration of other minerals; as a later introduction in veinlets; and as a product of general silicification. It appears in veins in massive form, and is hard and compact, but it may also yield well-formed pyramids in cavities. It may constitute ribbon quartz with lamellar structure. It may even be powdery. It can be detected as very fine grains, intimately associated with the sulphides.

*Feldspar,* probably largely orthoclase, occurs in crystals both large and small. Often only fragments of crystals appear, and both these and the well-bounded ones are generally badly decomposed. With quartz it forms the groundmass of most of the eruptive rock.

*Calcite* is abundant as a secondary mineral, especially as a replacement of other minerals, notably feldspar. It may be one of the gangue minerals and favors the copper ore in which relation it often composes the entire gangue.

*A ferromagnesian mineral,* probably an amphibole, has suffered such extreme alteration as to be almost entirely destroyed.

*Barite* appears in large masses as vein-filling on the surface and as large weathered blocks. It has not been observed underground. Whereever seen, it is always in association with the sediments, never with the rhyolite.

*Siderite* is common as a gangue in some of the smaller veins but is of

ubordinate importance. Iron-stained calcite may sometimes be taken or it.

*Apatite* and *zircon* are fairly common as small crystals disseminated hrough the rock.

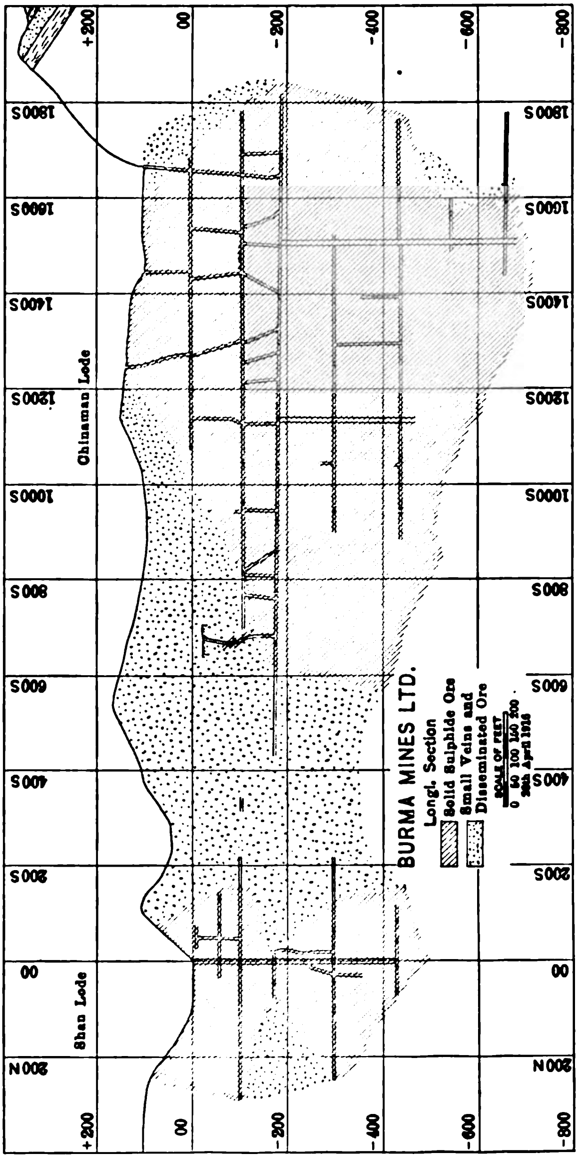



FIG. 10.—LONGITUDINAL SECTION OF OREBODY.

*Tourmaline,* occasional grains in tuff.

*Biotite,* occasional shreds, probably secondary.

*Sericite* is the predominant alteration product of the groundmass,

and away from the ore zone also of the feldspars. It occurs as a mat of slender microscopic crystals.

*Kaolin* is the principal alteration product of the feldspars in the ore zone.

*Chlorite.* A light-green, non-pleochroic, alteration product is common in all types of the rock, and is probably a form of chlorite.

*Galena* occurs as coarse cubical galena in quartz stringers; as impregnations in the country rock; as replacements of silicates or quartz; and as a mixture in all degrees of intimacy with sphalerite and to a lesser degree with chalcopyrite.

*Cerussite,* as well-formed crystals in vugs in galena. It is present in considerable amounts as low as the 171-ft. level and to a lesser extent further down. It occurs also in the altered outcrops.

*Anglesite* appears in relations similar to those of cerussite.

*Pyromorphite,* as well-formed crystals in relations similar to those of the two preceding minerals.

*Sphalerite* occurs as large solid masses; as an impregnation of the country rock often finely disseminated; as replacements of other minerals, especially of the groundmass and feldspars of the rhyolite; as an intimate mixture with galena and occasionally with chalcopyrite, and as large masses of earthy sphalerite. The normal oxidized zinc minerals have not been observed, but goslarite forms rapidly and in large amounts along the walls of the drifts in the upper levels.

*Chalcopyrite* occurs in the smaller veins and along the edges of the Chinaman Lode and as the principal ore of the Shan Lode on the 300-ft. level (Fig. 10). Is often associated with galena and sphalerite but the relation is never so intimate as between the galena and sphalerite.

*Malachite, azurite and native copper* occur in small amounts in the oxidized zone.

*Chalcocite* appears as a secondary product above the water level; largely as a replacement of sphalerite, the replacement being generally not complete, so that a core of sphalerite is surrounded by chalcocite, usually of a soft earthy variety.

*Bornite* occurs in small amounts with the chalcocite.

*Pyrite* occurs in small amounts in the orebody as the last sulphide deposited. Great amounts of it lie along the edges of the orebodies, largely as a multitude of microscopic cubes disseminated all through the rock.

*Hematite* and *limonite* are found in the gossan above the orebodies.

*Erythrite* constitutes a pink incrustation along the drifts.

*Nickel and cobalt* sulphides are present but have not been identified.

### *Genesis*

The orebodies at Bawdwin have been formed by the metasomatic replacement of the rhyolite tuff by sulphides deposited from hot solutions, which rose from below along an intensely crushed and sheared zone. From

the data at hand it is not possible to make any definite assertion as to whether the mineralizing solutions are themselves a late phase of the rhyolite activity or not. Whatever evidence there is, however, appears to indicate that the rhyolite is not responsible for the metal-bearing solutions. About 25 miles northeast of Bawdwin, at Mohochaung, there are orebodies in sandstones and shales which are entirely similar to Bawdwin as regards the character of the ore itself. The beds in which they occur are probably roughly contemporaneous in age with the Bawdwin sediments. There is no rhyolite present at this point. Granite is, however, present about 3 miles to the west of Bawdwin and at roughly the same distance to the west of Mohochaung. The granite is a coarsely crystalline muscovite-biotite granite. It is a reasonable presumption to consider the granite mass as extending below the orebodies at both Mohochaung and Bawdwin, probably at a great depth below the surface, and it may be to the influence of the granite that the mineralizing solutions should be ascribed. This is, however, simply a hypothesis with no actual facts to support it.

The fault zone at Bawdwin passes through all the varying types of tuff, from the type composed largely of included fragments of other rocks to the fine-grained homogeneous rhyolite tuff. The effect of the kind of rock upon the ore deposition has been marked. The fine-grained homogeneous tuff is practically barren, the fine-grained highly silicified rock composed of fine rock fragments contains small bodies of ore along fault planes but with practically no replacement of the rock itself. With an increase in size of the fragments composing the rock there is an increase of replacement by metallic sulphides. The type composed of large numerous feldspar crystals set in a fine-grained groundmass is distinctly the most favorable rock and is, in fact, the rock in which all the large zinc-lead bodies are found.

The favorableness or unfavorableness of the various types of rocks toward ore deposition are due to several reasons.

1. The compactness or, inversely, the porosity of the rock.
2. The nature of the rock constituents as regards their ease of attack by solution and replacement.
3. Effect of the faulting on the rock—whether the rock gives way along a few prominent planes or whether the movement and consequent crushing is spread over a wide zone.

The coarse heterogeneous tuff, especially the one with large feldspars, was due to the nature of the rock itself which was not a compact one and consequently allowed an easy entrance of mineralizing solutions. The feldspars offered an easy first point of attack for the solutions and the rock itself, owing to its mixed character, suffered crushing and faulting over a wide area. All the above factors become progressively less active as one passes toward the fine-grained homogeneous type of tuff.

The rhyolite tuff was greatly shattered and crushed along the zone of faulting, probably to a much greater extent than is now observable, as the rock has been reconsolidated by later silicification. The ascending solutions permeated all through the crushed mass, and along the zone of greatest shattering practically entirely replaced the original rock with sulphides. The feldspars which were probably the first point of attack were either replaced by sulphides or altered to sericite or kaolin. The alteration to kaolin may, however, be due to later descending solutions carrying sulphuric acid, although somewhat mitigating against this theory is the fact that the kaolin alteration persists several hundred feet below the water level. After the feldspars, the fine-grained groundmass was attacked, and finally the quartz. Deposition also took place in a subordinate degree in open spaces, as is evidenced by banding and rough comb structure and by drusy and botryoidal forms of sphalerite. Vugs and openings along fault planes are present in all parts of the mine, being, however, considerably more numerous in the upper levels where they are probably largely of recent origin. In the central core of solid ore none of the original country rock is left; where any gangue exists it is a very fine-grained quartz probably introduced with the sulphides. The sulphides were deposited in successive stages which, however, probably overlapped. The sphalerite is the oldest, followed by chalcopyrite and galena and finally by pyrite, the iron sulphide being to a great extent the last, although it was probably introduced in small amounts all through the period of the ore deposition. The origin of the chalcopyrite was contemporaneous with the introduction of considerable calcite, the latter mineral being a characteristic gangue of the copper ore. The relative ages of the sulphides as outlined above are shown not only in their relation to each other but also to some extent by their general location. The zinc forms the central core with the lead and copper along the edges of the zinc. This generalization is true only to a certain degree as the successive movements which opened up new channels for ore deposition along the edges of the already existing bodies also shattered these bodies. The mixture between the lead and zinc is, however, such an extremely intimate one that it is difficult to assume that it is formed solely by galena introduced into openings, however minute, in a shattered body of sphalerite. The relation of the two minerals to each other as shown in thin sections clearly indicates that they were not deposited simultaneously but that the lead is distinctly younger than the zinc. It would appear that the intimate relationship may be partly due to a replacement of the sphalerite by the galena. The galena having been introduced along minute cracks in the sphalerite gradually spread out from these cracks by replacement of the zinc, thus reproducing on a microscopic scale the method of formation of the orebody as a whole.

It does not appear that changes of any magnitude have occurred

in the condition of the orebodies since their original formation. There is no distinct zone of secondary enrichment (barring the small amounts of chalcocite), a fact best indicated by the silver values which show no marked change from the upper part of the orebodies to the lowest levels, several hundred feet below the water level. There is a gradual increase of silver values on descending, but this is entirely due to a corresponding increase in lead. As has been stated previously, the lead-zinc ore maintains the same character from the surface to the lowest levels, varying somewhat in the proportion of one to the other but not due to any later rearrangement. As stated above, there is a gradual increase in silver and lead on descending. This is not, however, at the expense of the zinc, which also increases slightly, but is due to the almost entire disappearance of gangue minerals and of low-grade patches in the solid core.

The age of the orebody itself is extremely difficult of determination. The overlying sediments have been eroded from directly above the orebody and whatever patches of ore have been found in them at other points are of small size and may have been formed from already existing bodies in the tuff. This is not, however, certain and the barrenness of overlying sediments may be due to their unfavorable character as regards replacement by sulphides. If the overlying sediments are later than the ore deposition, it would mean that the orebodies are early Ordovician or late Cambrian. If the sediments, however, antedate the period of ore formation it becomes impossible to determine the age of the orebodies.

## Features of the New Copper Smelting Plants in Arizona

**Discussion of the paper of A. G. McGregor, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1257 to 1279.**

L. D. Ricketts, New York, N. Y.—The advance which has been made in smelting has been in the line of cheaper cost of handling, due to larger units and decrease in losses. At the International smelter, Mr. McGregor designed everything to handle the ore under cover. But we do find an unaccounted for loss of 0.7 per cent. of our copper, which is serious. Mr. Wallace has made a suggestion that I think metallurgists ought to know of and think about. He made it years ago—and that is that one of the chief losses is undoubtedly in the charging and discharging of the calcine cars. His idea is that possibly the dryers and heaters might be put immediately over the reverberatories so that they could discharge in conveyors. They could feed on both sides, and thereby the dust raised by discharging in the cars from the hoppers could be overcome.

E. P. Mathewson, Anaconda, Mont.—I might say a few words on the plants we have visited so far. I was particularly struck with the cleanliness of the plants, which is next to Godliness. I noticed also that the skull-cracker arrangement they have for handling their converter slag is very good. And they have paid considerable attention to ventilation, and are taking steps to take care of the health of the men. "Safety first" has been well looked after in all the plants we have visited so far. The steel structures are right up to date, and the buildings are of the most modern type, and everything is conveniently arranged. I think the designers of the plants in the Southwest are to be congratulated on what they have done.

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## The Basic-Lined Converter in the Southwest

**Discussion of the paper of L. O. Howard, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1539 to 1543.**

The Chairman (Walter Douglas, New York, N. Y.).—I presume, gentlemen, that Mr. Howard's experience with the Great Falls basic-lined converter has been as regards the tonnage produced from a single lining, perhaps, a world's record. About 6 months ago Mr. Howard shut down and tore out what was left of the first basic-lining which was installed at the Old Dominion, the lining having produced something over 70,000,000 lb. of copper. This was due probably not only to the very excellent grade of the matte which was produced by the smelter there,

but also to the fact that after the tonnage began to reach record proportions Mr. Howard lived with that lining night and day, and watched it most attentively. May we hear from any of the gentlemen present with reference to this paper of Mr. Howard?

E. P. MATHEWSON, Anaconda, Mont.—I would like to say a few words in regard to the basic-lined converter. I noticed at El Paso that they had not adopted the Great Falls type of converter, but had stuck to the Peirce-Smith type of horizontal converter. This type has certain advantages over the Great Falls type and certain disadvantages. We have in the Northwest still larger furnaces of the Great Falls type, many of them 20 ft. in diameter. These furnaces are doing good work, due to the advantage that the Great Falls converter possesses. The sole disadvantage, I think, is that in tilting the furnace, if care is not taken by the operator, some of the tuyères are under a greater depth of matte than others; there is liable to be splashing and loss of blast from this same cause. In the horizontal type the tuyères are at the same level, and the level can be adjusted to meet the blast available, so that very little splashing results. There is a disadvantage on the other hand with the horizontal type converter in that it is hard to distribute the silica evenly over the matte without making several openings. This difficulty is not met with in the Great Falls type, as the silica may be dumped without care, and will distribute itself evenly over the matte. But it will be very interesting, as I say, to note the comparison between these two types on a similar grade of matte. As I understand, the matte used at El Paso is about the same grade as used at the other points in the Southwest and about the same as in the Northwest. We have records of our furnaces extending back over 2 years, and I think we have forgotten that these linings have any life limit. A little care—which means that the operator must not be careless—keeping the heat down to reasonable limits, and you will not have difficulty with the lining. The main part of the lining we seem to be able to make last indefinitely, care being taken to keep the silica low in the slag.

KUNO DOERR, El Paso, Texas.—I am sorry to say that the new converter at our El Paso plant was put into use only about 10 days ago, due to the fact that we have had difficulty in getting magnesite brick to line it with. This converter is of the P-S type, 13 ft. in diameter by 30 ft. long, whereas our old Peirce-Smith converters are only 10 ft. in diameter by 26 ft. long. In the few days in which the new converter has been in operation we have observed a great improvement in it over the operation of the old converters. It has a large stack, or gas outlet, and makes practically no splash at all—that is, what splash there is goes back into the converter itself, whereas in our old converters we have had a great deal of trouble from accretions in the stack and hoods, and there has

been a very large amount of cleanings to take up and resmelt. Thus far we have had none of this in operating the new converter.

Our average blister production per lining in the 10-ft. Peirce-Smith converters was about 7,000 tons. We have never had to reline the bottoms of any of our converters, and our repairs usually consist of relining with new brick in the tuyère zone only. A patch made without shutting down the converter except between blows is not considered in our records as a new lining, but any shutting down and cooling off of the converter for repairs terminates our production record per lining. Thus we actually get much more than 7,000 tons of blister on the average number of brick in a converter lining. Our mattes vary somewhat, but will not run much under or much over 40 per cent. copper.

One source of trouble with our old converter installation was the fact that the converters were not served by electric cranes, but all the matte from the furnaces to the converters was transported in open ladles over our industrial railway, and the run from furnaces out onto our slag dump and back again, to get the necessary height, was about ½ mile. The consequent cooling, and pouring of matte into converters through launders caused some of the troubles we had from accretions and excessive amount of cleanings; but the greater diameter of the new converter, larger gas outlet, and better hood are principally responsible for the improvement we have accomplished.

Mr. Howard's exceptional record of long life of linings, I imagine to be partly because he has had the pick of such converter silica as he has wanted to use, whereas we, being in the customs smelting business, have had to take what we could get.

You will thus observe that we have no production records as yet that would be of any interest concerning our new converters, but next year at this time we hope to be able to give a good account of our big converters. Our 10-ft. Peirce-Smith converters consumed about 9,000 ft. of free air per minute at, say, 12 lb. pressure, and the last records I secured before leaving El Paso show that our new 13-ft. converter is taking 20,000 ft., and that means that it is doing business.

## Determination of Dust Losses at the Copper Queen Reduction Works

Discussion of the paper of J. MOORE SAMUEL, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 114, June, 1916, pp. 1079 to 1098.

THE CHAIRMAN (WALTER DOUGLAS, NEW YORK, N. Y.).—Perhaps there is no problem that causes the modern metallurgist more worry than the question of unaccounted for loss. He has not the advantage of the metallurgist of the early days here, who could sleep peaceably with the settled conviction that there were certain copper mineral constituents of the ore which were volatile, and there was therefore nothing to worry about. The slag contents and volatilization accounted for the balance of the copper that went in the top of the furnace after the weighing of the bullion. There have been great strides in recent years toward reducing this loss. Perhaps the Anaconda company is the one we have to thank for the original pioneer work along the lines which we are all today following. We would be glad to hear from some gentlemen present with reference to their experience with this question of dust losses.

E. P. MATHEWSON, Anaconda, Mont.—You mentioned the Anaconda company, Mr. Chairman. I would like to say something about what we have been doing there, and the troubles we have been having. We have a tremendous volume of gas to handle. We adopted the system of large chambers, lessening the velocity of gases, and incidentally cooling the gases. Unfortunately in one sense and fortunately in another, we were compelled to increase the amount of gas handled in the flue system, and the results were then not so good. Still, we get a very good recovery. Our system of taking a sample is similar to the one described by the writer of the paper. We have tried all kinds of apparatus for getting a fair sample. We divided the area of the flue into imaginary squares, and have taken samples for a certain period of time from each square; and have used the asbestos bags. At present, there is a large unit of the Cottrell apparatus almost ready to connect up on the roaster flue to catch the dust from flotation concentrates. The worst things smelters have to handle at the present day, they are coming in the Southwest as well as the Northwest, are of material much of which is as small as 500 mesh, and the ordinary methods of catching dust will not apply to flotation concentrates. The Cottrell process is going to be tried out at our plant for this material. We have devised special forms, with a view to preventing the formation of dust from this material, and we are pleased with the results so far obtained. There is another point in connection with this subject that I want to put before the members here, so they will be thinking about it. You know there has been a great deal of litigation between the farmers

and the smelting companies on account of the damage some smelter smoke causes. In the early days at Anaconda we were turning out a great deal of dust from our small flues and short stacks. That we overcame and we stopped the damage to such an extent that the courts sustained us throughout, and proved that our improvements were really improvements, and we had ceased to materially damage the farmers in our vicinity. The one thing that our good friends of the Bureau of Mines are apt to overlook in checking up smelters is the fact that smelters must be run on a commercial basis. Metallurgy is the art of getting money from ore; and if you cannot get money from the ore, it is not metallurgy. It is a loss. It is not business, either. We have to look out for that when we figure on saving the material in the gases. There is a great deal of good material in the gases that go from smelters, and there is a great deal of material that is not good. There are certain places, as in the center of Germany and England, where it would pay to take everything in the gas and give it a chemical analysis, separating each element and putting it on the market. The best thing to do in handling smoke from smelters is to follow nature's laws as nearly as possible. If you take the smelter smoke and drench it with water, you get a mixture that cannot be deposited in the streams, and there is really nothing than can be done with it. It is worse than if you let the smoke go out, without attempting to purify it. If you cool the gases to a point where it is safe to put them through a bag house, you get sulphur dioxide cooled to a point where it does more harm than if you had let it alone. The thing to do is to find a happy medium, and get the gases escaping at such a temperature that they will go high in the air and diffuse. Gases do not diffuse as quickly as many would have us believe.

S. J. Jennings, New York, N. Y.—There is one point in this unaccounted for loss that has not been mentioned. The ore when received at the smelter is weighed, and in most cases thereafter is dumped on to a bed, sometimes from a greater height than others, but always from some height. When you have the dry condition that obtains in the Southwest with occasional high wind, you are going to have a dust loss before it gets in the furnace. My observations this morning have made me believe that this dust loss is material. I was interested in what Mr. Mathewson had to say. We have in Shasta County, a smelter which treats 800 or 900 tons of sulphide ore a day. It is putting out somewhere in the neighborhood of 350 tons of sulphur daily. All the smoke is filtered so that the gas which is going out of the stacks of the bag house is invisible. The gas that comes from the settlers and converter is likewise led into the bag house, so that there is no visible smoke in the smelter. Yet, some 15 miles away from that smelter, farmers have the imagination to claim that damage is being done by sulphur dioxide, their claim being that sulphur

dioxide forms pistons in the valley of the Sacramento River; when the wind blows with the river current the pistons of concentrated sulphur dioxide travel at least 15 miles and cause discoloration of vegetation. The smelter, on the other hand, claims that there is no material damage done by the sulphur dioxide—that other things cause the discoloration—and the smelter thinks it has proved its case. The material that results from the filtering of the smoke is a most complex fume, containing a great many of the rarer elements—arsenic, bismuth, platinum, copper, silver, zinc and some gold. By means of the electrolytic treatment of zinc, we have found that it is commercially profitable to treat this fume which has accumulated for some 4 or 5 years. The thing to do is to accumulate all material containing values and after a while you will find a process by which a profit can be made. That seemed to be our experience in Shasta County.

C. E. Arnold, Miami, Ariz.—I should like to make an inquiry as to the point raised by Mr. Jennings regarding dust losses. At the International smelter at Miami it is noticeable that the bedding bins are entirely housed in, while those at the Calumet and Arizona smelter are exposed to the action of the winds. It would be interesting to hear from Mr. McGregor whether, in this particular, the design at the Miami plant was influenced by the experiencing of excessive dust losses at the Calumet and Arizona plant.

A. G. McGregor, Warren, Ariz.—Of course, we realized that there must be some losses from the wind blowing over the beds. At the International Smelting Co.'s plant at Miami, we have a very valuable material to handle, and we wished to provide against all losses as far as possible, so we housed in the beds.

The Chairman.—The question seems to be largely a commercial one, that is how great expenditures are justified in order to increase the savings in dust and fume. A plant smelting a low-grade furnace charge can hardly, for the saving obtained, afford to install a process such as that at the International smelter at Miami, where the charge will run from 25 to 35 per cent., without offsetting the savings obtained by an increased charge which would exceed the commercial profit from the operation.

The point which Dr. Ricketts makes, that there may be a considerable mechanical loss of fines from the bed systems in the Southwest during the high winds of Spring, is well taken and it might and probably would pay to enclose the ore beds to obviate or reduce this loss.

The only Cottrell plant constructed in Southwestern smelters is that of the International smelter, and I hope Dr. Ricketts can give us some of the results of this installation.

L. D. RICKETTS, New York, N. Y.—Mr. Chairman, with your permission I will speak about the loss. I thoroughly believe that smelters should not allow their ore to drop through the air exposed to the currents of wind. I think the beds ought to be entirely covered, and a sprinkler used, if necessary. The men need not be where the beds are formed. At the Inspiration, designed by Mr. McGregor, we housed in the beds on account of the rich material—I believe that this was suggested by Mr. Flynn—and we are getting results that lead me to believe that it pays to house the beds.

In regard to Cottrells, I had an article in the *Engineering and Mining Journal* a few weeks ago.* I believe the loss of values in the use of the Cottrell system will be down where they can be regarded as nil. Where we have a loss it is in turning down a converter. As far as the material from the converter stack is concerned, I estimated that we were losing about 50 lb. of copper a day. I believe that the Cottrell system, as an element in the recovery of value, has come to stay.

The dust losses from the reverberatory furnaces with our method of feeding, which you will see when you go to Globe, are so slight that we do not feel justified in putting in treaters to recover the copper from that smoke. I believe this corresponds with the experience with the dust chambers for the amount you will recover. As I said, our most serious loss is from the converter and from the charging and discharging of calcine cars.

## Leaching Tests at New Cornelia

Discussion of the paper of H. W. MORSE and H. A. TOBELMANN, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1593 to 1610.

THE CHAIRMAN (H. W. MORSE).—Gentlemen, for the first time in the history of the American Institute of Mining Engineers, we have a full session on the subject of leaching—especially on the leaching of copper ores. This branch of metallurgy is rather in its infancy. We have had one big plant running for a year or more, and pretty successfully; and another one which will be started in February or March, the New Cornelia. We have another one at Garfield for the Utah company, which is just designed and construction begun; and from several other points come reports of interesting test work. The first paper on the program this evening is one of some of the test work on the New Cornelia ore by myself and Mr. H. A. Tobelmann.

LAWRENCE ADDICKS, New York, N. Y. (written discussion).—The Ajo proposition has had, as I understand it, two great advantages: it has not been obliged to measure leaching against flotation or some other metallurgical process, and it deals with a copper content high enough

* Vol. 102, No. 9, p. 396 (Aug. 26, 1916).

to pay for straight sulphuric acid-iron cementation work if forced back upon such a plan.

These advantages have made it possible to prosecute a consistent policy of development along straight hydrometallurgical lines.

The history of these experiments shows a series of investigations into more or less complicated schemes with a gradual return toward simplicity, the present plan involving utilization of $SO_2$ in regular acid-making practice, precipitation of excess impurities upon the ore itself, final control by cementation and recovery of the copper by the use of lead anodes along more or less copper-refinery lines.

This is about as simple a scheme as could be devised in true cyclical form, and while I think no one will dispute the wisdom of adopting just such a scheme as the first step in large-scale operation, I venture to predict that as experience is gained in actual practice the tendency will be to return to greater complication, as one item after another can be examined separately and tested for cost. The great difficulty in launching a leaching process lies in the multiplicity of interrelated problems which must at first be dealt with jointly. Also in the fact that no two ores seem to present identical problems for solution.

I think this feeling I have outlined is exemplified by the discussion of the reasons for choosing lead anodes as given in the paper and I want to add a few words to this discussion.

The crux of the whole electrolytic problem lies in the mechanism of the oxidation and reduction of iron sulphate. A carbon anode must be thoroughly depolarized or the liberated oxygen will attack the anode itself and the graphite will disintegrate leaving a valueless sludge. In the case of a lead anode, lead peroxide and lead sulphate are formed but the disintegration is very slow as these substances form more or less adherent and conducting coatings. This means that with carbon anodes we impose at once the necessity of dealing with the entire equivalent of ferric sulphate if ferrous sulphate be the depolarizer. With lead anodes the issue can be largely but not wholly dodged through the escape of elemental oxygen. This in turn leads to the necessity of thorough circulation in the case of carbon anodes as against indifferent circulation in the case of lead.

Thorough circulation, however, results in intimate contact between the cathode and the ferric sulphate in the electrolyte with the natural and undesired reaction between these two substances.

Therefore, the price of our increased recovery of copper per kilowatt-hour is the obligation to keep liquors running more than some 0.2 per cent., and preferably much less, in ferric sulphate, away from intimate contact with the cathode, which means either a diaphragm or an efficient utilization of $SO_2$ or other reducing agent.

Everyone who has advocated the first remedy has come to grief, except

possibly Hybinette in his two-level system in copper nickel refining, and everyone who has wrestled with the second problem has found that there are intermediate and as yet little-understood reactions to be dealt with. Nevertheless, I believe that as continued operation gives experience and opportunity for experiment, a way out of these difficulties will be found and the carbon anode yet succeed in the practical electrolysis of sulphate solutions.

G. D. Van Arsdale, New York, N. Y.—Both those responsible for the tests of which the results are given in this paper and the Institute are to be congratulated; the first for the successful results finally obtained on a difficult and new problem, and the profession on the liberal policy which made possible the publication of this very valuable summary of practical leaching data.

The thoroughness and care with which this work has been done may be judged from the time, nearly 4 years, during which these experiments have been carried on.

It is especially gratifying to the writer that some of the main points developed during the series of experiments on leaching by Phelps, Dodge & Co. at Douglas have been entirely borne out by this work; the first of these being that ferric iron can be controlled by the use of sulphur dioxide, the second that solutions that had previously been considered entirely too "foul" as an electrolyte for copper deposition can safely be used, and the third that by the use of depolarization the anode problem is entirely solved, either by lead or graphite, both of which had previously been considered as impossible for electrolysis of sulphate solutions of copper.

The paper is particularly valuable in giving a discussion of the conclusions leading to the adoption of the final process by the New Cornelia company and as bearing on their choice of lead instead of graphite as an anode material. My reason for discussing, and to some extent differing from, these conclusions is that I feel that from the results of Phelps, Dodge & Co.'s work it is necessary to emphasize the fact that, while the choice of lead in this case may be undoubtedly justified, yet there are conditions under which graphite should be selected and these conclusions therefore should not be taken as general conclusions against its use under other sets of conditions.

Graphite, which was first used under the writer's direction on a considerable experimental scale as an anode material for this kind of work, before this time had been considered as absolutely unsuitable for the electrolysis of sulphate solutions of copper. I have a letter from the manufacturers in which this statement is made, and the same thing is stated in Greenawalt's book on copper leaching. There is no doubt that its use has certain practical disadvantages, the principal one of these being that a tank-room ventilation system will probably be needed to enable it to be used. While this means a radically different tank-room

design, it need not mean anything at all impracticable or leading to increased tank-room costs beyond the negligible amount of power for ventilation. On the other hand, its use has very considerable points in its favor which in my opinion outweigh those against it, generally speaking.

Regarding its durability, I believe it is entirely safe to say, from the results of careful tests made by Phelps, Dodge & Co., that under proper and easily controlled conditions its depreciation costs, or in other words the amount of disintegration, per pound of copper produced will be negligible.

The ampere efficiency of graphite is, as stated, badly reduced by a ferric iron content of electrolyte, with which still fair efficiencies may be had with lead, this meaning as stated the maintaining of a limit of about 0.2 per cent. ferric iron for graphite as against a limit of about 0.4 per cent. with lead. It should be carefully noted, however, that as regarding the amount of power per pound of copper produced a reduced ampere efficiency at a voltage of about 1 volt does not mean nearly as much as the same reduction with a voltage of over 2 volts. In other words, one of the very considerable practical advantages of graphite is the much lower voltage at equal current densities with lead. Regarding the ampere efficiency obtainable with graphite, it is possible that a casual reading of the paper under discussion might lead to the conclusion that a low ampere efficiency was characteristic of graphite. That this is not true was proved conclusively by Phelps, Dodge & Co. in a series of tests last summer which demonstrated that it is commercially practical to keep the ferric iron under 0.2 per cent. and under these conditions to obtain fully as good or better efficiencies than those cited in the present paper, with yields over a considerable period of 3 lb. of copper per kilowatt-hour at a current density of about 12 amp. per square foot, which it should be noted is 76 per cent. in excess of the average current density as given in the table on page 1600 of the paper under discussion. With the lower current densities as given in this paper, our pounds of copper per kilowatt-hour would have been still higher.

It is, of course, not reasonable to expect that a high $SO_2$ reduction efficiency will be obtained if the $SO_2$ is blown out of the solution before it has had time to act. In the worst case, this factor of the time needed for the reduction of ferric iron by $SO_2$ means a large, though not at all prohibitive, storage capacity, and it may be stated that one of the results of our work at Douglas last summer was to demonstrate that the ferric iron can be kept within our 0.2 per cent. limit by a storage of a portion only of the main solution.

It is true that the anodic efficiency of graphite is as stated on account of air agitation over 100 per cent., that of lead being 30 to 35 per cent. Expressed in another way, however, this means, with anything like a

reasonable $SO_2$ reduction efficiency which can be had as we have shown, for practical purposes that the amount of acid regenerated by lead is much less than with graphite. Now if one has an acid plant already built or ordered or intends to erect one, the comparatively small amount of acid produced by the use of lead anodes is not so important, but where this is not the case this considerable extra amount of acid produced by the use of graphite is a considerable practical advantage in its favor. Where, as in this case, an ore does not require more than 3 lb. of acid for leaching per pound of copper produced, the amount of acid regenerated by the use of the graphite system will be ample for leaching purposes. A comparison, therefore, between the two systems should include the cost of an acid plant against lead as regards installation costs, and, as regards operating costs, the extra cost of acid needed together with freight if brought from a distance. It seems to me, therefore, that the statement in point 1 of the summary on page 1607 should include this factor.

Without intending criticism of the decision made in this particular case, I believe, nevertheless, that the following comparison as to the relative merits of lead and graphite will be true and more generally applicable than those given.

1. Installation costs for the two systems will be about the same, except that, where an ore requires more than 1½ lb. of acid per pound of copper produced, the additional costs of an acid plant plus railroad equipment for carrying it when produced at a distance from the leaching plant, plus storage tanks, piping, etc., at the leaching plant, must be charged against lead.

2. Careful determinations extending over a considerable period have shown that the disintegration of graphite and its consequent depreciation per pound of copper produced is negligible. The prevention of graphite anode disintegration means simply maintaining the conditions necessary for proper depolarization. When this is done graphite will not disintegrate appreciably, but if these conditions are not maintained it will go rapidly. Exactly the same thing is true for lead, but, since the anodic efficiency of lead is only one-third that of graphite, the disintegration of lead anodes under the same service may be expected to be at least as rapid as graphite.

3. Peroxidized lead detached from an anode will be lost, and the scrap value of partly peroxidized and badly corroded lead anodes, when high freight rates are considered, will not be very high. Furthermore, when the slow rate of disintegration of graphite anodes finally reduces their thickness very appreciably, they can be reassembled and the scrap loss reduced thereby to a small amount.

4. The power required for precipitation with lead is at least three times that for graphite under the same conditions.

5. More than twice as much acid is regenerated with graphite as with lead. This, as stated above, means not only an extra installation but also an extra operating cost for this additional acid needed, when lead is used.

6 and 7. Since the anodic efficiency of graphite is very high, higher circulation and some form of agitation in the cells are required. In this connection it may be stated that, although air agitation has been used in the work at Douglas with good results in some respects, I have never been in favor of this method of agitation as compared with others, unless more than 3 lb. of acid per pound of copper are needed. The excess anode efficiency over 100 per cent. is of course mainly due to the oxidation caused by the air, and it is clearly illogical to use an oxidizing agent for mechanical purposes where reducing conditions are wanted.

8. The extra storage capacity needed for the time of reduction by $SO_2$ as shown above need not be more than a fraction of the total solution bulk. Furthermore, if a good $SO_2$ reduction is obtained, the tank-room ventilation difficulties are correspondingly reduced.

9. and 10. Since the anodic efficiency of graphite is three times that of lead, it of course necessarily follows that larger tower capacity and greater volume of solution sent to these are required. However, the saving in installation by not needing an acid plant will pay for this and the extra storage capacity needed under the preceding paragraph at least several times over.

11. The point of the necessary tank-room ventilation has already been spoken of.

12. I must disagree absolutely, so far as my experience goes, with this conclusion as to tank-house capacity being necessarily larger with graphite than with lead. While the cost of power is an important factor in this connection, we have not considered for our requirements that the low-current densities, averaging 6.8 amp. given in the paper, are economical under our conditions, and the figures we have obtained with current densities around 12 amp. per square foot, in which average ampere efficiencies at this higher density fully equal to or better than those given in the paper and obtained over considerable periods appear to throw this comparison decidedly in favor of graphite.

13. It is a little difficult to see just why operation with graphite anodes will be any less "fool-proof" than with lead. Granted that the conditions for keeping the ferric iron below our limit are known, I cannot see any reason whatever for requiring any more care or skill for maintaining this limit than for maintaining an equally vital one differing from this by only 0.2 per cent.

My opinion, not expressed as a criticism of the decision made in this particular case, but as a general one, is that the advantages and undoubted lower operating costs of graphite decidedly outweigh those

with lead. In other words, it would only be considered advisable under very exceptional conditions to make the very large additional investment required for an acid plant over and above the acknowledged nearly equal costs of the two systems, and still with this extra investment to obtain a higher operating cost.

The Chairman.—I would like to say that Mr. Van Arsdale should be thanked for a good many discussions and arguments, and that the conversations and arguments with him were of very great advantage to us in this work at the New Cornelia; and in a good deal of what he says I would agree. It is highly probable that we will be able some day to work out the use of graphite to make a saving in power cost, and to generally improve the process. As far as we could see from the test at the New Cornelia, this was going to be a difficult undertaking, so we decided to put it off for a few years. I think we will later come back to large-scale tests on the use of graphite. Certainly the gain to be expected when this has been worked out is a great one. I, personally, was sorry to stop the research work on the graphite.

S. J. Jennings, New York, N. Y.—I would like to ask for some information. I see that this paper has considered merely two kinds of insoluble anodes, lead and graphite. I understand a considerable amount of work has been done with other insoluble anodes and I would like to ask if tests have been made at New Cornelia with any other insoluble anode such as fused magnetite.

The Chairman.—We have made none at Cornelia, except with lead and graphite, and a mixture of the two—that is, coke cast in lead.

F. S. Schimerka, Clifton, Ariz.—In connection with the paper presented, I wish to ask the question why the process of precipitating the ferric iron in a neutral liquor on fresh ore was not finally employed and preference given to the introduction of sulphur dioxide to effect the elimination of trivalent iron?

In regard to the use of lead anode sheets, I would like to know whether lead-covered iron grates would be applicable, and I ask this question because I remember that I have seen such anodes employed in liberator tank practice but have not been able to collect data concerning their efficiency compared with that of solid sheets used for the same purpose.

The Chairman.—That was the basis for starting the large plant. In the 7-ton plant and in the 40-ton plant, at first, the ferric iron was precipitated, but later for an unknown reason we got no such result, the ferric iron going up and the ampere efficiency going down.

C. G. Grabill, Matehuala, Mex.—I would like to ask if it was possible that the difficulties arose from the depth of ore in the two tanks.

In the first one you have a small volume of solution; and in the second, you have a very much larger body of solution. In the first condition, with the small volume, you have much more air present and oxidation taking place. If the charge is increased—rather, if the percentage of the solution is increased—that condition does not obtain in the latter part of the action. I would like to ask if you have done any work along that line—to investigate that point?

The Chairman.—That is one question we left to find out in the 5,000-ton plant. Mr. Flynn, have you anything to say?

F. N. Flynn, Clifton, Ariz.—The leaching that we did at Clifton, for something over 20 years, was the simple process of an acid leach, with iron precipitation, and the wasting of all liquors. The modern leaching which you are about to attempt is something which you have developed and proven to your entire satisfaction and until such time as it is working out, practically, I don't feel that I would have any criticism to make. I don't feel competent to criticize. It does seem to me that you are working in the right direction. The only question that comes to my mind is the percentage of liquors you will have to throw away. That, I understand you have worked out to your entire satisfaction.

The Chairman.—I think that 6 months' operation of the big plant will make us all feel more confident than we do now.

Stuart Croasdale, Denver, Colo. (communication to the Secretary*).—I think the New Cornelia ores at Ajo may be considered as typical of all copper ores in the United States that are amenable to acid leaching in a raw condition, since all known ores of this kind contain a certain amount of soluble iron oxides and alumina which will be a contending factor in electrolytic precipitation of the copper.

It is interesting to observe that the results obtained by the authors have verified in almost every instance the predictions made before they started, yet their work has been so efficiently and thoroughly done that it forms a most valuable contribution to the hydrometallurgy of copper, and particularly to the section on electrolytic precipitation. They are to be congratulated on having such an opportunity to definitely advance our knowledge so much in this direction.

I regret that one thing more was not tried, at least on a laboratory scale, and that is, roasting the ore before leaching. In the Ajo ores the iron exists in all forms of oxidation from chalcopyrite to ferric oxide. The processes of nature have been so gentle that much of this iron is as easily soluble as freshly precipitated hydroxide. Aluminous compounds in the rock also exist in all stages of kaolinization and are likewise soluble in dilute acids. A slight roast would oxidize all of the iron, including the

* Received Sept. 29, 1916.

residual pyrite, to ferric oxide and would dehydrate the aluminous compounds and salts to the oxide. Since ignited ferric and aluminum oxides are not readily soluble in dilute acid, it would seem that the lixivium obtained from roasted ore would be practically free from deleterious salts, which would obviate the necessity of depolarizing and "bleeding." Assuming this to be true and the solubility of the copper oxide not diminished, the cost of roasting would be compensated by the recovery of the copper now held in the tailings as sulphide, which is likely to be in ever increasing quantity, and as soluble copper retained by the argillaceous material; also by the expense of roasting pyrite to produce sulphur dioxide, and by the expense of precipitating part of the copper on scrap iron at each cycle of the lixiviant.

## The 2,000-Ton Leaching Plant at Anaconda

Discussion of the paper of FREDERICK LAIST and HAROLD W. ALDRICH, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1281 to 1293.

F. N. FLYNN, Clifton, Ariz.—I would like to ask Mr. Mathewson what percentage of his leaching liquor is wasted at this time? It has a bearing on the question in connection with the New Cornelia, and I think it is a very important question.

E. P. MATHEWSON, Anaconda, Mont.—I am not sure what the percentage is now, but we don't discard the strong liquors. Something like 25 per cent. of the wash water taken out each day is sent over a special set of scrap iron tanks, and the copper recovered there. We had a little trouble at first. We did not proceed to use the sponge iron for the reason that the plant we originally contemplated was abandoned for flotation. Our original idea was to use a leaching process on all the tailings of the mill, but we found it would take a great deal of time to build a plant; and the cost of operating the plant we estimated to be about the same and the recovery the same. It was a question of time with us, and we decided to adopt flotation. That changed our plans. We now have the leaching plant, treating the tailings of the old dump, and the flotation applied to the current tailings. The sponge iron was figured for the large plant. We can get all the scrap iron necessary for the small plant—plenty from the scrap produced in the main plant of the smelter.

## Possibilities in the Wet Treatment of Copper Concentrates

Discussion of the paper of LAWRENCE ADDICKS, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1565 to 1573.

F. N. FLYNN, Clifton, Ariz.—As a number of my associates in Arizona know, for a great many years, I have felt that leaching was one of the coming problems. We are about to start experiments at Clifton, along the idea of leaching wet flotation concentrates direct, without roasting—differing in this respect from Mr. Addicks' suggestion. We are prompted in starting these investigations by the difficulty in transporting and handling flotation concentrates, and consideration for drying and roasting dust losses. I have had the idea for 5 or 6 years past, that those very fine concentrates could best be treated by leaching at the concentrating plant, thus eliminating our transportation difficulties to the smelter, and dust losses at the smelter. Given flotation concentrate, if it must be dried and roasted, it would, in our case, undoubtedly be cheaper to smelt the calcine, but if it can be leached raw and wet and the drying difficulties and dust loss eliminated, it would be very advantageous to leach it at the mill. I realize that it will be a difficult problem, and one which may take several years to develop, but if developed, it would pay for many years' expense of investigation, and we have been encouraged by our general manager to investigate its merits.

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## General Subject of Leaching

Discussion at the Arizona Meeting, September, 1916.

THE CHAIRMAN (H. W. MORSE, Los Angeles, Cal.).—I would like to open this meeting for a little while to the general subject of leaching. We ought not to hold back if we have any new schemes for the future. I know that those of us who are tangled up with this leaching work will appreciate any suggestions that will make us work harder and scheme harder to pull through these past difficulties on leaching.

F. S. SCHIMERKA, Clifton, Ariz.—Under the direction of J. W. Bennie, General Manager of the Shannon Copper Co., Clifton, Ariz., it has been my valued privilege to coöperate in the elaboration of a hydrometallurgical process aiming at the recovery of copper in mill tailing resulting from a gravity concentration of a low-grade semi-oxidized sulphide ore occurring in a porphyritic gangue.

The experimental work which has been carried on with a 25-ton plant is completed, and the company is building at present a leaching plant on a larger scale, the first unit consisting of a 150-ton roasting furnace and accessories for leaching the calcine.

I wish to lay before you only the most essential points of our process, the outstanding feature of which is the non-application of acids or any other chemical in the leaching operation proper. After a sulphatizing roast conducted under well-defined conditions in a mechanical roasting furnace of the multiple-deck type, the calcine is treated with water only. The separation of the copper liquor will be effected by decantation, and the copper, at least for the present, precipitated by scrap iron. The character of the tailing, which is highly basic and remains so even after roasting, prohibits the application of sulphuric acid, which is consumed to the amount of more than 10 lb. for each pound of copper dissolved from the calcine. Our mill tailing, all of which passes through a 1-mm. opening, contains an average of 20 lb. of copper to the ton, 1 per cent., we may say. Fifty-five per cent. and more of the total copper is present in oxidized condition, mainly as malachite and azurite, the balance is chalcocite. Sulphur is present in the amount of 1 per cent., mostly as pyrite. The precious metals are practically absent, and that no attempt at their recovery need be made has materially simplified our working procedure.

The manner in which the roasting of the raw tailing is conducted is, of course, vital to the process, there being no other means employed to convert most of the copper into water-soluble sulphate than the agencies of heat, oxygen and sulphur during the passage of the material through the furnace. A close regulation of the temperature in the roaster by means of pyrometer control is essential. Tests have shown that the re-formation of water-insoluble basic copper sulphate and oxide takes place when the temperature rises closely to 900°F., and we aim at keeping the temperature on the hottest floor between 830 and 860°F. We have encountered no difficulty in accomplishing this. It is, however, necessary to limit the amount of sulphur in the charge to a quantity which in an empirical way has been found to produce the best results, and which excludes the possibility of local overheating, as would take place by the oxidation of a large amount of pyrite. Moreover, any excess sulphur will, by formation of sulphur dioxide, decrease considerably the oxidizing effect of the hot gases striking through the furnace. Our roaster gases contain less than 0.1 per cent. of sulphur dioxide when issuing from the stack. For these two reasons, we aim at keeping the charge as poor in sulphur as we can afford to do without injurious effect upon results. Taking the copper contents in the tailing as a basis, we adjust the sulphur in the charge to 1½ lb. for each pound of copper in the tailing. This necessitates the addition of ½ per cent. of sulphur to the charge in the form of iron pyrite.

For fuel we employ oil burned in an external firebox, which is air-jacketed to prevent loss by radiation. The fire gases which, mixed with the required volume of air admitted through openings in the firebox, at

the entrance into the furnace, are kept at 800°F., are sent into the 10-deck roaster on the ninth floor. We do not employ muffles. The fuel consumption in the 25-ton roaster was 5 gal. of oil per ton of tailings treated, frequently less, and we are confident that we can decrease it to 3 or 3½ gal. at the most in the large roaster on account of heat-saving devices incorporated in its construction. Also, there will be another source of heat provided by the exothermic reactions that take place during the roasting process in the furnace. The additional heat derived from these reactions is quite considerable and has made itself felt to an appreciable degree in the small pilot furnace.

In the new plant, the separation of the copper liquor from the leached calcine will be effected by counter-current decantation in three Dorr thickeners. Directly from the roaster the calcine is discharged into a circular mixing tank where it receives return liquor from one of the thickeners. From the mixing tank the pulp and liquor pass into three Dorr classifiers placed in series to effect a separation between the sand and slime. The slime in the overflow from these classifiers passes successively through the three thickeners, the first of which delivers its clear overflow to the precipitation boxes. As the pulp progresses through the battery of thickeners it is washed by water and waste liquor from the precipitating launders traveling in the opposite direction.

During our test runs with the 25-ton roaster, the extractions ranged from 65 per cent. to 77 per cent. of the total copper in the tailing. For the present the copper in the liquors is precipitated on scrap iron in a system of launders, although electrolytic separation is taken into consideration by the management for the future. Concerning the electrolytic deposition of the copper and a decision to introduce it in our plant, we are looking forward to the results which will be obtained by this method at Ajo in the New Cornelia Copper Co.'s plant now under construction, where the process has been worked out by such pioneers as Dr. Morse and Mr. Tobelmann, who have presented their paper on this subject before this meeting.

B. B. GOTTSBERGER, Miami, Ariz.—For some time we have been experimenting on the recovery of oxidized copper. While trying to convert the oxidized copper to sulphide by means of calcium sulphide and sulphuric acid, we concluded that what conversion was obtained, resulted from the solution of the copper in acid and its subsequent precipitation as sulphide. Laboratory work demonstrated that a process along these lines could probably be made to work, but desiring to avoid the use of hydrogen sulphide, if possible, we concluded to look for another method before trying this scheme on a working scale. Early in the present year we did some laboratory work along the lines of the process we have just seen at Chino and at the present time are trying out this

idea in an experimental plant handling about 50 tons per day. I am very hopeful of a successful result from this work.

H. E. Williams, Calumet, Mich.—The Calumet & Hecla Mining Co. has been experimenting for some time, under Mr. Benedict's direction, upon the problem of reclaiming copper from the old tailings banks in Torch Lake, which contain about 40,000,000 tons of recoverable sands. After many laboratory and test-plant experiments, they have finally adopted the ammonia leaching process, and have built a 2,000-ton plant, which is now in partial operation.

The plant has eight unlined steel leaching tanks 54 ft. in diameter by 12 ft. high, provided with the usual wood grating, cocoa matting and canvas filters, removable steel covers and one central and six circumferential discharge gates. After being reground and treated on tables, the tailings are pumped into the plant and deposited in the tanks by revolving distributors. The leaching solution contains about 1.5 per cent. of $NH_3$ and an equal percentage of $CO_2$, and after dissolving the copper, is pumped to the still house, where it is treated in ammonia distillation apparatus with low-pressure steam which precipitates the copper as oxide. This is sent to the smelter, while the $NH_3$ and $CO_2$ distilled off and condensed is pumped back to the leaching plant to be used again. From sand containing about 10½ lb. of copper to the ton, we recover about 8 lb. We have had to develop a roughing still, which has delayed the still-house work, otherwise the plant would now be operating at its rated capacity. I am sorry I cannot give you a description of the chemical reactions; but I am not a chemist and must leave that for Mr. Benedict in the future.

G. D. Van Arsdale, New York, N. Y.—I would like to ask Mr. Mathewson the reason he adopted wood tanks instead of concrete tanks.

E. P. Mathewson, Anaconda, Mont.—We had experimented with wooden tanks, and when we put up the 1,200-ton tanks, we thought the wooden tanks would be satisfactory. After they had been in service for about a year we noticed deterioration, and we had them lined with lead. If they had been built of concrete, they would have been more expensive. For a plant the size of ours, we think the wooden tanks, lined with lead, are preferable; but with larger ones, we think it would be preferable to build them of concrete.

## Petrography of the Mount Morgan Mine, Queensland

Discussion of the paper of W. E. GABY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1725 to 1744.

L. C. GRATON, Cambridge, Mass.—May I ask, in connection with one point in the summary, to what extent the author regards actinolite equivalent to tourmaline as an index of high-temperature and presumably magmatic conditions of deposition of the accompanying sulphides?

W. E. GABY.—I merely advanced the idea that the mineral actinolite might have some such significance. In the copper deposit at Braden in South America, the ore occurs in an andesite breccia which is cemented by a matrix of sulphides, quartz, and tourmaline. Tourmaline is a high-temperature mineral, and when we find it associated with ores, these are regarded as having come from a volcanic or deep-seated source, hence a primary hypogenetic deposit. Thus we can count on greater continuation of the ore with depth than in some other kinds of deposit, and I thought that the actinolite, being also a high-temperature mineral, might have some such bearing in the case of Mount Morgan. I did not advance this idea as a final conclusion from a study of such phenomena, but thought that it might bear some discussion in view of the fact of the peculiar occurrence of this mineral in a notable orebody. It also occurs with the copper ores at Ducktown, Tenn., and in both cases may only be due to general regional metamorphism.

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## Geology of the Warren Mining District

Discussion of the paper of Y. S. BONILLAS, J. B. TENNEY and LEON FEUCHÈRE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1397 to 1465.

IRA B. JORALEMON, Warren, Ariz.—This paper has covered the situation so well that I have few suggestions to offer. Two points, however, are shown more clearly in the lower mines of the Calumet & Arizona Mining Co. than in Copper Queen workings, and may be worth mentioning. The first is the usual association of high-grade primary sulphide orebodies with much larger masses of pyrite and silica, averaging less than 1 per cent. copper. The richer areas occur sometimes around the borders of the lean pyrite bodies, and sometimes entirely within them. While as yet we have not been able to find any certain reason for the location of high-grade sulphides in low-grade masses, it seems likely that the age of fracturing is one controlling feature. The first stage of mineralization, with pyrite and silica, had the effect of sealing up the rock. The later solutions, richer in copper, could pene-

trate into the low-grade masses and bring in the rich copper sulphides only along fracture zones which were kept open by a more or less continuous motion during the time of mineralization. Fractures of the same age as the mineralization are therefore most favorable indicators of rich primary orebodies.

The second point I should like to emphasize is the very irregular pre-Cretaceous topography. This is best shown by drill holes east and south of the productive area. In the basin between Lowell and Warren, about 5,000 ft. east of the Briggs Shaft, drilling showed from 500 to nearly 1,000 ft. of Cretaceous conglomerate, filling an old canyon in the Carboniferous limestone. In the large valley south of Warren there was a far greater canyon. A drill hole about 4,000 ft. south of the Country Club, or 6,000 ft. south of the crest of the hills below Warren, was sunk to a depth of over 3,200 ft. without reaching the bottom of the Cretaceous conglomerate. This is the deepest diamond-drill hole ever put down in the Southwest. From the top of the Carboniferous limestone hills south of Warren to the bottom of this drill hole there is a difference in elevation of nearly 4,500 ft., in 6,000 ft. horizontally. Allowing for erosion of the hills and for considerable probable depth of conglomerate below the bottom of the drill hole, the total depth of the pre-Cretaceous canyon must have been 7,000 or 8,000 ft. This is pretty nearly the record canyon in history. With such a varied land surface, the pre-Cretaceous oxidation may have reached a level far lower than the present deepest mine workings in the Warren district.

F. L. Ransome, Washington, D. C.—There are a few points in the excellent paper just read upon which I should like to make brief comments.

The authors correlate definitely the thin quartzite at the top of the Abrigo at Bisbee with what I have named the Troy quartzite in the Globe-Ray region. I have myself suggested this correlation in a recent paper but think that more work is needed before it can be concluded that they are the same.

A very important and interesting feature brought out clearly in the paper just presented is the relation of oxidation and enrichment to the former Cretaceous surface. In my report on the Bisbee district published many years ago I stated that a good deal of the enrichment had probably taken place before the deposition of the Cretaceous Glance conglomerate. The underground workings, however, were not extensive enough at that time to show how regularly the zones of oxidation and enrichment dip south in conformity with the general dip of the basal Cretaceous beds.

The statement made in discussion by Mr. Joralemon regarding the great apparent thickness of Glance conglomerate found in drilling in the plain south of the Bisbee hills should be considered in connection with

known overthrust faulting in that general region. The conglomerate may be both faulted and tilted, so that a drill section is not necessarily a measure of original thickness.

The observations made by the authors on contact metamorphism at Bisbee agree substantially with my own results. Although the Bisbee ore deposits are generally regarded as of contact-metamorphic origin they can hardly be considered as typical examples of that class. Such metamorphic silicates as are present are generally small or microscopic.

L. C. GRATON, Cambridge, Mass.—When the summer session of the Institute was held in Butte 3 years ago, we were given a paper by Mr. Sales on the geology of that remarkable district which may, I think, properly be regarded as constituting a new milestone in the advance of the application of geology to mining, in that it represented the accumulated data and evidence and speculation that had resulted from systematic, intensive and constant study by the geologic staff of the Anaconda company for a period of some 15 years. Obviously, such advantages for observation, interpretation and verification afforded to conclusions a validity that could have been secured in no other way. Without question, Mr. Sales' paper will continue to be the standard authority on the Butte deposits for a long time to come, and even when the horizons on which its observations were based may have become exhausted and abandoned, the principles which it disclosed will undoubtedly be found in large part applicable to the deeper regions to which operations may extend.

The present paper, dealing with a district of such extensive development, such great production, such rich ores and such complex deposits as Bisbee, is certain to be of interest to all engaged in the study of mining geology. Like the paper on the Butte deposits, this one also is the product of systematic and steady effort for a number of years by the members of a company geologic staff, aided by the coöperation of their colleagues in the district.

Of the many directions in which this account of Bisbee geology merits attention, the feature that perhaps impresses one most strongly concerns the method, or at any rate, one of the methods, by which the results were achieved. Without in any way minimizing the difficulties of geological accomplishment at Butte, it may be said that, owing to the essential monotony of rock character and to the nature of the fractures in and along which the ores have been either deposited or dislocated, the outstanding problem in that district is of a structural kind. In Bisbee, the variety of rocks and the consequent variations and irregularities of the channelways that traverse them afford an ample supply of structural problems which the geologist must solve. Yet superimposed on these are problems of another sort and of great complexity; they are

essentially chemical or mineralogical in character and relate not only to the initial deposition of the ore but to its subsequent alteration and to the enrichment or dispersal of its metal content. Problems of this kind have to be attacked by different methods from those employed for the solution of structural problems. The method which has been found most effective depends upon the microscope. No reader of the paper by Messrs. Bonillas, Tenney and Feuchère can fail to note the extent to which they have utilized the facts revealed by microscopic examination.

In this respect, therefore, it seems to me that this contribution marks still another step in the progress of scientific ore-finding. For although the method employed has been used by others, this, so far as I am aware, is the first recorded instance in which field methods and laboratory methods have been systematically coördinated on a comprehensive scale under the advantageous opportunities for constant and extended observation that are open to the geological department of a large mining company. The authors certainly deserve our congratulations and our thanks, and I think the Copper Queen company and other companies pursuing the same policy are to be commended for their far-sightedness in recognizing as valuable and permitting their geologists to follow methods of attack which some of narrower outlook might regard as too "flossy" and scientific to be of any practical service. Much is undoubtedly to be expected of such a practical, commercial application of this combination of methods which has received so gratifying a start in the present paper.

In connection with one particular detail covered by the paper and referred to by Dr. Ransome—the subject of contact-metamorphic ore deposits—I should like to express assent to the statement of the authors that no sharp dividing line can be drawn in Bisbee between the type of deposit which nearly everyone would probably agree should be called a contact-metamorphic deposit and a type which lacks many of the characteristics that one expects a contact-metamorphic deposit to possess. In other words, any boundary must be drawn arbitrarily; and perhaps no two people will draw it in exactly the same place. Dr. Ransome no doubt had this idea in mind in the discussion he gave to the subject in his report on the Bisbee deposits and also in his present remarks when he speaks of *typical* contact-metamorphic deposits. The question immediately arises, however, as to what is a typical contact-metamorphic deposit. Shall we regard as typical only those deposits in which the effects of igneous metamorphism have been most intense or extreme? Or shall we call typical those contact-metamorphic effects which are not so extreme and are met with more commonly? For in studying a number of districts in which contact-metamorphic deposits are involved, my associates and I have been impressed more and more with the fact that the situation in Bisbee is neither unique nor unusual.

It would appear that what is now called contact-metamorphism is simply a group of effects that stands at one end of a continuous series of effects, and that agreement must be reached as to where one shall stop in that series and say: "Contact-metamorphism ends here." Indeed, as evidence accumulates, it is somewhat doubtful in my own mind whether much is to be gained in drawing such a sharp division line. Plainly, here in Bisbee, the geologists have found no reason, in fact, have found it impossible, to draw a sharp line. It is a question in my mind whether we should not either broaden and improve our definition of contact-metamorphism in connection with ore deposits, or drop it as a significant term of practical application; for present usage is likely to lead to ambiguity and confusion and to become a cloak for loose and inaccurate thinking and expression.

THE CHAIRMAN (G. F. G. SHERMAN, Bisbee, Ariz.).—As Mr. Graton has spoken of the attitude of the companies with regard to geologists, I think we should disclaim some of the credit which he gives us. While we believe that they have been of great assistance to us, we must say that geologists did not find all the ore. We had a great deal of ore before we had geologists. A great many orebodies are found by the miners and many are stumbled on by accident. But in this complicated country, we shall always need as much assistance and information as we can get from whomever we can secure it.

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## Coöperative Effort in Mining

Discussion of the paper of JOSEPH P. HODGSON, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 113, May, 1916, pp. 867 to 869.

JOSEPH P. HODGSON.—As to "Safety First," I did not deal with that, but I will say that in 1913, the number of men killed was nine, and that year the "safety first" program was commenced; the next year, 1914, the number was reduced to five. In 1915, I am pleased to say that the number was reduced to two in the Copper Queen's entire property, and we have none so far this year.

THE CHAIRMAN (G. F. G. SHERMAN, Bisbee, Ariz.).—I can say, further, that we have from eight to nineteen men per month incapacitated for more than 14 days. Those are classified in this State as serious accidents. The number of men injured has apparently increased, but that is due to our improved method of recording and classifying accidents, no matter how slight.

C. W. GOODALE, Butte, Mont.—The importance of coöperation among the members of the staff of an industrial organization is well brought out by Mr. Hodgson's paper.

The Anaconda Copper Mining Co. has done something along this line, but has not gone as far as the Copper Queen Co. At the Anaconda works, on the last Tuesday of every month, the manager meets the heads and foremen of all departments, for the purpose of discussing costs and efficiency. The superintendents are asked to give the reasons for any changes in costs, and also to suggest methods by which greater efficiency may be obtained. If any member of the staff has been away visiting other plants, he gives an outline of his observations, and then a general discussion follows.

In the efforts of the company to improve working conditions and effect a reduction in the number of accidents, coöperation is of the greatest importance—not only among the foremen and shift bosses, but among the miners and smeltermen.

When the Bureau of Safety was established, about 3 years ago, our first step was to make an analysis of causes of accidents, with a view to attacking the principal causes with special vigor. A review of accidents in the Butte mines for the preceding year showed the following figures:

| | Per Cent. |
|---|---|
| Fall of ground | 43.21 |
| Caught by mine cars | 12.97 |
| Handling tools | 14.54 |
| Handling material | 11.87 |
| Falling | 10.00 |
| Falling timber | 3.62 |
| Machinery | 1.11 |
| Around cages | 0.74 |
| Explosives | 0.70 |
| Miscellaneous | 1.24 |
| | 100.00 |

In the meetings of the safety engineers and foremen these figures were considered, and the fight against accidents began. It was soon found that in order to make the miners coöperate in this movement, we should have to inflict some penalties on them for failure to look out for their own safety. For instance, when men are found working under conditions which are unsafe, and which they can remedy, they are laid off 7 days, and for the second similar offense the penalty is a layoff for 14 days. For the third offense they are discharged.

We have effected quite a satisfactory reduction in accidents in the mines of the company. During the year 1915, we had 1.42 fatal and serious accidents per 10,000 shifts, and 1.09 in the first half of 1916—a reduction of 23 per cent.—and this in the face of a mine fire which occurred in February, 1916, and which caused the loss of 21 lives.

In the reduction works our efforts have not accomplished so much. Perhaps one explanation is that the causes of accidents in reduction works

are more varied in character than in the mines, and it has been difficult to attack any one particular cause with special emphasis. Furthermore, it should be stated that a large amount of construction work has been going on during the last 3 years, and I think it is generally admitted that there are more risks in such work than in operation—particularly when construction and operation are going on together, as has been the case in some of our departments.

At the Anaconda works the company has spent more than $60,000 in safeguards which have been suggested at meetings of the Safety Committees, and the effect will doubtless be shown in accident records as time goes on.

Charles A. Mitke, Bisbee, Ariz.—The practical man's lack of opportunity presents a difficulty to his satisfying his interests in technical subjects. The object to be attained in introducing this course was to show that there is no dividing line between the so-called practical and technical sides of mining; that if any ideas, devices, methods or applications are practical, then they are also technical because all practical methods have a technical basis. The converse is also true that all technical methods which have commercial value and are worth introducing are practical and have a practical basis. However, the terms practical man and technical man still exist, and it is only by coöperation of both classes that the best results can be obtained through their combined coöperative effort. Four years ago none of our shift bosses or foremen would attempt to write or read a paper at any meeting. However, after certain members of the operating staff presented papers, then they were also induced to prepare papers. These men were given all possible assistance. Their subjects were first discussed with them individually so as to bring out the important points and then were written and revised to emphasize these points and in their final form presented at the meetings. Two courses were pursued during the past year—a general course at the mine foremen's dinners and a definite course in mining methods, etc., at the Copper Queen mining conference. An example of subjects taken up in the general course was a lecture on "The Importance of Keeping up the Grade of the Ore," while an example of the subjects used at the conference was the discussion of "The Details of Doing Development Work." At mining conferences a constant effort was made to bring out the basic principles which govern the varied mining operations here as well as in other fields. After they were taken up, the application of these principles was then considered in connection with specific and local mining problems. It was reasonable to believe that this plan would be interesting and at the same time would present the subject from a broader point of view. When members of the conference were asked to prepare papers, they received every encouragement to take

pains and make them as thorough and complete as possible. It also led to another field. This led in some cases to suggestions from the shift bosses as to the details of our operations. These suggestions from the shift bosses were then embodied in papers and presented at subsequent meetings with the guidance and assistance of the instructor. During this coming year we plan to increase the scope of the field. A course to miners will take up the principles and first course in mining methods; a course for shift bosses and engineers, will give more advanced work in mining problems and greater detail in mining methods; and a course for the foremen's dinners will include subjects of a broader nature that will apply to all concerned. The course at the foremen's dinners consists of lectures, while that at the Copper Queen mining conference for shift bosses and engineers is a combined course of lectures, problems, readings, study, etc. There are many things that a shift boss ought to know and, since they can only be learned through study and application, these courses have been instituted.

C. E. Arnold, Miami, Ariz.—Mr. Chairman, Mr. Hodgson made a practice of putting on some instructors. It would be interesting to hear what the results of that have been.

J. P. Hodgson.—Some months ago we took up the study of classifying ground. After the ground was classified we experimented in the different classes of ground in an attempt to standardize our drilling methods underground. It was found that in trying to standardize we bumped into old miners who, in their opinion, knew more about it than we did. So we decided to take the line of least resistance. When we found a man unwilling to learn a new method we did not fight him. We put him in a stope to work and took young men and trained them. We appointed several instructors and have them in the mine. They take young men who want to get ahead and instruct them for a period of 10 days to 2 weeks, then we supply them with a machine and equipment and put them in a drift, or raise, as the case may be. They get miners' pay and a bonus, depending upon the classification of ground upon which they are working and advance made. We have found this to be beneficial, and we have decided in the future rather than to fight our miners, to help them.

D. W. Brunton, Denver, Colo.—I would like to give some figures on speeds maintained in tunneling operations where persistent effort in teaching the miners who were engaged in driving tunnel headings resulted in an increase of speed from 250 ft. per month to 750 ft. per month, and in ordinary ground, after the workmen were thoroughly trained, there was little or no difficulty in maintaining speeds of from 600 to 750 ft. per month, with three 8-hr. shifts—in headings 7 ft. high by 8 ft. wide.

THE CHAIRMAN.—This subject verges on that allied subject, efficiency engineering, with which we have been experimenting. A characteristic of that subject is that it does not take very long to find out how little you know about a great many standard operations, and that they are not done as they should be done, and are not yet as we would like to have them. I don't know whether our experience has been like that of others or not.

## The Block Method of Top Slicing of the Miami Copper Co.

Discussion of the paper of E. G. DEANE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1515 to 1518.

THE CHAIRMAN (P. G. BECKETT, Globe, Ariz.).—The mining of large orebodies has in the last few years been such a big factor in the copper output of this State, and, in fact, of the whole country, I feel that we should have some valuable discussion this evening on the various mining methods described in the papers which are to be read; and I hope you will start a discussion on Mr. Deane's paper.

J. A. EDE, La Salle, Ill.—I would like to ask whether, the timbers being left in, they have any fires owing to that fact?

E. G. DEANE.—No, we do not. The timbers in the mat seem to lose life. I cannot tell you exactly what change takes place, but they do not burn readily. There is undoubtedly a chemical change in them—a change that makes them lose all strength. I spoke of the strength of the sills. When we first started, we put in 4 by 8 sills, but those sills had no more strength in 6 months than a 2-in. plank would have.

J. P. HODGSON, Bisbee, Ariz.—I would like to ask Mr. Deane if the central supply raise in the center of the block has been satisfactory; and also whether in extracting the ore they put small raises up so that wheelbarrows are not used in the stope?

E. G. DEANE.—The supply raises have been very satisfactory. We do use wheelbarrows, as our raises are spaced 50-ft. centers; it may be that later it will pay us to put up small inclined raises. But just above the 420-level, where we have been doing our slicing, it has not paid us to do this.

J. P. HODGSON.—I would like to ask another question, and that is this: From your experience in top slicing, where have you found that the overburden decreases in weight? At about how many floors downward in course of extraction?

E. G. DEANE.—We have not gotten down that far. We have taken about ten slices in some parts of the mine, and have as much weight as ever. In other parts we are slicing immediately under the capping at the present time.

## Mine Fire Methods Employed by the United Verde Copper Co.

Discussion of the paper of ROBERT E. TALLY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1545 to 1553.

CHAUNCEY L. BERRIEN, Butte, Mont. (written discussion).—Having had much actual experience with mine fires which have occurred or have been active in the mines of the Anaconda Copper Mining Co. during the past 4 years and having lately visited the fire zone of the United Verde mine with Mr. Tally, I wish to congratulate him for the success which he has had in controlling this fire and making the mine perfectly safe for his miners. By this work he is not only keeping the property in operation but is reclaiming many lost orebodies.

The ventilation system as described by him is practically the same as that used under my direction at the Mountain View mine fire in Butte in 1913 and at the Pennsylvania fire in 1916. This same system is at present being followed at the Leonard mine to extinguish a fire which has been active for 10 years although bulkheaded.

The mining system in the Butte mines at the origination of these fires was the same as that at the United Verde, namely, square setting accompanied by the use of much timber bulkheading in the stopes owing to heavy ground. The pyritic content of the ore in Butte is not as high, but on the whole conditions have been the same, with the exception of actual location of the fire in the mine.

I agree with Mr. Tally that flooding, the use of $CO_2$ or any gas, and the use of steam are all impracticable unless the fire zone can be absolutely sealed. His system of upcast escapement shafts in the fire zone, the use of bulkheads and fans to keep the gases away from the firemen, must be resorted to. By the method he describes actual stoping may be carried on safely if started from the bottom of the fire and if the fire zone extends to the surface. In other words, the fire zone must be isolated from the rest of the mine by bulkheads and solid ground, with no ground above which can be damaged by spread of fire or caving.

The serious fires in Butte have occurred on or between levels in the mines where it was necessary to prevent their extension upward, downward and laterally while the work of actually extinguishing them was in progress. This necessitated proper bulkheading should we lose control of the fire, the installation of ventilation similar to Mr. Tally's and the systematic drilling of holes with diamond drills for distributing water on the fire and around it.

The Mountain View fire of 1913* was extinguished by the method described by Mr. Tally and the driving of laterals parallel to the fire

* See Fire-fighting Methods at the Mountain View Mine, Butte, Mont., *Trans.* vol. 52 (1915).

stopes. From these laterals many diamond-drill holes were drilled through which water was run onto the fire. This fire burned through stopes between the 200 and 600 levels.

The Pennsylvania mine fire of 1916 burned between the 1,000 and 1,300 levels and for a considerable lateral extent and was finally extinguished by the Plenum system and extensive diamond drilling. The diamond-drill work was in both cases the main factor. The details of this work were carried on by C. Edwin Nighman, at present Fire Superintendent of the Boston & Montana group of the Anaconda Copper Mining Co. and H. R. Tunnell, Foreman of the Pennsylvania mine, under the direction of the writer. A description of this work will appear in the *Transactions* in the near future.

As I have said before, we are at present working on the Leonard-Minnie Healy fire with the prospect of regaining much ore, if not actually extinguishing the fire. The extent of the ground bulkheaded off from the rest of the mine is 600 by 400 ft. lying between the 600 and 1,300 levels.

This territory is filled with gases and an uncertain extent of smoldering fire. Our plan is to drive lateral drifts through the fire zone, keeping them in solid ground. From these laterals holes are being drilled with diamond drills into all stopes below, so that eventually we will have water entering all the stopes at 10-ft. intervals. We have been at this work about 1 year and have entered and reclaimed practically all of the 600 and 700 levels. We have also entered the 800 halfway and up to date the orebodies regained have warranted the installation of electric haulage on the 800. In doing this work we are using the Plenum system, and so far we have not had to resort to the use of Draeger helmets. While this work was really started as a matter of safety for the future of the mine, we are regaining orebodies at no greater expense than required for development of ore on new levels.

Our experience with fires in Butte has given us some thought as to their causes and led us to install all possible safeguards against their recurrence and to assure the necessary protection from new fires and the means to extinguish them.

The possible causes of mine fires are as numerous as of surface fires and one could set down a long list of them and the ordinary precautions for prevention. All of these should be known to the superintendent of any mine, and the subject is covered very thoroughly by George J. Young in Fires in Metalliferous Mines.*

Over a period of about 10 years in 20 or more operating mines in Butte, the *actual* causes of mine fires have been: lighted candles left on or near timbers, short-circuiting and overheating of electric wires, switches, etc., due to carelessness in upkeep and operation, oily

* *Trans.*, vol. 44 (1912).

waste, spontaneous combustion from foreign substances in gobs, and possibly incendiarism.

We realize that mine fires are possible even though the greatest precautions in the way of regulations, warnings, etc., are taken, and in the past 2 years we have installed every protection against the spread of fires, facilities for fighting them and means of rescue.

All of our main operating shafts are downcast, around the collar of each shaft we have 2-in. sprinkler lines; all water lines, air lines and electric cables are in downcast shafts, water-storage tanks of 3,000 gal. or more have been put in every 400 ft. along the shafts with pipe lines from them through main drifts, laterals and many stopes; water may be turned into all air lines also. All underground fan stations, transformer and switch rooms and electrical apparatus have been made fireproof for safe distances from them, overload and no-voltage release and auto starters are used on fans; all connections between mines are cut off by concrete bulkheads and iron doors except where stopes are continuous; a firebug (or watchman) goes through each shift boss' run in all mines after each shift; all electric switches are disconnected at shaft stations between shifts; telephones are used in all large mines; surface fans exhaust only; metal receptacles are used for oil waste and refuse and are in safe places; first-aid and Draeger helmet crews have been trained at each mine and hold positions where they can be notified of trouble immediately; signs denoting directions to other mines and main shafts are posted on all levels; an "out of the mine" danger signal is used; smoking in the mines is prohibited; metal sconces for candles are used entirely and failure to use them means discharge of man; mining methods are being adopted to eliminate as much timber as possible, and, finally, two fully equipped rescue stations on the surface are maintained.

At each of these rescue stations we have 30 Draeger oxygen helmets with accessories, lung motors, an automobile, a first-aid man on each 8-hr. shift, a smoke room for training and a man in general charge over all first-aid and rescue work. Classes are being held continuously and the number of capable firefighters is increasing. We also have a fire superintendent who acts with mine foremen in all underground fire work.

We have had six serious mine fires during the past 6 years and they have been extinguished or bulkheaded without loss of life except in the Pennsylvania mine fire of 1916. The loss of lives in this fire was due to the failure of the men in following directions given them for their safety. Work on all these fires extended over months. There have been about 12 other fires discovered and extinguished in a few minutes, or an hour at the most. This success has been due to preparedness, the good work of the helmet men, and extreme watchfulness. The expense of installation of any system or equipment for fire protection and fire fighting will be offset many times if such trouble arises.

C. W. GOODALE, Butte, Mont.—I wish to say that I have before me a discussion of the fire that occurred in the Pennsylvania mine in Butte, by Messrs. C. E. Nighman and R. S. Foster, but it is quite long and the conclusions are very much the same as those given by Mr. Berrien in the discussion which has just been read. I will say briefly that the Pennsylvania fire started Feb. 14, 1916, about 8:45 p.m., at or very near the 1,200 air-shaft station. This fire resulted in the loss of 21 lives; however, had these men followed the instructions of those sent to warn them, a maximum of nine, and probably only seven, fatalities would have occurred. At the time the fire occurred the main working shaft was upcast, and while this condition caused no loss of life, considerable discomfort resulted from the presence of smoke and gases, and rescue work had to be abandoned until this shaft was made downcast, which was accomplished by placing a suction fan over the air shaft. It should be stated that when the fire occurred preparations were under way to make the working shaft a downcast.

It is extremely important that working shafts be downcast wherever possible, for the following reasons: (*a*) In case of fire, ordinarily men can come to the shaft and be hoisted without danger of suffocation (in some cases action of fire has been known to change the direction of air in shafts); (*b*) Where cold air is coming down a shaft, the tendency of the ground to slough is greatly lessened; also, timbers last much longer when not exposed to hot, humid air coming out of the mine, and as working shafts must bear the strain of traveling skips and cages it necessarily follows that they should be favored where natural conditions permit. Furthermore, air shafts can usually be retimbered without curtailing production.

On the other hand, during the winter months in cold climates, ice collects in downcast shafts even where there is very little moisture. It is necessary to clear the ice by chopping it out, which is a disagreeable and somewhat dangerous task. Furthermore, it is hazardous and uncomfortable to do repairing in downcast shafts during cold weather.

At the present time the working shafts of the Anaconda Copper Mining Co. in Butte are downcast wherever possible.

THE CHAIRMAN (PERCY G. BECKETT, Globe, Ariz.).—I think, as Mr. Tally said and as Mr. Goodale has brought out, mine fires are exceedingly disagreeable and dangerous things to have in a mine; and I think it is a vital subject for us all. There are several big mining districts having fires burning in those districts for a long time. It is one problem to keep the fires under control, and another problem full of importance to us all to prevent fires from starting from the numerous sources from which they can start.

GERALD SHERMAN, Bisbee, Ariz.—We have been unfortunate enough to have some fires, and we have used very much the same methods which

have been described by Mr. Tally and Mr. Berrien. I am not sure that the members here are familiar with Mr. Berrien's paper of last year; but I think there is a distinction between what Mr. Tally is trying to do or doing and what Mr. Berrien is doing. From the shape of the orebodies in the United Verde, it is very much more difficult to put them out than in Butte. It is very difficult to get at them. The ground over the fires is soft, crushed and open; and I doubt if drill holes could be put in to get water on the fires. I think Mr. Tally's effort, although he has put out many fires, has been principally toward controlling them by the Plenum system. He recovers the ground by blowing the fire away from the working places and driving the gases to the surface through broken ground or some passage which does not interfere with other work. This does not put the fire out unless it is kept away from other timbered ground until it goes out of itself. The only way I know by which fires have been put out in this country is by putting water on them. Mr. Tally has put fires out by that method, and Mr. Berrien has also. In the cases at Butte, most of the veins were in comparatively hard rock. They are not vertical, as I remember it, but very steep. By drilling from the hanging wall, which is safe to work in, you can pour water on the fire successfully, and that is the way we did it. In one of our fires, we were lucky enough to get water on top of the fire in such a way as to drown it, and it was finally put out so we could get in the ground and prove it. Another was kept down by putting water on it until we were able to get to the fire area and by following the smoke to dig it out, but that was a small fire. In one of our fires, which has been burning for some years, and which is now isolated, we tried to drill holes over the top with diamond drills, but it was unsuccessful, as the ground was too much broken. We were never able to get near the point we were driving for. I think the precautions which Mr. Berrien has suggested are very well worth carrying out. All of our fires have been spontaneous, as far as we know, and they have all occurred, with one exception which is rather doubtful, in the centers of filled and abandoned stopes, which were inaccessible, and we were not able to reach them quickly. We had to isolate them and cut them off from active workings and keep them from spreading. I think they are undoubtedly caused by the heating of sulphides in stopes where they are loose and granular, and where there is little ventilation. I believe if raises could be kept open for free circulation of air, the heat would be carried off so that the temperature would not rise to the point of ignition, but I am not sure of it. In some of our shafts, we are laying pipes and sprays to water the shaft in case of fire, and also putting up doors to cut them off and stop circulation from the shaft, but we do not care to say very much for the present.

J. P. Hodgson, Bisbee, Ariz.—Mr. Chairman, from my experience in mining, covering a period of about 30 years, I would say that I think

the most prolific source of mine fires is the use of candles, owing to the carelessness of the men in leaving what we call "snuffs" around their working places. I know of at least two mine fires in Michigan where I was operating that occurred in this manner. One of these we put out with water; in the other we sealed the shafts and shut down the mine for 3 weeks before we extinguished the fire. Regarding our experience in copper mining, as Mr. Sherman has said, we have had a lot of experience in fighting fires, but we are not yet sure as to which is the better method. One thing I might mention in this connection that might be of assistance to others is that we keep on hand a 15-hp. electrically driven blower mounted on a truck, together with 1,000 ft. of wire for connections and 500 ft. of galvanized 10-in. blower pipe and a roll of brattice canvas. This equipment is kept in a certain place. When a fire alarm is turned in this equipment can be sent underground and put in working order in from 30 to 45 min. With this equipment we can force back the smoke in the drifts and sometimes succeed in reaching the fire, and in putting it out before very much damage is done. We have a fire in the Lowell mine that has been burning for over 6 years and we have gradually encroached on the fire area by putting in concrete bulkheads and thus getting ore that could not be mined before on account of the fire. We are at the present time on the 800-ft. level of the Gardner mine driving a series of drifts over a fire area similar to those explained in Mr. Berrien's paper. We are putting water in these drifts so that it may percolate through the broken ground to the fire zone below. This fire is in an old sulphide area which has been stoped out, and I believe in a year or two we will succeed in extinguishing this fire.

THE CHAIRMAN.—Is there any more discussion on this paper? I would like to ask Captain Hodgson from his experience fighting fires at Bisbee what he considers the main fundamental precautionary methods that should be taken in the average mine. Is it more a question of shafts in the right place, upcasts and downcasts, or water lines laid throughout the mine, or a question of blocking the mine off into sections that can be closed quickly, or what?

J. P. HODGSON.—I believe that all of the things you have mentioned are very good, but in such mines as the Copper Queen it is almost impossible to do all of them in every case as the property is so extensive and as a usual thing the fire breaks out where you don't expect it. After careful investigation we adopted this blower and have it in reserve so that we can get quick action after an alarm is turned in. It is lowered and in operation in from 30 to 45 min. in any part of the mine, and we believe it is very good for our conditions. Also, at the Copper Queen, we have fire doors to block off the different parts of the mine so that in case of a serious fire in one part it can be cut off and the balance of operations

proceed, but I fear I could not say which is the best way. All the things mentioned are good but the fire usually starts in the wrong place and without giving due notice. We also believe that our method of ventilation, keeping the mine under pressure, materially assists us in keeping back our fires.

C. A. MITKE, Bisbee, Ariz. (communication to the Secretary*).—I am interested in Mr. Tally's methods of fighting mine fires at the United Verde Copper Co. because he has developed a system of mine ventilation which has made possible the mining of orebodies that are adjacent to fire districts, as well as orebodies which are actually on fire. Of course, when preliminary work is being done in the way of development of these fire areas there is considerable heat to contend with; but after the ventilation has been arranged the miners are able to stope out the ore with almost the same comfort as in ordinary orebodies which are not on fire.

*Gardner Fire District.*—Since the pressure system of ventilation was successful at the United Verde Copper Co., it was adopted at the Copper Queen.

There are similar mine fires in the Copper Queen, as, for example, the 9-1 district of the Gardner division where a fire has been known to exist for some time. About 2 years ago a large amount of carbon dioxide came from this district on the 1,000 level, in which three men were overcome. Steps were taken to inclose it by means of doors, and to have air pressure of about 7 lb. per square foot surrounding the district to prevent its spreading. A raise was driven between the 800 and 900 levels and intermediate drifts separating the fire country from live workings. On the 800 a drift and a number of crosscuts were driven over the hot area. Water was run into these intermediate drifts and crosscuts so as to cool the country rock. An exhaust fan was used to draw the air from the top of the drifts and allow fresh air to come into the headings. At present the amount of gas and heat coming from this district has been decreased and it is believed that in a short time the district will be cool enough so that the usual ventilating system will continue to cool it without the continued use of water.

*Cananea Consolidated Copper Co.*—About 2 years ago there were several fires in the Veta Grande and Oversight mines in Cananea. These fires were located principally in timbered drifts and stopes. While some of them were easily extinguished, nevertheless one of them in the Veta Grande required considerable work done by men using oxygen helmets. The greatest difficulty was experienced in getting the fire under control, on account of there being no comprehensive scheme of ventilation.

*Rules.*—There should be a set of rules at every large mine which

* Received September, 1916.

should prescribe first for the safety of the men and then, for combating the fire. The importance of this may be illustrated by the Holbrook mine fire which occurred 3 years ago. As two men were taken out of a raise on the 500 level when they were overcome with gas, we asked the foreman if he had a pulmotor. He said "it is here in this drift." The pulmotor had been there for 20 min. and it was used effectively on a shift boss and assistant superintendent. When helmets were required, they were available immediately. The same was true with canvas, axes, saws, etc. During the night the fire was sealed up entirely and the mine was in operation the next day. The importance, therefore, of having all necessary supplies on hand in case of fire cannot be overestimated.

## Cost and Extraction in the Selection of a Mining Method

Discussion of the paper of C. E. Arnold, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1519 to 1522.

The Chairman (Percy G. Beckett, Globe, Ariz.).—When you are at Miami tomorrow, you will see a very interesting comparison on the surface of the two mining methods that are being employed there largely. In the case of the surface over the Inspiration mining method, you won't see very much left of the surface. Everything has gone down. In the case right at the line of the Inspiration ground and the Miami, you will see that the surface over the top slicing which the Miami company has been doing in that country has been riding down evenly and uniformly; and it gives a very marked comparison of the two types of mining, and, to a certain extent, of the different degrees of waste dilution. I would ilke to hear from Mr. Maclennan on this subject.

F. W. Maclennan, Miami, Ariz.—A paper based on percentage extraction by shrinkage stoping is a difficult one to discuss for the reason that it is hard to arrive at extraction figures until a section has been completely mined out. In the Miami mine we are using a method of shrinkage stoping with narrow stopes and pillars. This ore has been completely broken up and we have started in at the far end to draw off the ore mass maintaining the top of the broken ore dipping at an angle of 60° away from the direction in which we are retreating. A very important point I think about drawing a large mass of ore down at an angle rather than trying to maintain the top of the ore more or less horizontal is that drawing at a steep angle has a tendency to create a horizontal movement in the ore mass as well as the vertical movement of the ore traveling down to the drawing off chutes. We know that this takes place on account of the horizontal movement of marked

blocks which were placed in the broken ore mass at the time of stoping. This horizontal movement tends to cut off and break up pipes and funnels which are liable to open up through the ore from the drawing off chutes, allowing waste to drop down to the chutes before the broken ore surrounding the open pipe has been drawn in.

I noticed one remark made by Mr. Lehman in presenting his paper, to the effect that ore is drawn off from the finger raises until capping appears in these raises. When this occurs the drawing off is considered completed and these raises are sealed. Our experience in drawing off ore in the Miami mine would indicate that this is not always reliable. In many cases we have had chutes piped clear through to the surface, allowing capping to drop down to the chute when only 10 or 15 per cent. of the expected tonnage from this chute had been drawn. When this happens we seal the chute off temporarily and draw other chutes surrounding it for a few days and then resume drawing in the chute in which the capping appeared with the result that, usually, after drawing 15 or 20 tons of capping the ore starts to run again and may continue to run for several hundred tons of clean ore before capping again appears. For this reason I believe it is very important to establish a horizontal movement in the broken ore mass in order to break up these vertical pipes to the capping and also to help break up the thin pillars which were left between the stopes, and I also believe it is important to have the tonnage record from each individual chute so that an indication is had of whether capping appearing in a chute is only temporary or whether it marks the final stage of ore drawing. I think both of these points have a very important effect on the final extraction and dilution of the ore drawn.

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## Power Plant of the Burro Mountain Copper Co.

Discussion of the paper of CHARLES LEGRAND, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1531 to 1538.

THE CHAIRMAN (B. B. GOTTSBERGER, Miami, Ariz.).—I think this experience of the Burro Mountain Copper Co. with oil engines as described by Mr. Legrand comes at a very opportune time. The fuel almost universally used in the mine power plants of the Southwest is oil. Additional economy in its use for power production is much to be desired. In the case of the Miami Copper Co., we have lately found it necessary to enlarge our power plant and, based on the experience of the Burro Mountain Copper Co., we have decided to install oil engines similar to those in their plant. I think it would be very interesting to have some discussion on this subject. Mr. Langton, did you tell me you had prepared some remarks?

JOHN LANGTON, New York, N. Y.—This is an exceedingly useful paper. The plant it describes constitutes a long step in advance, and a bold step; and it is pleasant to know that the courage needed to undertake it has been rewarded by success. We have known that Diesel engines have been gradually and carefully developed by at least three manufacturers in Europe to units of large size, and of demonstrated reliability with the fuel oils there used. We knew that in getting the fuel oil into the cylinders and igniting it, European practice had dealt with fuels more difficult than viscous California oil. But where the same viscosity is accompanied by a different composition, the parallel ends at combustion. All contracts for California fuel oil as used for boilers and furnaces in Arizona contain a clause allowing a small quantity of sand. This oil is the only form of fuel we can depend upon, both in quantity and price, on which to base reliable estimates of the fuel-oil costs of operating Diesel engines in Arizona. As compared with the known conditions of European practice, there remained the serious question whether the sand allowance, together with any other ash content of asphaltum-base California oil, might not form enough grit to cause prohibitive wear in the cylinders. The Tyrone plant has done us the very great service of definitely determining this cardinal matter.

The Tyrone plant has also settled another minor but weighty point that has heretofore been a troublesome item in estimating operating costs; that is, the quantity and cost of lubricating oils consumed. The table of costs in Mr. Legrand's paper shows that lubricating-oil cost is more than 8 per cent. of the fuel-oil cost at Tyrone; and since the average cost of fuel oil at Arizona points is only $1.45, instead of the $1.85 to $1.96 per barrel stated for Tyrone, the figures he gives for lubrication would add not less than 10 per cent. to the fuel-oil cost in an Arizona plant. Considerable as this is, it is only half of what we have heretofore felt obliged to allow in estimating the cost of lubrication. This notable reduction is in part quantity and in part price per gallon. By inadvertence the quantity of lubricating oil and its cost per gallon were omitted from Mr. Legrand's paper. He has given me these figures with permission to state them. The monthly consumption of one engine running continuously is:

175 gal. of cylinder oil at 42c. per gallon.
170 gal. of engine oil at 28c. per gallon.

At the 1,250-b.hp. sea-level rating of these engines, these figures correspond to:

$1.16 cost of lubrication per brake horsepower-year of rating and
1.36 grams of lubricating oil per brake horsepower-hour of rating.

I have stated the quantity of oil in grams in order to compare with the

figures for estimation given by two European manufacturers. They were:

2 to 3 grams given me by one manufacturer and
2 to 2.5 grams given to Mr. Legrand by another manufacturer in another country.

The experience at Tyrone does not cover the question of what sulphur content is permissible in the fuel. I understand from Mr. Legrand that there was less than 1 per cent. sulphur in the fuel oil at Tyrone. In Europe, I found that sulphur was a great bugbear, but from such inquiries as I was able to make, and from what others tell me of their inquiries, there is a considerable difference of opinion as to what sulphur content is permissible. One maker stated explicitly that up to 1 per cent. sulphur gives no trouble in the cylinders, provided there is a suitable quality of cylinder oil, but that 2 per cent. is the practical limit of sulphur with any quality of cylinder oil. On the other hand, another maker referred to their engines running satisfactorily on Rumania oil with 3 per cent. sulphur. I was informed that the bad effect of sulphur seems to be wear of the cylinder, not by corrosion, but indirectly by destroying the lubrication. It is possible that the low cost of cylinder oil at Tyrone, both in quality and quantity, may be due to the low sulphur. That, however, does not affect the value and certainty of the Tyrone figures to us in Arizona, where the fuel oil is identical with the Tyrone oil.

Thanks to this valuable paper, we can now estimate with confidence both fuel oil and lubrication costs. Operating labor can be foretold with a close degree of accuracy, and the minor item of supplies with a sufficient degree of approximation. The interest and amortization charges are definitely fixed, as a matter of opinion or policy based on the conditions of the industry that the power plant serves. There only remains the very important item of maintenance. That is not yet determined at Tyrone. But when it is determined for that plant, I fear that it will be of but slight help to others. In plants consisting of a few large and costly units, individual good or bad fortune must always be a very large factor in repair costs. In estimating maintenance, all we can do is to try and be on the safe side and allow for a larger cost than we can imagine to be possible. Whether the results show surplus or deficit in this item is a question of good or bad fortune at the plant, and of whether the estimator is a pessimist or an optimist.

S. J. Kidder, Mogollon, N. M.—There is one question I would like to ask as to the cost of lubrication per brake horsepower year. It was stated that it was based on 1,250 hp. as the load of the engine.

John Langton.—The rating of the engine, I might have said, in order to estimate the carrying load, the cost of oil being comparatively small.

## A Comparative Test of the Marathon, Chilean and Hardinge Mills

Discussion of the paper of F. C. BLICKENSDERFER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1333 to 1348.

THE CHAIRMAN (B. B. GOTTSBERGER, Miami, Ariz.).—On your trip today through the Inspiration and Miami mills you have seen in actual operation the machines which represent the changes adopted in grinding methods in the Miami district from the beginning of operations in March, 1911. In one section of the Miami plant are three Chilean mills not yet discarded, representing the first machines adopted. In another section, pebble grinding is still in operation, while in the balance of the plant, pebbles have been replaced with iron balls. In all cases these machines are handling a feed which is the product of roll grinding down to about 1/4 in. in size. The latest step in grinding in the district, you saw at the Inspiration plant, where in Marcy mills a feed, the larger pieces of which remain on a 2-in. opening, is being reduced in one operation to a product of which only about 3 per cent. remains on the 48-mesh screen. A detailed comparison of the methods used at the two plants would be very instructive but our own work at Miami has not yet reached a point where such a comparison is possible, the reason being that our final product is not yet fine enough. By means of larger Hardinge mills operating in closed circuit with Dorr classifiers soon to be installed in one section of the plant, we hope to obtain under properly balanced operating conditions the same end result as at Inspiration. It might interest you, however, to know some of the results we have obtained here by the substitution of balls for pebbles. With pebble grinding, using three 8-ft. by 22-in. mills, followed by one 8-ft. by 66-in. mill for regrinding the coarse product of the primary mills, we handled 636 tons per 24 hr. in one section of the plant. Substituting balls for pebbles in the first operation and using two mills in place of three we obtained a tonnage of 714 tons to the section. With the use of balls both in the primary mills and in the regrinding mills, the tonnage for one section was raised to 822 tons per day. Coincident with this increase in tonnage, we also obtained a greater efficiency. The use of pebbles in all mills showed 21.5 per cent. plus 48-mesh material in the final tailing, which figure was reduced to 13.9 per cent. for balls followed by pebbles, and to 4.6 per cent. with the use of balls both in the primary and regrinding mills.

We should be able to have an interesting discussion on this subject this afternoon if some of the mill men present will participate, particularly

as we have before us Mr. Blickensderfer's paper, presenting what may be one step more in the evolution of fine grinding.

R. B. YERXA, Miami, Ariz.—I would like to ask if the Marathon mill can operate at such a tonnage as to produce the same product in fineness as that produced by the Hardinge and Chilean mills?

F. C. BLICKENSDERFER.—At the time this paper was published, such had not been the case. However, the Marathon mills of the larger type, now in operation at the Burro Mountain Copper Co., are grinding considerably finer than any product mentioned in this paper. Those mills at Tyrone are, roughly speaking, operating on 300 dry tons per 24 hr., and are grinding this down from 15 to 18 per cent. on plus 48. The power consumed is about 48 and the energy units corrected for tonnage are 130,000.

A. P. WATT, St. Francois, Mo.—I was much interested in reading Mr. Blickensderfer's paper comparing the efficiency of the Chilean, Hardinge and Marathon mills. The question of fine grinding is now receiving considerable attention in southeast Missouri. At present the jig middling, which is through 10-mm. on 2-mm., is generally crushed in rolls. One company, however, is using a Chilean mill and another company is using a Huntington mill for this purpose. Recently, however, the use of pebble and ball mills has been considered for solving the problem of fine grinding. Hardinge and Marcy mills are in regular operation in the district for the grinding of middling and two companies are to install Marathon mills for the same purpose.

By reason of our grinding problem I was much interested in Mr. Blickensderfer's paper, but as I read it a few questions arose in my mind. One was that in the test the Marathon was using metallic rods and was being compared against the Hardinge which was using pebbles. It is well known that the use of steel pebbles—so termed—in the Hardinge mill increases its efficiency. I wondered if that fact had been given consideration in later tests by running the Marathon against the Hardinge, using steel pebbles instead of flint pebbles in the Hardinge. I noticed that the Marathon feed contained about 63 per cent. solids while the Hardinge mill feed contained but 40 per cent. Would not an increase in the percentage of solids in the feed to the Hardinge increase the efficiency of that machine?

I would like to ask as a matter of information whether the rods in the Marathon mill still retain their cylindrical shape after continual use or if they become elliptical? Also, do the rods retain the same diameter at both ends after use, or do the rods at the feed end wear more than at the discharge end?

F. C. BLICKENSDERFER.—In answer to the first question, as to whether or not we have tried steel balls in the Hardinge mill, I will say,

"no." The idea of using steel balls in cylindrical grinding machines was just being introduced. The second question regarding the wear of rods is one which I failed to make sufficiently clear. The rods used in the small Marathon mill of 7-ft. length, wore almost uniformly from end to end until they reached a diameter of ½ in. After that some of them would flatten out, take an elliptical shape, and all rods would wear with a slight tapering toward the feed end, due no doubt to the fact that the particles of coarse feed kept the rods at a greater distance apart in the region of the head end than at the discharge end, so that the surfaces of the rods at the head end were constantly exposed to rock. The tail end was occupied by finer particles of pulp. Lately, in our present practice, due to abnormal conditions regarding the supply of steel, we have not obtained successful results by using the present supply of rods. This, I think, is due to poor steel. We have had rods 1 in. in diameter break in two. We have had rods nearly ½ in. in diameter go into the shape of an ellipse, and then again we have had defective castings. These are mechanical defects that are well on the way to perfection, and will not seriously retard the ultimate success of the mill.

R. B. T. Kiliani, New York, N. Y.—Mr. Blickensderfer's paper is of considerable interest as it gives the first published and authoritative data on the operation and performance of the Marathon mill and to this extent it may be considered an interesting contribution to our knowledge of milling machinery. No data are given as to the metallurgical results obtained with the different machines, and to this extent, it is not conclusive as to their relative suitability for the work under consideration. The extraction and grade of concentrate obtained by the use of various types of grinding equipment are naturally of at least equal importance to their actual mechanical efficiency. In other words, the net profit per ton of ore handled is the final criterion of economic efficiency. The mills were not operated at the same time, and no attempt was made to vary the conditions of operation to obtain the maximum efficiency from each, except possibly in the case of the Marathon mill. For these reasons, it is not a "comparative test," although, as above mentioned, thanks are due the author for some interesting data.

The Chilean and Hardinge mills were operated together for a period of 63 days, but after the first 30 days, it became necessary to shut down the Hardinge for relining, as it was started with an old lining. Owing to the type of lining used, flint pebbles set in cement, a longer time should have been allowed for this work, and especially for the cement to set, than the 2.36 days stated by the author. The short life of 159 days for the lining of the Hardinge mill may be ascribed to this fact, as well as other conditions which will be taken up later. The actual percentage of time lost through all delays in the operation of Hardinge mills at other plants

is well under 1.5 per cent., when proper care is taken that they operate under conditions which have been found to be best. The life of the same type of lining at the plant of the Arizona Copper Co., immediately adjoining the Detroit Copper Mining Co., is as high as 400 days, with an average life of over 200 days.

Two other conditions to which may be ascribed the short life of the lining are the dilution of the pulp and the speed at which the mill is operated. It is a well-known fact that for most efficient operation of a pebble mill, the solids in the pulp should be in the neighborhood of 60 to 70 per cent., and under these conditions, the wear of the lining is very much reduced. The speed of the mill of 29 r.p.m., mentioned by the author, is too high for this type of lining, since the action of the pebbles is such that they are thrown across the mill and strike the lining instead of acting upon the pulp. This, of course, causes the lining to wear out faster than would otherwise be the case. These two conditions of pulp consistency and speed of the Hardinge mill also affect the consumption of pebbles, which is very considerably higher than that at the No. 6 concentrator of the Arizona Copper Co., which was 1.334 lb. per ton ground, for the year 1914, as against the figure of 2.33 lb., as given in the paper under discussion.

More important, however, than the effect upon consumption of lining and grinding mediums of these two factors of speed and pulp dilution, are their effect upon the capacity obtained from the mill. It has been found that a dilute pulp with a high percentage of moisture, tends to very much decreased capacity and somewhat higher power. Curves and other data showing the effect of pulp dilution on capacity, fineness of grinding, efficiency, and power have been published for the past 7 or 8 years, and it would therefore be merely a repetition to give these in this discussion. Suffice it to say, however, that the efficiency of pebble mills has been increased over 50 per cent. by thickening the pulp fed to them. Had experiments been made to ascertain the proper speed and pulp density, the results with the Hardinge mill would have been much more satisfactory.

As already mentioned, no information is given as to the metallurgical results obtained with the different machines, other than the statement that the total recovery after the Marathon mill "was almost exactly the same" as with the Chilean mills. It is a well-known fact that the recovery by gravity concentration when grinding in Chilean mills is lower than when the ore is ground in pebble mills of the conical type. This does not necessarily hold true when flotation follows gravity concentration, but it is found that the grade of the concentrate is lower, and that it contains a higher percentage of insoluble matter, when grinding in Chilean mills. The Marathon-mill discharge, however, owing to the fact that it does not contain very much minus 48-mesh material, requires regrinding in some other type of machine in order to obtain the extraction and grade of con-

centrate resulting when grinding the same ore in the Hardinge mill. By referring to the author's data, we find that a considerably greater proportion of the Hardinge-mill product is finished than is the case with that of the Marathon mill.

Mention is made in the author's paper of the fact that metal pebbles are being used in place of flint for grinding in Hardinge mills, and the question arises why these were not tried at the plant under consideration, as it is a known fact that the efficiency of the machine is increased by their use, provided the same volume is used as of flint pebbles. It has also been found that they are cheaper than flint, considering all items of expense, except in certain exceptional cases. Another point in favor of their use, is the fact that increased recovery can be obtained without the expected higher grinding cost. Another fact which might be mentioned at this point, is that operating the mill in closed circuit with a Dorr, or other similar classifier, increases the efficiency and therefore decreases the grinding cost. It might be claimed that a similar increase in efficiency would be obtained by operating the Marathon mill in a similar manner, but this, to the writer's mind, is doubtful, owing to the much greater circulating load, due to the fact that this machine is unable to grind fine, except in larger machines where the capacity drops and the power increases very rapidly.

For a true comparative test, both machines should be operated side by side, taking the same feed and delivering the same final product, and the metallurgical results obtained with each taken into consideration. The Hardinge mill should also be operated with a metal lining and should use metal pebbles, since it would then be using the same class of grinding mediums as its competitor. It would naturally be absurd to suggest the use of a flint lining and flint rods in the Marathon mill, but the other has been proven to be entirely practicable. Should such a test be made, it would be interesting to compare the results with those reported in Mr. Blickensderfer's paper.

F. C. Blickensderfer.—It is mentioned in this paper that the total unit tonnage of 440 tons per 24 hr. was sent through the small Marathon mill. On the other unit we had three Chileans in operation. This test extended over 4 days, so we were reasonably safe in drawing our conclusions. There was a difference of less than 1 per cent. in the extraction of the two units, operating as I have outlined to you. It has always been our custom to assay the mesh sizes of all screen analyses made in the concentrator. I am not able to answer the question as to what condition the material on the screen sizes was in for concentration. There are no accurate methods for separating mineral from gangue in exactly the same manner as the gravity machines do. The nearest thing I know of is by means of a heavy solution of mercuric iodide and potassium iodide, with a specific gravity of about 3.20. That portion of ore which sinks is concentrates;

that which floats is tailings. There is a very definite separatory action in this method. In incorporating this method of separating the concentratable material from the gangue in the screen sizes, much difficulty was encountered, especially in treating slimes, so that, to my knowledge at present it is an open question as to the determination of the free mineral in the screen sizes.

The Hardinge mill slimed slightly less of the total material fed to it. The Hardinge mill slimed a little more of the mineral fed to it, as compared with the Chilean and Marathon. The method of determining to what extent the mill slimes mineral was as follows: All of the material in the feed which remains on a 200-mesh screen was assayed and its copper content obtained by multiplying per cent. weight by per cent. copper. Then all of the material in the product which remains on a 200-mesh screen was assayed and its copper content similarly obtained. Subtracting the copper content thus obtained in the tails from the copper content in the heads gives the copper content of the material actually slimed in the process of grinding. Proof of Statement that Hardinge Mill Slimes More Mineral than Chilean.

Assume Weight of 100 Tons

| | Tons | Per Cent. Copper | Tons Copper |
|---|---|---|---|
| Total + 200-mesh pulp in Hardinge feed...... | 91.63 | 1.03 | 0.944 |
| Total + 200-mesh pulp in Hardinge product.... | 68.15 | 1.01 | 0.688 |
| Material actually crushed to −200-mesh...... | 23.48 | 1.09 | 0.256 |

Assay of material actually crushed to −200 = 1.09 per cent. copper. And in case of Chilean mill:

| | Tons | Per Cent. Copper | Tons Copper |
|---|---|---|---|
| Total + 200-mesh pulp in Chilean feed....... | 83.46 | 1.15 | 0.960 |
| Total + 200-mesh pulp in Chilean product.... | 58.98 | 1.19 | 0.702 |
| Material actually crushed to −200-mesh...... | 24.48 | 1.05 | 0.258 |

Assay of material actually crushed to −200 = 1.05 per cent. copper.

Hence, the Hardinge mill has slimed more mineral to produce an assay of 1.09 per cent. copper than the Chilean in producing an assay of 1.05 per cent.

S. J. Jennings, New York, N. Y.—I would like to ask Mr. Blickensderfer a question. In his explanation of the action of the Marathon mill, he shows a diagram which does not seem to differentiate between the two possible actions that take place in the mill—one is a crushing action and the other is a grinding action. If we rotate a cylindrical mill somewhat rapidly so that the pebbles are entrained on the up side and fall, we would have a crushing action. If the rapidity of rotation is not sufficiently great to entrain the pebbles on that side but merely to raise them up sufficiently and allow them to go back, you have a continuous grinding

action. I would like to ask Mr. Blickensderfer if the Marathon mill was run at such a speed as to get a crushing action, or a grinding action. I assume from the fact that the rods wear evenly, with the small difference he has noted, he has merely a grinding action in his mill; but I would like to know that definitely.

F. C. Blickensderfer.—We have going on in the Marathon mill both a grinding and a crushing action. In the explanation referred to, I purposely inserted the words "strike" and "rolling together." Of the charge of rods, when observed inside the revolving mill about one-third is flying through the air and the other two-thirds are exercising a rolling action on one another. This is better illustrated by standing near a Marathon mill and hearing the constant striking of rods as they leave the periphery and travel across to the opposite part of the mill.

R. S. Handy, Kellogg, Idaho.—In the Marathon mill, in the matter of keeping them parallel—when the rods wore to small diameters, was there any difficulty in the rods getting crossed and not maintaining parallel relationship?

F. C. Blickensderfer.—We found that about 8 days' time was the limit that these mills could run without having the rods changed. The rods in the 8 by 12 mill catch hold of something and they twist up and form all kinds of grotesque shapes; it is time to stop the mill. The operator should keep close watch and remove the small twisted rods whenever a few appear in the top of the charge. In most cases, we straighten them out and put them back in again. The smaller ones are thrown away, and the load of rods brought back to about 7 tons, which is determined by filling the mill exactly one-half full of rods.

R. B. Yerxa.—I would like to ask what the tonnage was on the operation with Chilean mills when the test was made to find out what the recovery was between the two different methods of grinding.

F. C. Blickensderfer.—The tonnage was approximately the same. We had plunger-type feeders which governed the tonnage by the distance the gate was opened. We carried those gates at the same height, and the mill bins being filled at the same time, the tonnage of the two units would be approximately the same—within 5 per cent. I would say.

The Chairman.—I think we all ought to be grateful to Mr. Blickensderfer, in the first place for this paper, and again for his very full answers to our questions. I think that at times when we get to discussing a question of this kind, we are apt to give the impression that we may be criticizing, but I hope Mr. Blickensderfer does not feel that way in this case, as I believe he has brought something forward that is going to make us think. Present-day methods of concentration for handling the so-called disseminated copper ores call for fine grinding and it will only be

by comparison of results obtained at different plants that progress will be made.

Referring to the results in the present paper, I am impressed by the fact that taking −48-mesh product as the desired result, the Marathon mill yields a comparatively coarse product—one which in our case would not be suitable for the work we have to do. If Mr. Blickensderfer would not mind answering one more question, I would like to ask if he thinks putting the mills in closed circuit with drag classifiers would help in the fineness of the product.

F. C. Blickensderfer.—In the efficiency comparison of these machines, I call your attention to the fact that the Marathon mill when running with a tonnage of 440 turned out an efficiency, corrected for tonnage, of 143,000. When it was running as a finishing mill, taking 235 tons, in 24 hr. its efficiency was 107,000. Now, I take it from those two figures that if the Marathon mill was filled up, so to speak, with coarse material from its own discharge, much better results could be obtained. I have never seen any figures on those mills grinding in closed circuit.

C. W. Merrill, San Francisco, Calif.—I am not going to ask Mr. Blickensderfer a question. I am going to congratulate him. We men who have been studying the evolution of fine grinding in connection with gold milling for the past 20 years have always rebelled a little at the spherical contact—the point contact that we have had to contend with in all of the ball mills. We have felt that it was an extravagance of power that eventually would be eliminated; and when we prepared our paper for the International Engineering Congress on the subject of fine grinding, we ventured to predict that the knowledge we had of the small Marathon mill would result in just such work as Mr. Blickensderfer has explained to us today. I think there is a new principle involved in this machine—that of linear contact, as compared with spherical contact—and I want to express my appreciation of the work Mr. Blickensderfer has done.

F. C. Blickensderfer.—I feel as though I ought to make a few remarks. I should like to thank the Institute as a whole, and the members, individually, who have been so kind as to bring up these points. They seem to have brought out a better understanding regarding this mill. In publishing this paper, I wanted to be sure that we were right in drawing our conclusions concerning the Marathon mill. I feel that everything which has been said has been corrected in so far as it was possible to correct it. It might be of passing interest to note that every sample taken from the mill, all screen analyses, and all meter readings were made by the writer. It is not for the Marathon mill alone that I stand, but it is for the principle of the grinding element which the gentleman has just mentioned; and I think, and predict with equal empha-

sis that within another 5 years your ideas on the subject of fine-grinding machinery will have a different viewpoint.

ROBERT FRANKE, Miami, Ariz.—I would like to make a few remarks in regard to the use of equating the work performed by crushing. The tendency of this use has been rather noticeable during the last few years, and in this discussion has digressed to stating the capacities of grinding machines in terms of "energy units." I realize the need of resorting to an equating medium for efficiency comparisons, and favor its practice, but feel that we should be cautious, in view of several unknown and disputed factors, to apply this principle only within its limits of reasonably known accuracy, or under very analogous conditions. Otherwise, erroneous deductions may be the result.

The factors to be regarded are these: Rock is a heterogeneous substance, and in the porphyries generally is composed of feldspars bonded by a siliceous matrix. As crushing proceeds from a coarser to a finer state the softer components are first reduced to colloidal sizes until practically only the harder components remain for final crushing. Hence there exists an increasing unit resistance to crushing for which a determined allowance should be made in equating the production units.

Rittinger's and Kick's laws, upon which the interpretation of work performed in crushing is based, are in dispute, because of a disagreement as to the manner in which crushing forces act. Relatively, these two laws are fairly parallel for the coarser range of sizes, but become very divergent with the finer sizes.

The average size of $-200$ material is an unknown quantity, and in our present knowledge thereof can only be unreliably approximated. Moreover, it is also to be kept in mind that with this range of sizes there are encountered two classes of material, sands and colloids, each of which undoubtedly have different surface-volume relations.

In view of these factors, it is my opinion that the conversion of the work performed by crushing into energy units, for comparison of efficiency, should only be attempted when the fines of the feeds and products to be compared are practically equal, say within 2 per cent. of each other Further, the practice of stating the capacities of grinding machines in terms of "energy units" should not be adopted until the absolute value of these units has been determined, or misunderstandings will result. In lieu thereof, the practice of stating such capacities in terms of "Tons through a given mesh per horsepower unit" (which mostly is the primary object aimed for), for a given ore and given feed, would be more reliable and be resorted to until research along these lines will broaden the field for more exact application.

F. J. H. MERRILL, Los Angeles, Cal.—I would like to ask if the manufacturers of grinding balls have in any way made a suggestion in providing

sections of rods or in casting balls with spherical cavities in the hope that the balls will engage the adjacent one. I think the Jeffrey company makes balls of that kind.

THEODORE B. COUNSELMAN, Duluth, Minn. (communication to the Secretary*).—I have read Mr. Blickensderfer's paper with special interest because at the time these tests were being carried on I occupied the position of Efficiency Engineer at No. 6 concentrator of the Arizona Copper Co., Ltd.

A comparison of the work of the 8-ft. by 36-in. Hardinge pebble mills, at the neighboring concentrator of the Arizona Copper Co., Ltd., immediately suggests itself. This plant contained 12 mills exactly the same as the mill used in this test. In the following table, the figures for this plant are averages for the year 1914.

| | Detroit Copper Co. | Arizona Copper Co. |
|---|---|---|
| Power, horsepower | 56.56 | 56.65 |
| Tonnage | 244.00 | 196.70 |
| Dilution, per cent. solids | 42.80 | 55.00 |
| Pebble consumption per day | 527.00 | 262.00 |
| Pebble consumption per ton | 2.160 | 1.334 |
| Life of pebble lining | 159.00 | 200.00 |
| Tons per lining | | 40,000.00 |

The feed to the mills at No. 6 concentrator, was deslimed in drag belts. The tonnage is low because it was not necessary to crowd the mills. Ten would have been, and frequently were, sufficient for the crude ore tonnage at that time being handled. When the mills were first installed, 500 lb. pebbles per day were charged for an entire month, without seeming to create an excess. This amount was gradually cut down to 250 lb. where it remained.

The pebble lining used in all these mills was first introduced by David Cole, then in charge of the remodeling of No. 6 concentrator. It consisted of the largest-size Danish pebbles, set on end in neat cement or very rich concrete, the pebbles being rather carefully fitted together, much in the manner of building a dry-wall. The lower half of the mill was lined the first day. The mill was turned over the second day and the other half lined. It could then be put into operation, if desired, but was ordinarily allowed a day or two longer to take a final set. Such a lining required 10,000 lb. of pebbles, 50 sacks of cement, and cost installed, including cutting out the old lining, about $160. The life would vary from 150 days to as long as 400 days actual running time, with an average of about 200 days. Linings composed of silex blocks would last only about 60 days.

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* Received Oct. 7, 1916.

Lifting bars were tried in these mills. These were formed by rows of silex blocks, set on end, in the cylindrical portion of the mill. Repeated tests could detect no advantage from them.

A curious fact was brought out by several tests to determine the best dilution of the pulp. Mr. Blickensderfer mentions the "accidental oversize" in the discharge of the mill. The same thing was noticed at No. 6 concentrator, and the mill operators would correct it by "sticking a hose in the feed launder." This seemed so contrary to accepted ideas, that carefully conducted tests were made between several different pairs of mills, each pair having feed from the same source, but water being added to one, while the feed to the other was as dry as possible. In repeated tests conducted with the utmost care, it was shown that with greater dilution, there would be less accidental oversize. It was decided at that time that the most satisfactory way to operate these mills would be in closed circuit.

The Marathon mill at the West Yankie Concentrator was of great interest to all of us. The low power consumption was, of course, the direct result of the small diameter of the mill. I very much doubt if the same attractive difference in the power consumption of the Marathon and the Hardinge mills, would hold true with a larger-sized Marathon designed for finer grinding. In fact this is shown by figures from Burro Mountain.

The capacity and general character of the work of this mill seems to be explained by the fact that its action is that of a series of long-faced rolls. The wide face precludes the possibility of accidental oversize getting through, and it would be conceivably possible to pass an enormous tonnage through the machine, if no questions were asked as to the resulting grinding. The Marathon mill, judging from results given in Mr. Blickensderfer's paper, produces very little slime. This was noted all through the tests on the Marathon mill. Had this machine been invented 10 years before flotation was perfected, it would have solved many a difficult milling problem. Now, however, copper metallurgy has so changed that it is advantageous to make as much slime (minus 200 mesh) as can economically be made at each grinding. In 1913–14 the Miami Copper Co. replaced their worn-out Chilean mills with Hardinge pebble mills in order to get a more granular product. Now with flotation installed they are using balls instead of pebbles and grinding in closed circuit, in the endeavor to make more slime.

In comparing the results of the Marathon and Hardinge mills in these tests, the fact should not be overlooked that the Marathon mill was equipped with steel rods, while the Hardinge mill used flint pebbles. It would be ludicrous to suggest that the Marathon be equipped with flint rods, but to suggest that iron balls be tried in the Hardinge would be quite to the point. In one case that I know of the capacity of the

mill was increased more than 60 per cent., and the fineness of grinding increased by substituting balls for pebbles.

The measurement of grinding efficiencies has received a good deal of attention, and two well-known theories and methods have been advanced, of one of which, and a modification of it, Mr. Blickensderfer makes use in his paper. The other would give a very comparable result. To my mind, however, both methods fall down in that ore is not a homogeneous substance. The ores of the Morenci district, for instance, are relatively easy to granulate to, say, 20 mesh, but very much more difficult to pulverize to 200 mesh. The only real comparison between grinding machines would be to equip otherwise identical sections of a mill with the two machines to be tested, and determine the net profit made by each section. In one case that I know of, balls versus pebbles in Hardinge mills, have been compared in this way, with the advantage very much in favor of the ball-mill section, despite the fact that the relative mechanical efficiency was only slightly better.

I do not think that the Hardinge mill has been really tried out in Mr. Blickensderfer's tests. It should have been operated with the steel lining and the steel balls, and working in closed circuit with a Dorr, or similar classifier. Had both mills been equipped in this way, and operated to give products of equal fineness, I think there would have been little to choose between them. I should like very much to see this comparison made.

The Marathon mill should have a field where slime is anathema, where the mineral can and should be saved at a relatively coarse size. As Mr. Blickensderfer points out, there were several mechanical defects in the first machine, which have to some extent been corrected in the second. These defects can doubtless be eliminated entirely, if the machine proves to have a wide enough application.

The decline in the popularity of Chilean mills is due more to mechanical drawbacks than to quality of work. The wear of tires and dies is not serious, but as Mr. Blickensderfer himself notes, it is the auxiliary parts, the screens, screen frames, mullers, feed spouts, ploughs, etc., that give the trouble, not only in repairs but in operation.

David Cole, El Paso, Texas (communication to the Secretary*).—The paper by Mr. Blickensderfer presented at the session at Miami was very interesting to me because I witnessed the operation of the machines when they were being tested. I can vouch for the great pains that were taken by Mr. Hall and Mr. Blickensderfer in making the tests as nearly parallel as possible as to feed going to the competing machines, the power consumed by them, and the measuring of results as accurately as possible. Mr. Blickensderfer stated in his discussion that

*Received Oct. 4, 1916.

the tables compiled by him were done with great care and "are about as accurate as they could well be," and I have pleasure in corroborating his statement from the standpoint of observation on the ground at the time. We are indebted to Mr. Blickensderfer not only for his paper, but also for the very courageous way in which he defended the position he had taken, when under the rapid fire of questions that resulted.

The errors of comparison are fundamental and the question raised by the gentleman from Missouri (I don't remember his name) discloses one of the most important ones. Except for the purpose of showing different results when using grinding media of different gravity, it is fundamentally wrong and illogical to compare the work of a grinding mill of the tube-mill family, using flints, with one using steel grinding media. It would be illogical to suggest that the Marathon mill be operated with a set of rods made of agate, and it is equally illogical to compare the work of a Hardinge mill using flints with a Marathon using steel rods. Little importance should therefore attach to the comparison.

The comparison of work done, based upon the scientific theory of Stadler, Gates, Kick, et al., is beautiful on paper, but there are a lot of us who hesitate to accept the theory as "law." We are inclined to regard a direct comparison of grinders arranged side by side, getting feed from a common source through a mechanical distributor, and making a product that affords as nearly as may be the same screen measure, and at any rate affording an equal metallurgical opportunity for the subsequent treatment, as the Supreme Court in these grinding matters. The Marcy versus Hardinge ball-mill controversy is soon to have this kind of a hearing at the Inspiration plant, and the results will be watched with great interest, and if Mr. Blickensderfer can arrange to have the Hardinge mill at Morenci blocked down to about 6 ft. in maximum diameter, have it lined with steel with lifter bars, and filled with a charge of properly assorted steel balls, and then repeat the comparisons mentioned in his paper, our knowledge of fine grinding will be much advanced.

I believe in the use of the rods. I feel that the line of contact affords a better opportunity for the power being expended to be applied to the particles to be broken or crushed. I believe that the large-ball idea will prove the better scheme for breaking down the coarse particles, say to 8-mesh size, but that the rod idea will win out for the finishing process, particularly in closed circuit with an overflow classifier. But I do not think that the difference will be so overwhelming as to put the ball type in the discard, so to speak, for the balls may be modified in shape so as to get the line of contact result. What will prove to be the best length of rod, or the proportion of length to diameter, have not been determined. Perhaps a length equal to a single diameter, but with a dual axis will prove to be good. A "grinder" of this form presents a circular side and

end elevation and a square appearance in plan. It is balanced around its center of mass similarly to a sphere and affords lines of contact of lengths varying from nil to the largest diameter. It would, of course, tend to wear into a sphere, but it ought to do a lot of grinding before it finally surrenders to that form.

There is one point distinctly in favor of rods, and that is that the steel mills everywhere are equipped to produce them at a minimum expense. I carried on some experiments in a small way at Morenci looking to the development of some "laws" about shapes of mullers, but found nothing better than rods, and since these are easiest to get for the reason above given, the investigation was halted on that ground.

## Mining and Milling Practice at Santa Gertrudis

Discussion of the paper of HUGH ROSE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1295 to 1332.

JAY A. CARPENTER, Tonopah, Nev. (communication to the Secretary*).—This description of the Santa Gertrudis mill is of great interest to the operators of similar silver mills in Nevada. At the San Francisco meeting last September there were two excellent papers presented on Nevada silver mills; one by A. H. Jones on The Tonopah Plant of the Belmont Milling Co., and the other by E. E. Carpenter on Cyaniding Practice of the Churchill Milling Co., Wonder, Nev. At the same meeting, I presented a paper on Slime Agitation and Solution Replacement Methods at the West End Mill, Tonopah, Nev., covering this particular feature of our milling practice. A comparison of the data presented in these four papers is interesting in that, while the general design and practice is much the same in all cases, there is a wide difference in the design and practice in the various departments of the mills.

It is to be regretted that Mr. Rose did not round out this excellent article by following Mr. Jones's and Mr. Carpenter's lead in giving the average gold and silver content of the ore milled and the resulting tailing, with the consumption of chemicals and the cost of milling.

The practice of the Santa Gertrudis mill in the use of cyanide and zinc offers an interesting comparison with that of the Nevada mills, and a discussion of this point may bring out the reasons for the differences that exist.

Mr. Rose describes the ore as "clean with only small quantities of base metals." The fact that the zinc dust precipitate is 85 per cent. gold and silver, and that melting with only 3 per cent. of fluxes the bullion is 945 fine, points to the fact that the bases present in the ore are quite inert to the action of the cyanide solutions. In fact, the ore ap-

* Received Sept. 21, 1916.

pears to be very similar to the Tonopah and Wonder ores; yet the strength of the cyanide solution in the mill and the amount of cyanide used per ounce of bullion is nearly double that of the Nevada plants, while the zinc consumption is about the same.

The figures given in the accompanying tables are from articles in the *Transactions*, except those of the West End mill, which are taken from the annual report and mill data for 1915.

In the accompanying tables the gold and silver content of the Santa Gertrudis ore is given the same as Mr. Rose gives under "a representative analysis" in discussing flotation, and the ounces gold and silver precipitated are assumed, since extraction figures are not given, at a little above 90 per cent. of the ore content, which corresponds closely to Tonopah results. The total time of the ore in the different mills is roughly figured from the total ore tankage in the mills, and the figures are relatively correct.

These two tables give considerable data that should be of special interest to the students of cyanide solutions and the cyaniding of silver ores.

S. J. KIDDER, Mogollon, N. M. (communication to the Secretary*.)—Mr. Rose says that in clarifying their solutions they pass all solutions to be precipitated through sand filter tanks and that the addition of about 40 per cent. by volume of sawdust to the sand considerably improves the clarifying efficiency of the filtering medium. In this connection the writer would like to ask Mr. Rose if such use of sawdust is not liable to cause serious loss unless the sawdust is finally recovered and burned and the ashes saved? In the larger silver plants in Nevada it was the general experience that wood shavings, sawdust, cotton waste, sticks and rubbish in general which contained carbonaceous matter, if in contact with pregnant solutions for any length of time, would invariably cause considerable amounts of silver and gold to be precipitated. For this reason it was the usual practice to burn all such material and save the ashes, which were shipped from time to time to the smelters with either concentrates or slag. It would seem that the addition of 40 per cent. by volume of sawdust to the sand filter through which pregnant silver-gold solutions were to be passed would certainly result in a decided loss by precipitation unless the sawdust was saved, which Mr. Rose does not mention.

HUGH ROSE (communication to the Secretary†).—Carefully taken samples of this product after a year of service assayed about 120 grams silver per ton, this being more or less the grade of solution passing through clarifiers. These same samples washed with distilled water in a small vacuum filter, where there is no possibility of washing out anything but

* Received Sept. 5, 1916.
† Received Oct. 26, 1916.

## *Tables Accompanying Discussion by Jay Carpenter.*

### *Cyanide Data*

| Mill | Contents of Ore | | Per Cent. Values Removed by Concentration | Strength of Solution | | Consumption Sodium Cyanide | | Hours of Ore | | Temperature of Solutions |
|---|---|---|---|---|---|---|---|---|---|---|
| | Oz. Au | Oz. Ag | | At Batteries | At Start of Agitator | Per Ton Ore | Per Fine Oz. Bullion Recovered | In Agitators | In Mill | |
| | | | | Lb. KCN | Lb. KCN | | Lb. | | | |
| Wonder.......... | 0.216 | 19.48 | None | | 4.5 | 1.72 lb. NaCN | 0.095 | 60 | 120 | About 90°F. |
| Belmont.......... | 0.234 | 22.34 | About 12 | 5.0 | | 3.5 lb. KCN or 2.71 lb. NaCN | 0.153 | 48 | 96 | 90°F. |
| Santa Gertrudis.... | 0.067 | 12.00 | None | 8.0 | 11.0 | 3.15 lb. NaCN | 0.280 | 50 | 96 | Presumably 90°F. |
| West End.......... | 0.245 | 23.42 | None | 3.4 | 3.7 | 2.60 lb. NaCN | 0.120 | 70 | 110 | 115°F. |

### *Zinc Data*

Contents of Ore the Same as in Table Above

| Mill | Method of Precipitation | Oz. Au and Ag Precipitated per Ton Ore | Approximate Tons Solution Precipitated per Ton Ore | Oz. Au and Ag Per Ton Solution | Oz. Ag in Tails | | Lb. per Ton Ore | Zinc Used per Oz. Au and Ag in Bullion | Per Cent. Au and Ag Precipitate | Per Cent. Fluxes Added | Fineness of Bullion |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Partial | Complete | | | | | |
| Wonder.......... | Thread | 18.15 | 6.8 | 2.7 | 0.12 | 0.12 | 1.63 | 0.089 | 63.5 | About 15 | 968 |
| Belmont.......... | Dust | 17.67 | 6.0 | 2.6 | 0.16 | About trace | 0.95 | 0.054 | 80.6 | 14 | 932 |
| Santa Gertrudis.... | Dust | 11.06 | 4.5 | 3.0 | 1.00 | About trace | 0.69 | 0.063 | 85.0 | 3 | 945 |
| West End.......... | Thread | 21.70 | 3.6 | 6.0 | 0.40 | 0.07 | 1.21 | 0.056 | 65.0 | 15 | 930 |

dissolved values, reduced to 6 grams silver per ton. Were any precipitate present, it would not be soluble in water and would appear in the re-assay after washing. While there is a remote possibility that precipitation might take place, the chances are very small compared to the risk we run with the large amount of decayed timber brought into the mill every day in the ore flow, and from which up to the present time we have been unable to notice any bad effect. We have frequently had decayed wood pulp collected from various parts of the plant, principally from the foam on the top of the secondary Dorrs, where there would be every opportunity of considerable precipitation and enrichment, but with so far no sign of precipitation. The contained values run approximately what the solution assays, and a water wash will in each case remove them, proving that they are not there as metallic or precipitated values.

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## The Antecedent Mineral Discovery Requirement

Discussion of the paper of E. D. GARDNER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1693 to 1707.

VICTOR G. HILLS, Denver, Colo. (communication to the Secretary*).—If anyone advocates the abolition of the antecedent mineral discovery requirement for the purpose of making it easier to secure title to public land, I hasten to say that I am not of that number. I am in hearty accord with all that Mr. Gardner has to say in regard to easy patenting as a handicap to small camps, and that in general it acts to retard the mining industry. And the following sentences I want to repeat in order to emphasize and endorse them, "None of the writers on this subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some people would appear to believe." However, I believe that the discovery prerequisite should be abolished because it fails to effect the protection for which it was designed, and I look for other provisions that will better serve the purpose.

We propose to abolish the law of apex not because the theory is objectionable but because the question of physical fact gives rise to never ending litigation. Now the design of antecedent mineral discovery is good; the court rulings are unobjectionable; but when it comes to the physical fact this requirement is second only to the apex law as a healthy litigation breeder. The only circumstance that has prevented the prediscovery lawsuits from becoming as famous and as expensive as the apex

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* Received Nov. 3, 1916.

suits is that these cases are fought prior to patenting when usually there are not the valuable orebodies proven to stimulate greater expense. The question of discovery can not be raised after patent. In the pre-patent days disputes and conflicts are settled largely by compromise and the public is cheated right and left. In the course of my observation of claim disputes, and it has been long and extensive, the only question seriously raised, as a rule, is whether the contestant's "discovery shaft" has rock in place or is only in wash, and even the question of wash has been successfully disputed and a good assay allowed to save the day. If the discovery shaft has reached rock in place it is too well known that any crack in the rock, any oxidized seam, or any pegmatite streak in the granite, will "hold" if the question comes to a jury, and the other claimants usually throw up their hands to the stake with the oldest date. If there is no conflicting claimant the location is recognized and patented without so much as a thought of the public interest. Thus the discovery requirement, by being depended upon to safeguard against the frauds of securing springs, water holes, range privileges, power and reservoir sites, rights-of-way, summer residence sites, timber land, natural curiosities, etc., becomes a lever to assist in fixing such frauds. A crevice in the rock can be found almost anywhere, and there is nothing with which to contest a mineral title. When, as occasionally happens, two parties race for discovery on the same piece of ground, it is not ore they look for but any crevice in the rock which will serve as an excuse for a "vein." Quoting from Mr. Gardner's paper (p. 1696), "The discovery requires ment is designed, and operates, as a check on the disposal of land- of the people under the mining laws for purposes foreign to the intent of the law, viz., the speedy and bona fide development of the mineral resources of the public domain." The design is all right, but the operation is a shameful failure. Indeed, in the next to the last sentence of the article we read (p. 1707), "By far the greater number of mining claims that have been desired for purposes other than mining have had no discovery of mineral." It is not the use, but the abuse of the discovery requirement which renders it desirable to discard it. We should remember the adage that a law which can not be enforced is worse than none.

Let no one rise and say that we should have some one detailed from the U. S. Geological Survey to examine every discovery, at least prior to patent application. That would make only a little more interesting sport than heretofore. The geologist would come into court and say that there was not sufficient showing to constitute a mineral discovery, and his report would be true and just; then would come the "prospector" with his assay, scraped out of a knife-blade seam (and taken in a sack which had been previously used in sampling high-grade ore); the next witness would say that there was "a sufficient showing to justify a miner in following the same with a reasonable expectation of finding pay ore"

—that is standard court language which we all know by heart; the next witness would say that he once saw the famous ........ mine, which has produced millions, when it looked no better than the one in dispute; then would come the jury and give the claim to the "poor prospector," every time!

Now the question will naturally arise, "What substitute have you to offer for the protection of the public domain?" In reply, I would say, that in the first place the situation could scarcely be worse with the discovery antecedent removed. Next, the proposal to require recording a notice of annual assessment with vouchers and strict penalties has been so universally endorsed that we may regard it as certain to be included in any new law. This provision will entirely do away with the blackmailing scheme of resurrecting dead claims when some one else has proven the ground valuable. Also, since the only claims which can remain alive after one year are those on which the locator makes a bona fide expenditure, it will very largely do away with the fraudulent practice of holding ground for timber, springs and other surface values. There is an effective difference between an actual expenditure of $100 and an assessment which can be contracted for from $10 to $25. Further I should favor at least $10 per acre for annual assessment. This change alone will do more to protect the public domain than the pre-discovery requirement has ever done. The fact that more is charged for mineral land than for any other class (and the preliminary expense of patenting is also greater) safeguards agricultural land, summer residence sites, etc. The proposal made by one writer,[1] that, when patenting, $100 per acre be charged when there is no discovery, is worthy of consideration; but I would rather be rid of the discovery question altogether.

The one great thing which would do away with all of our troubles on the discovery question, and also a lot of other mining-law troubles, is the divorce of surface and mineral titles. I frankly acknowledge the radical nature of such a proposal, and presume that a vote among the fraternity would show me in a sad minority, but I exercise the courage of my conviction. The famous 33 questions which have been put forth by the Mining and Metallurgical Society do not bring out this most vital question, perhaps because it was regarded as practically useless. However, several members in their discussion recur to this subject, and evidently this matter forces itself upon the mind of every thorough thinker on the subject.

The use of the surface and the extraction of minerals do not, except to a limited extent, naturally belong together, and any law which persists in keeping the two inseparable must be full of injustice and trouble breeding. Why not meet the main difficulty squarely? Grant the pros-

[1] *Engineering and Mining Journal*, vol. 12, No. 16, ¶ C, p. 721 (Oct. 14, 1916).

pector the full and exclusive right to the minerals which he claims to be searching for, without discovery, together with all of the surface which he desires to actually use, but allow all other citizens the surface right-of-way for roads, trams, pipe lines, ditches, etc., and have the Government retain control of mineral springs, reservoir sites, and the like. Then all of the abuse found in the use of mining locations for hold-up purposes will disappear, and with it most of the urgency for pre-discovery

Our mining companies, after all their ground is patented, frequently buy and sell surface without mineral rights and mineral titles without the surface. They simply meet a natural physical condition. Why should not the Government do the same? Let the patent applicant pay $5 per acre for the mineral rights and $5 per acre additional for whatever surface he requires. If there be a specially valuable tract of timber, beyond the needs for mining the property, a spring or a town site, let the Government inspector assess the value of the same and grant the patentee the choice of buying or eliminating such portions of the surface. Any such surface exclusions can be monumented at the time of the official survey and the Surveyor General will have a map of both surface and mineral titles with scarcely any additional expense.

Separating surface and mineral titles also disposes of the "Classification" question, which I do not consider a practical thing in any case.

I favor making annual assessment work at least $10 per acre and pre-patent work $50 or $100 per acre. This is no more than Colorado, with its 10-acre claims, has always paid, and such a provision would help as a safeguard in the absence of the antecedent discovery.

If the separation of surface and mineral titles can not be accomplished, why can not the Government at least retain control of certain surface privileges during the pre-patent period, and at patent application make an assessment of special surface values and require payment for the same in addition to the nominal per-acre charge?

## The Diastrophic Theory

Discussion of the paper of MARCEL R. DALY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1137 to 1157.

EUGÈNE COSTE, Calgary, Alberta (communication to the Secretary*).—This new theory to account for the accumulation of commercial deposits of oil and gas, is deliberately and admittedly based on the hypothesis that the origin of these products is organic. Mr. Daly says (page 1142), "Should the organic origin of the petroleum that is found in pools be granted, the following interpretation is offered for the mechanics of its accumulation." But suppose the origin thus accepted by definition be incorrect, then what becomes of the diastrophic theory, Mr. Daly's interpretation? As Mr. Daly remarks himself (same page), "It is obvious that the history of petroleum will have to be entirely different, dependent upon its origin, either from emanations coming from the depths, or from organic decomposition in the strata themselves." Then why should one propound a theory, through which he is endeavoring to trace the history of petroleum, by starting with a definition of its origin? The origin should be the deduction, the forced conclusion of the theory, and not its starting point. Facts alone should be considered first, in any theory worth the name, until sufficient proofs have accumulated to permit of the deduction of a complete explanation for these facts. This would lead one to the origin through a real theory founded on facts, and not based on a preconceived idea perhaps true, but possibly fanciful. Mr. Daly's way can only lead on one road, namely the organic road, and, I repeat, it may be the wrong road.

In the case of the petroleum deposits, as I have endeavored to show in several papers on the subject, to which the reader may refer,[1] the facts clearly lead to the force conclusion that their origin is not one of accumulations of hydrocarbons at first disseminated in the sedimentary strata but the reverse process of subsequent local infiltrations and impregnations of hydrocarbon emanations from the depths. This new theory, therefore, founded on an erroneous conception of origin cannot be a help in the solution of the petroleum problem, not any more than were the anticlinal theory or the hydraulic theory, both of which were also based on the same error, although Mr. Daly is inclined to think that the authors of these two theories did not accept the misconception boldly enough (page 1142).

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* Received July 31, 1916.

[1] *Journal of the Canadian Mining Institute*, vol. 3, pp. 68–89 (1900), vol. 6, pp. 73–128 (1903) and vol. 12, pp. 273–302 (1909).

*Trans.*, vol. 35, pp. 288–297 (1905) and vol. 48, pp. 504–517 (1914).

*Transactions of the Institution of Mining and Metallurgy*, vol. 21, pp. 91–192 (1911–1912).

A careful reading of Mr. Daly's paper shows plainly the consequences of a wrong start as it leads him to the following fallacies:

1. That oil and gas are (first phase of Mr. Daly's process) primary decomposed products of organic matter formed in the sediments shortly after their deposition at ordinary low temperatures, and held by them for a short period of time, until squeezed up into a porous rock deposited above just at the right time to receive them. This is a very different conception from the view held by other organists (geologists believing in the organic origin of petroleums) who hold, on the contrary, that only a very long time (aeons of ages) will cause the distillations of organic matter in the strata—this immensely long time finally accomplishing the same result as heat in ordinary distillation processes. But if Mr. Daly's view be correct, deposits of that kind should be frequent in nature, and yet no one has ever been able to observe them. As a matter of fact they do not and never did exist, as organic matter decomposes into entirely different products which either escape immediately out of the forming sediments, or finally transform into carbonaceous matter or coal without any other hydrocarbon but marsh gas being produced.

2. That porous rocks shortly after their formation (second phase of Mr. Daly's process) may be supposed to be saturated with an emulsion of oil and gas in water, the oil and gas having been squeezed out of the muds below into these porous rocks, and remaining there in some mysterious way, instead of continuing their migration to the surface. Again I will observe that we do not see anything of the kind in nature—wells near the sea shore often striking pure fresh water (entirely devoid of oil or gas and even of salt water) in the recently formed sands or porous rocks beneath.

3. That the supposed gas and aqueous contents of incompressible strata or layers (page 1145, line 12) which according to Mr. Daly could not previously be compressed out to the surface only a few feet or a few hundred feet away, are now (third phase of Mr. Daly's process) compressed laterally for miles and miles to finally accumulate into far-away oil and gas pools. Apart from the fact that these porous layers cannot be both at the same time compressible and incompressible, we know that porous sand layers form in the sediments disconnected lenticular beds seldom continuous over large areas; this would certainly stop the ingenious long lateral migration of Mr. Daly, and would send the oil and gas out to the surface and so would the faulting and fissuring in the highly disturbed portions of the mountain chain considered through which the oil and gas are supposed to migrate laterally. One certainly cannot admit that the shales covering the sands which, according to Mr. Daly's first phase of the process, could give out so easily their hydrocarbon contents have now become so impervious over distances of many miles, as to make possible this long lateral migration through the sands preventing all the while

during this long travel the vertical escape to the surface only a few hundred feet away.

Mr. Daly says that "a partial origin of the rock pressure would thus have to be traced to orogenic deformation." This is rather vague and is only based on the remark that the pressure in Ohio and in Indiana, in the Trenton Limestone wells, was much smaller than in Pennsylvania and West Virginia; but we must not forget that the depths at which this gas was found in Ohio and Indiana were also much less than in the wells where the double pressure was recorded in Pennsylvania and West Virginia. As a matter of fact, the gas pressures in all fields are principally function of the depth indicating plainly the source from the volcanic magma below.

Mr. Daly's explanation of how capillary pressure seals an oil pool is ingenious, but may I ask why it only acts to prevent the pressure from dissipating outward into the surrounding sediments and why it did not also prevent the reverse movement of the accumulation of the gas and oil inward from the surrounding sediments into the sands?

F. G. Clapp, New York, N. Y. (communication to the Secretary.*)—The points of excellence in Mr. Daly's paper entitled The Diastrophic Theory are so numerous, that it is unfortunate he seems to have based his paper on a misunderstanding of the structural theory, as ordinarily understood. According to Mr. Daly, "the force which is supposed to have caused the motion is the gravity of the hydrocarbons." And again "Gravity or buoyancy is to be considered the sole agency through which accumulation has been brought about and, as such, is supposed to be adequate to explain accumulation under any condition of dip." So far as I know, no petroleum geologist believes these statements; so that Mr. Daly's paper must not be taken as a criticism of existing theories, but of his conception of them.

So far as I know, no petroleum geologist supposes the *motion* of the oil, gas and water in oil fields to have been caused by the force of gravity. To quote from an early issue of *Economic Geology*:[1]

> "The 'anticlinal theory' is only one of the factors in the accumulation of oil and gas pools; and a geologist, in order to locate a pool, even approximately, has to consider every other particle of evidence found in structure of the subsurface rocks, changes in intervals, texture of the 'sands,' character of the overlying beds, differences in pressure, relations to water in the rocks, shape of the surrounding pools, and character of the oil in the vicinity; and every factor must be given its due weight, before a recommendation can safely be made. After all other factors have been considered, it might seem as if the 'anticlinal theory' has been lost sight of as unimportant; yet the fact remains that structure is the essential condition which finally determines the distribution and size of the pools when all other conditions are favorable."

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* Received Aug. 28, 1916.

[1] F. G. Clapp's Discussion of paper of M. J. Munn, March, 1909, *Economic Geology*, vol. 4, No. 6, pp. 565–570 (September–October, 1909).

Passing this technicality, however, we can wisely weigh Mr. Daly's remarks that oil fields exist only on the side of mountain ranges *away* from the source of pressure. Certainly this appears to be applicable in the case of the Appalachian fields, and might well explain several vexing questions; as, for instance, the absence of oil fields east of Pittsburgh, Pa., west of which they are so numerous. In this connection, it is important to raise several questions, as, for instance:

1. Why is natural gas found east of Pittsburgh in abundance, while oil is generally absent?

2. Why is gas found on the *inside* of the Transylvanian Basin in Hungary, while oil seems limited to the outside regions, in Roumania and Galicia?

3. How will this theory explain the pressure of salt water in large quantities, in the regions away from the source of pressure?

4. How about oil associated with faults, as in Oklahoma, California and Wyoming?

5. Can the diastrophic theory be effective in dips of ½° to 5°, in regions several hundred miles from any mountain belt or major axis?

While the theory as propounded may account for one cause of the movement of petroleum in certain directions, the question of whether this is the *main* cause of accumulation must depend somewhat on the answers to the above questions. In my opinion, they can not all be answered favorably to the theory. Of course, however, a mere theory of cause or origin can avail little in petroleum engineering. It is where the oil is found that counts. We know that it exists in definite types of structure, when these are affected by certain other modifying conditions; but that these same structures, with a different set of conditions, will hold no oil. Successful oil-location depends preëminently on *inference*—*i.e.*, a comparison of conditions in prospective fields with those in fields already known elsewhere in the world.

R. W. Park, Washington, D. C. (communication to the Secretary*).—Mr. Daly's exposure of the frailties of the commonly accepted theories as to the mode of accumulation of oil is more acceptable than are most criticisms of this type, for after knocking our theories down over our eyes he does not leave us to grope blindly, but leads us to a brand new theory, both plausible and persuasive, that is to take the place of our old friends. Yet Mr. Daly's theory seems to be heir to some of the weaknesses characteristic of our old theories.

Most of the arguments that Mr. Daly raises against the anticlinal and hydraulic theories have been raised at one time or another by one writer or another, and there are probably now few geologists who will seriously contend that the force of gravity alone is adequate to accomplish

* Received Sept. 16, 1916.

widespread lateral migration of oil and gas, and the local accumulation of those materials in commercial amounts.

As Mr. Daly sees it, the chief failure of the anticlinal and hydraulic theories is that they do not take into consideration the effect upon the oil of the diastrophic forces that have affected the region. But most of the recent discussions of the origin of petroleum have laid stress upon the importance of these very forces, not, however, their importance as transporting agents, but as transforming agents, in changing the organic débris in the sedimentary beds into oil and gas. Mr. Daly tacitly assumes either that oil exists in the loose muds of the sea bottom practically from the moment of their deposition, or that it is formed soon afterward, during the period of compression due to sedimentation. In any case he appears to believe that oil exists in the sedimentary beds before they are deformed. However, until some well-authenticated examples of the formation of oil in unconsolidated sediments is brought forward, it would appear that the assumption that oil is formed in this manner is a less reasonable one to make than that oil is formed in part at least by the forces that caused deformation and perhaps local metamorphism. Of course it is possible that these forces not only have aided in the formation of oil, but also in its movement from one place to another; but it seems very unwise not to consider the first possibility at all.

Mr. Daly's discussion of the effect produced upon a prism of solid materials by pressure applied at the ends is interesting. After describing the purely theoretical example he applies the general principles involved to a succession of sedimentary beds, but in doing so he immediately agrees to consider only the upper part of the prism, that is, the part above the "neutral plane." One is led to wonder as to the depth this "neutral plane" lies beneath the surface, and just what the conditions are below it. Are the anticlinal axes the zones of contraction and the synclinal axes the zones of extension below this plane? If so, would oil, gas, and water tend to accumulate below the neutral plane in synclines rather than in anticlines? In the southern coast ranges of California many of the folds are shallow, and the "neutral plane" would in many cases certainly not be so very many hundred feet below the surface; so the question as to the accumulation of oil in them is not quite so academic as one might imagine.

According to Mr. Daly's theories, anticlines are important, so far as the accumulation of oil is concerned, only as they mark zones where the porosity is greater than in the surrounding region. In many of the California fields the folds are of variable character. Some are sharp and closely compressed; others broad and open. Between the two extremes almost any intermediate type may be found. It would seem natural to assume that along the more sharply flexed anticlines, the beds

would be more fractured, or at least more extended, as Mr. Daly would say, and that they would thus determine the position of the more porous zones. Yet it is very frequently the case that it is not along these sharper folds that the oil accumulation is greatest at present, but in the upper parts of the larger, broader, and more gentle folds. Moreover, in the California fields the beds are so variable—so tremendously variable—that the differences in porosity due to lithology must be far greater than differences in porosity due to position along the folds; certainly the large broad folds can not have so altered the porosity of the bed that lithology will not still be the controlling feature. Yet in spite of this, the California fields offer a most splendid example of the occurrence of oil along anticlinal folds or in parts of the region where those folds dominate the structure. It would seem that there must be some other reason than increased porosity that causes the oil so to seek these structures.

If the forces that have caused the deformation of the region are the forces that have caused the migration of the oil and built up the pressure under which the oil and gas are now confined, one would expect that this rock pressure would be fairly constant over considerable areas. The field of action of these forces has been extensive, and, if we accept Mr. Daly's theory, these forces have swept the fluids from a wide area, segregated them, and collected them in out of the way corners. It seems hardly reasonable to suppose that if the rock pressure is due to such a general cause the fluid in one little corner would have a certain rock pressure, while that in a neighboring corner would be under another pressure that is radically different. Nor does it seem probable that within one little pool the pressure would vary greatly; yet in the California fields variation in pressure in a given stratum seems to be the rule rather than the exception. This variability may of course be explained in part by variation in lithology and other natural features, and also by artificial features, such as the mode of handling a well. It is not always the later wells that show the lesser pressure, so the variation can not be explained away completely as a decrease for the pool due to release by some wells. But even considering the factors that make for local variation, it seems that the differences found between wells only a few hundred feet apart are abnormally large.

Mr. Daly's discussion of the method by which an oil pool may be sealed is interesting indeed, yet it would seem that the very factors that he calls upon to seal up an oil pool and to maintain a given rock pressure might equally well be called upon to prevent migration originally. If a given force causes the oil to move through the rock pores and collect under pressure at a given place, and if the force ceases to be active, why will not the oil move back over the path it originally traveled?

According to Mr. Daly's theory the forces causing the oil and gas to migrate also caused the water to move. Indeed, he considers that

the water moved carrying the hydrocarbons with it. Water, then, should be under the same pressure that the oil is under, as the force causing accumulation is the same, and the factors preventing dissipation the same for water as for oil. Yet in the California fields where water sands and oil sands are interstratified, no pressure comparable with the pressure in the oil sands is recorded for the water sands which lie between these oil sands unless a distinct flow of petroleum gas is noted with the water.

Mr. Daly notes the fact that in the Eastern fields there is an increase in rock pressure with the increase in age of the strata in which the oil or gas is found. The same can hardly be said of the California fields, for the greatest pressures noted occur in the younger beds which rest unconformably upon the shales in which the oil originated. The sands that are distinctly stratified with these shales contain oil under a much lower pressure.

In discussing rock pressure Mr. Daly points out that the chief explanations heretofore offered have been (1) that it is due either to hydrostatic pressure, (2) weight of superincumbent strata, (3) gradual accumulation of the inclosed gas, (4) capillary diffusion. He shows that some of these explanations are untenable, for the forces called upon are clearly not competent to have built up the high pressure now encountered. He therefore assumes that much of this pressure is due to the forces that have caused the deformation of the region.

An explanation that in many ways seems to be more reasonable is that the rock pressure is built up within the reservoir, not by the gradual concentration of gas, but by the change of some of the oil into gas. Also, perhaps by the same, perhaps by other reactions, heavy, viscous hydrocarbons are formed that so clog the rock pores that the movement through them of even the gaseous hydrocarbons is prohibited and the reservoir becomes sealed.

Rock pressure so developed would then have no direct relation to the pressure that originally caused the migration of the oil and it might be greater or less than that original pressure. Rock pressure need not be "fossil pressure," as one would be tempted to call it if one accepted Mr. Daly's interpretation.

Just what the reactions would be that would cause the formation of the gases within the reservoir can not be said, but as to those which result in the formation of the heavy viscous material there are a few more data. It is a pretty well known fact that, in the California fields, the highly mineralized water characteristic of the oil fields has a very pronounced effect upon the oil, and that where such water has entered the oil sands the oil in the vicinity is extremely heavy and tarry. This tarry material is in some places so viscous that it can not be pumped. Such material seems to be composed, in part at least, of oxygen and

sulphur compounds and they seem to have been formed by the interaction of the hydrocarbons and the mineral salts in the water. This tarry material is quite competent to, and probably does, fill the rock pores and prevent the passage through them of the more fluid hydrocarbons.

## The Application and Earning Power of Chemistry in the Coal Mining Industry

Discussion of the paper of EDWIN M. CHANCE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 112, April, 1916, pp. 711 to 714.

EDWARD H. COXE, Knoxville, Tenn. (communication to the Secretary*).—Mr. Chance has omitted to mention one very important use for the chemist in connection with the preparation of coal, and that is in connection with coal washing.

Efficient coal washing, and especially of the finer sizes, can be accomplished only under the supervision of a chemist, as the proper regulation of the washer can only be had by frequent sampling and analysis of the raw and washed product and the refuse, say at least semi-weekly; and only in this way can a check be kept on the quality of the product and the amount of good coal wasted in the refuse.

In large operations, frequently enough coal can be saved from going to the refuse pile to more than pay the salary of the chemist.

## The Composition of the Rock Gas of the Cripple Creek Mining District, Colorado

Discussion of the paper of GEORGE A. BURRELL and ALFRED W. GAUGER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 113, May, 1916, pp. 843 to 863.

J. S. HALDANE, M. D., F. R. S., Oxford, England (communication to the Secretary†).—The valuable paper of Messrs. Burrell and Gauger is of special interest to me as, through the courtesy of the management, I had an opportunity of visiting the Portland mine, Cripple Creek, in 1911, in connection with the investigations on acclimatization to high altitudes, carried out on Pike's Peak, near Cripple Creek, by Professors Yandell, Henderson and Schneider, Dr. Douglas and myself.[1] The analyses of one or two samples of the vitiated air were entirely confirmatory of those of the authors. A fatal accident owing to two men being lowered down a shaft filled with the gas had occurred in a neighboring mine just before our visit.

* Received Apr. 10, 1916.

† Received Aug. 25, 1916.

[1] Philosophical Transactions of the Royal Society, Ser. B., vol. 203, pp. 185–318 (1913).

The composition of the gas is so similar to that of the "black damp" met with in coal mines, metalliferous mines of different kinds, and wells, that there can be little doubt that its origin is similarly due to oxidation processes in the strata. It is very remarkable, however, that the gas is produced in such large amount in the Cripple Creek mines. The rock is apparently very porous or full of fissures, and also contains some substance which oxidizes freely. In some metalliferous mines, and to a less extent in coal mines, this substance is pyrites, the $CO_2$ being due to subsequent liberation of this gas from carbonates by the sulphuric acid formed; but in many cases the oxidation process is certainly a different one, and further examination would be needed in order to determine what oxidation process occurs in the rock at Cripple Creek. It follows from the authors' analyses that the black damp formed contains sometimes so little $CO_2$ (*e.g.*, samples 753, 754, 761, 772, 794) that it is lighter than air, as is not infrequently the case with black damp in coal mines, wells, etc.

Further observations on the relation of the issue of the gas to changes in barometric pressure would be of much interest. As I showed in a paper contributed to the *Transactions of the Institution of Mining Engineers* (Great Britain) in 1896, the special danger to well-sinkers from black damp arises from the fact that although they are well aware of the need for testing the air with a candle before descending, they do not realize that the test must be repeated at each descent. A well which, for instance, has been quite clear of gas in the morning may, in consequence of even a slight fall of barometric pressure, be overflowing with black damp in the afternoon. The enormous quantities of gas or air which may issue from a well with a fall of barometric pressure are very surprising.

It is a question of some interest whether it is better to apply ventilation by pressure or by exhaust in cases like that of the Cripple Creek mines. The objection to pressure is that in the event of the fan being temporarily stopped there may be an immediate discharge of black damp owing to the fall in the air pressure. In practice, however, there may be counterbalancing advantages.

As regards the effects of the black damp on the men, it must be borne in mind that the miners will all be acclimatized to the oxygen deficiency due to the altitude (about 10,000 ft.) of the Cripple Creek district. The results of the Pike's Peak expedition showed that there are three factors in this acclimatization:

1. The breathing is increased, so that at Cripple Creek, the lungs are 30 per cent. better ventilated than at sea level.

2. The layer of living cells which separate the blood from the air in the lungs plays an active part in forcing oxygen inward into the blood, where at sea level this layer is only passive, so that the oxygen passes in by simple diffusion.

3. The blood is about 20 per cent. richer in hæmoglobin and red corpuscles at Cripple Creek. Taking into account the influence of acclimatization, the effects of a given percentage of black damp in the air will probably be nearly the same at Cripple Creek as at sea level.

In a paper contributed this summer to the *Transactions of the* (British) *Institution of Mining Engineers*, I pointed out that the excessive tendency to emphyrema and bronchitis among older miners is probably due largely to breathing air contaminated by too much black damp. I was mainly responsible for the provision in the British Coal Mines Act of 1911 that the percentage of $CO_2$ should not be allowed to exceed 1¼ per cent. This maximum standard is easily attainable, and seems well worth maintaining. A very simple method of ascertaining how much black damp is present in the air is afforded by the "tube and taper" method which I introduced about 5 years ago.[2] This method was very useful to me in searching for a sample of vitiated air at the Portland mine, where it was by no means easy to find such a sample.

The general health conditions at the Cripple Creek mines appeared to be much better than at most other metalliferous mines known to me. The absence of miners' phthisis was a very marked and extremely interesting feature in this district, and suggested new ideas as to the causation of this very formidable disease.

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## Diesel Engines Versus Steam Turbines for Mine Power Plants

Discussion of the paper of HERBERT HAAS, presented at the Arizona Meeting, Sedtember, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1171 to 1183.

GEORGE W. HAWKINS, Tucson, Ariz. (communication to the Secretary*).—The paper by Mr. Haas will no doubt be followed with considerable interest, as it covers the power-plant problem in quite a comprehensive way, including not only the usual operating costs but also the fixed charges on the investment, and for load factors ranging all the way from 25 to 100 per cent. and fuel oil ranging from $1.25 to $2 per barrel. The very fact, however, that his paper does cover such a wide range of conditions, makes it the more imperative that any conclusion drawn from his analysis be very carefully considered for any specific case, for in making such analysis the type of prime mover and size must be selected for some one condition which might not be the proper selection had some other condition been assumed as the basic one, resulting in an unfavorable comparison under all conditions except that condition assumed.

From the reasons given by the author for his selection of the large turbine units, it is evident he had in mind very high load factors, ranging

[2]Fully described in my book *Methods of Air Analysis*, London, 1912.

*Received Oct. 9, 1916.

from 75 to 100 per cent., and high fuel cost. Under these particular conditions, then, his comparison would be a fair one, but any comparison for different conditions than these, such as low load factors, or low fuel cost, would be very unfair, for in the latter case the plant should be designed along entirely different lines, resulting in different first cost and operating expense. That this criticism is a just one is evident from his tabulation of power costs, for according to that tabulation the lower the load factor the greater is the difference in operating costs in favor of the Diesel engines. For example: With oil at $1.25 per barrel, at 6,000-kw. load the difference in total operating costs between the steam plant and the Diesel plant is 0.217c. per kilowatt-hour in favor of the Diesel plant, whereas at one-quarter load this difference has increased to 0.342c. per kilowatt-hour in favor of the Diesel plant. This being contrary to the generally accepted opinion and also to his own statement that high load factor and high price of fuel are conditions which favor the Diesel engine, leads one to make a careful analysis of the basis of his figures.

## Diesel Engines

### *Selection of Units*

The full-load plant capacity is stated as 6,000 kw. in three units with one unit to serve as a standby. Inasmuch as it would be necessary to operate all of the units to get 8,000 kw., leaving no spare (which is not considered safe practice with Diesel engines), the comparison of the steam plant and Diesel engine plant at 8,000 kw. must be eliminated. With the steam plant as selected it would be possible to operate continuously at 8,000 kw., but not with the Diesel plant. The comparison, therefore, should be limited to an output of 6,000 kw., 4,000 kw. and 2,000 kw.

The author has selected units rated at 2,000 kw. each, these being of the vertical, single-acting, two-cycle type. The writer has carefully followed the development of Diesel engines in this country and does not know of a single installation having this size unit. Two of the largest Diesel engine builders take the position that, in the present state of the art, they would not attempt to put out units over 1,000 kw. Up to date the largest units in the United States are those at Tyrone, installed by Nordberg Mfg. Co., which are rated at 1,250 b.hp., or 800 kw. It is possible, of course, to build larger engines than the above by simply multiplying cylinders, maintaining the same horsepower per cylinder. This, however, does not reduce cost, except for generators, nor does it reduce operating expense. From the author's fourth reason (given on page 1175) for the selection of this size unit, it is evident he figures on a unit having greater horsepower per cylinder. While it is true that in Europe units in excess of 2,000 hp. have been built in the two-cycle type, they are not considered, even by the manufacturers, to be out of the experimental

stage; nevertheless, they have gained some experience in the manufacture and operation of these large units, which experience would not be of much value to American builders, as the possession of suitable patterns and instructions is not sufficient to insure successful construction of Diesel engines, practical shop knowledge of construction being just as important in turning out successful engines as proper design. This size unit is therefore entirely out of the question for adoption in mining plants or any plants where reliability is a prime necessity. Therefore, engines must be selected not larger than 1,000-kw. capacity. There would then be six 1,000-kw. units for the load, with two spares, giving a total installed capacity of 8,000 kw.

### *Cost of Plant*

Based on the use of large units, the author has assumed a total plant cost of $90 per kilowatt. As smaller units are now being considered, the plant cost will have to be considerably revised.

The writer has seen estimates of costs of various plants ranging from $110 to $140 per kilowatt, depending upon size of unit, location, etc. The author, in a previous paper appearing in the *Engineering and Mining Journal* of Apr. 26, 1913, gives the cost for a plant of 1,400 kw. as $135 per kilowatt, this being based on 450-kw. units. Owing to the fact that the units under consideration here are larger and the plant itself of larger capacity, the writer believes that $110 to $120 per kilowatt would be considered a fair estimate of the cost of this plant, this cost including not only the engines and generators, but the piping, foundations, oil tanks, crane, building, jacket water, cooling system, etc. The installed cost of this plant would then be between $880,000 and $960,000 as against the author's first cost of $720,000.

### *Fixed Charges*

Fixed charge of 12 per cent., consisting of interest at 6 per cent., amortization at 6 per cent., has been assumed by Mr. Haas. The amortization, or sinking-fund figure is based upon 15 years' life of plant, sinking fund bearing 4½ per cent. interest. The life of the Diesel engine is an unknown factor, the present engines having been developed within comparatively recent years. The proper amount to figure for sinking fund is therefore merely an intelligent guess. Some Diesel engine builders figure that there should be an allowance made to cover the entire replacement of the Diesel engines every 9 or 10 years.

As against this, the life of the steam plant is well known. Steam plants are in good operating condition 20 to 25 years after installation, and the steam turbine has been perfected to such a degree that it is not

likely that there will be any radical departure from present types for a good many years to come. Whatever figure is used for the assumed life of the Diesel engine plant, the life of the steam plant should be figured at least 50 per cent. longer. If 15 years is assumed for the Diesel engine, resulting in a sinking fund of 6 per cent. with money reinvested at 4½ per cent., 25 years should be figured for the steam plant, giving a sinking fund of 2.25 per cent., or 3.75 per cent. greater fixed charges for the Diesel plant than for the steam. Considering that the first cost is about 33⅓ per cent. higher than assumed by the author, the increase in this item of fixed charges is a very material one.

*Fuel*

Fuel cost is, of course, the largest item of expense at high load factors, and any wrong assumption in this item would affect results very materially. The author has used a figure of 0.64 lb. of oil per kilowatt hour in arriving at his full-load fuel cost, and in fact, his fractional-load fuel cost. This figure is about what builders usually guarantee for four-cycle engines, and it may be obtained on acceptance tests, but there is of necessity always a falling down in actual operation from the test results, due to the method of operation, the way engines are kept up, etc. The fuel consumption, therefore, under normal operation, even when the plant is new, would be at least 5 per cent. higher. Owing to the excessively high pressure used in Diesel engines, any slight wear of cylinder would reduce economy very materially due to the enormous leakage possible, and hence, the average economy over the life of plant is likely to be considerably poorer than for the first few months of operation.

Owing to the very short time the Diesel type of plant has been in operation, it is impossible to tell just what this falling off in economy would be from operation the first few months, but that there will be such a falling off is certain. However, this is not being considered in this analysis. Engines of this large size, however, would probably be of the two-cycle type, which is not as economical as the four-cycle. The author makes the statement in the former paper referred to that the two-cycle engines have fully 10 per cent. higher fuel consumption than the four-cycle engines, making the actual operating economy 0.736 lb. per kilowatt-hour. His fuel consumption,therefore, should be 15 per cent. higher than he has figured.

Further, he has assumed that plant would operate on ordinary fuel oil running 320 lb. per barrel. Diesel engine manufacturers are willing to guarantee operation with heavy-gravity fuel oils, but except in a few instances, even where such guarantees have been made, the owners have after a short trial gone to the lighter-gravity oils, owing to the heavy

cost and trouble of keeping engine in running condition. To be perfectly safe, therefore, in deciding type of prime mover, one should not base operating costs on the use of heavy-gravity oils, and as the lighter oils run from 15 to 20c. per barrel more than the heavy oils in common use in steam stations, and run considerably lighter per barrel, as low as 285 or 300 lb., there is the possibility of further increase in the cost of fuel oil over that assumed by the author. The additional fuel cost at full load would therefore be at least $20,000 greater than given by the author, and may run as much as $46,000 greater if the lighter-gravity oils should be adopted for the Diesel plant. In this analysis, however, it is assumed that fuel oil can be used.

The above applies to full-load operation. At fractional loads there will be a falling off in economy of the Diesel engine in the same way that there is for steam turbines. Theoretically, it would be possible to get full-load economy at any of the fractional plant loads due to the number of small units, enabling the units in service to be operated at practically rating. This is not practical working condition, however, as Diesel engines have a fixed overload, and on a swinging load it would not be possible to have in service just the proper number of units theoretically required. This would mean that the units in service would be operated at fractional loads, in which case the economy would drop off quite materially, the same as with any type of prime mover.

### *Maintenance and Lubrication*

The next item of importance is maintenance. The author has assumed maintenance as 1 per cent. of the plant cost. This cannot be based on actual operating costs in this country for any lengthened period, as no plants of any size have been in operation long enough to give any reliable records. It would be like assuming the average maintenance cost of an automobile from the records of the first few months of its operation. On the other hand, it may be this is based on operation in European countries. However, considering the difference in price of labor, price of material, class of labor, etc., it is practically impossible to make proper adjustment to arrive at a figure for operation in the United States. The safer plan would be to investigate maintenance costs of actual plants in the United States, increasing these items to cover average maintenance over the period of the life of plant, and then make any adjustment necessary for more favorable conditions in this plant.

In the author's article in the *Engineering and Mining Journal* he assumes a figure which is 3 per cent. of the first cost of the plant for maintenance, making the statement that the figure is based on prolonged cost records of Diesel engine power plants. The writer has in mind two plants of fairly good size where the first 2 years' operation was even higher than

this, and it is safe to assume that the maintenance cost would increase as the plant gets older.

The cost of lubrication is one that varies widely, depending upon the design of engine, care and operation, etc., varying all the way from one-fourth to one-eighth of the fuel bill, so that the figure given by the author can be considered as a general average, although it might mount up to a considerably higher figure.

### *Labor*

The item of labor, considering the greater number of units, will be higher than assumed by Mr. Haas. In addition to the engineer and his assistants and electrical operators, there should be one oiler for two engines, which would make nine oilers per 24-hr. period. This adds a considerable amount to the labor rate.

## Steam Plant

### *Selection of Units*

Mr. Haas states that his reason for selecting 6,000-kw. turbines is the superior economy of the 6,000-kw. turbine over, say, a 3,000-kw. turbine, which justified the very much heavier first cost. This is the proper way to select size of unit, but it can apply only to one particular assumed condition. If the plant is to run at 100 per cent. load factor the superior economy of the 6,000-kw. unit will offset the fixed charges on the greater investment, but if the plant is to operate at one-half, one-third or one-quarter load, the selection of a 6,000-kw. unit may be a very bad one, for the total operating cost might figure out much lower with a smaller unit because of superior economy at the load considered and lower first cost due to smaller installed capacity.

### *Cost of Plant*

Assuming that his selection of 6,000-kw. turbines is the proper one. his plant cost is about right, but if plant were to operate at, say, one-half load or one-third load, an investigation of the total operating cost would probably lead to the selection of two 3,000-kw. units to carry the load with one spare, or a total installed capacity of 9,000 kw. against an installed capacity of 12,000 kw., resulting in the plant costing probably 25 per cent. less than assumed by Mr. Haas.

### *Fixed Charges*

As stated above, if life of the Diesel plant is considered as 15 years, the life of the steam plant ought to be, for fair comparison, assumed at

25 years, resulting in 3.75 per cent. lower fixed charge for the steam plant than for the Diesel plant.

### *Fuel*

While the assumed steam-plant economy checks with average practice, it is not by a good large percentage as high as is actually being obtained in some recent plants under daily operating conditions with units of approximately the same size as assumed here, this increase in economy being obtained by advance in boiler design—allowing increased steam pressure and higher superheat—the use of steel-encased boilers, the use of automatic oil firing—giving better furnace regulation—the use of properly designed cooling system, and the advance in condenser design allowing higher vacuums, etc. These are not merely possible improvements in steam-plant design, but they have actually been incorporated in some existing plants, resulting in the high economy stated. Even these economies can be exceeded where plant is installed at seaboard where colder condenser water is available, giving higher vacuum, and consequently better economy of prime mover.

Further, it is just as easy to burn a lower-gravity oil as to burn lighter oils, the low-gravity oils running greater weight per barrel and costing the same, or slightly less. On the other hand, it would not be wise to use this low-gravity oil for Diesel engines.

The fuel item, then, could easily be as much as 20 per cent. lower than assumed by the author, which would mean practically $52,000 smaller yearly fuel bill at rated load. When operating at the lower loads, his fuel bill is still further off because, as explained above, for a plant operating under these conditions one would not think of installing a one-unit plant, and hence, although the smaller unit would have a lower economy, it would be operating at full load and therefore at better economy than the large unit at fractional load. For this reason, the fuel items at fractional loads are far too high, resulting in an unfair comparison.

### *Maintenance, Etc.*

The maintenance item of a steam turbine plant is one of the smaller items of cost, as it simply means the expense of cleaning boilers, small miscellaneous apparatus, and an occasional repair, so that the figure Mr. Haas has used is exceedingly ample.

The water bill, of course, is much higher than for Diesel plants, but it is not such an item as to have very much weight in the decision of the type of plant to be installed; that is, as far as the cost of it is concerned. The question of sufficient supply is an entirely different matter.

The labor cost of operating a steam plant is less than for a Diesel,

as a steam plant requires very little attention compared with a Diesel engine; and with an oil-fired plant and automatic system of oil firing, the boiler room labor is cut down to a minimum.

## Diesel and Turbine Plant Diagrams

In addition to the tabulated analysis, Mr. Haas submits diagrams, Fig. 3, showing diagrammatically the relation between the operating costs of steam turbine plants and Diesel engine plants at various efficiencies, fuel costs and load factors. These diagrams are quite misleading in that they do not include all the operating costs. The reason given by the author for omitting these is that the remaining operating costs are either of small moment or practically the same for both types of plants. This is not true, as both maintenance and lubrication of the Diesel plant are excessively high and have no parallel in the steam plant. He states that lubrication alone varies from $2 to $4 per kilowatt-hour. The maintenance item will also run about the same rate, so that if these items were included it would make quite a difference in the comparison.

Further, the quarter-load comparison is unfavorable to the steam plant on account of the efficiency assumed. Checking back from some of the figures, it is found that the economy at quarter load is reduced about 50 per cent. over what it is at full load. This would probably be true if a one-unit plant were used, but as explained above, under such operating conditions a two- or three-unit plant would be selected, and the economy would then not be more than 15 per cent. to 20 per cent. poorer than the plant operating at full load.

## Conclusion

The writer has made some figures corrected along the above lines which show that the operating costs of both types of plant are materially different from those given by the author. At 6,000-kw. output, which is considered full rating of the plant, the total operating cost per year for the Diesel engine plant is $345,000 and for the steam plant $305,000, making 0.657c. per kilowatt-hour for the Diesel plant and 0.58c. per kilowatt-hour for the steam plant. One hundred per cent. load factor, however, is only a theoretical condition and is seldom, if ever, obtained in practice, if units are properly selected when installing plant. Three-quarters load, or 4,500-kw. output, represents more nearly an operating condition, but under this condition it is still more favorable to the steam plant, the cost per kilowatt-hour for the Diesel plant being 0.76c. and for the steam plant 0.65c.

The above is based upon using the same grade of fuel oil for both the steam plant and the Diesel. If lighter-gravity oils were used for the

Diesel, the operating cost of the Diesel plant would be still greater, and this is always a possibility as this point is not yet beyond question.

Summing up the above arguments, the author's paper is shown to give an untrue comparison of the relative operating costs of the two types of prime mover, on account of:

1. Selecting Diesel engines of far larger rating than have ever been put into successful use.
2. First cost of plant too low, resulting from this selection of units.
3. Assuming actual operating economy equivalent to builders' guarantees and not reducing this to cover results obtained in actual operation.
4. Maintenance too low, being merely an assumed figure and not checking with published records.
5. Economy of the steam plant has been assumed far lower than is actually being obtained in the latest turbine plants.
6. Fixed charges on steam plant being too high in comparison with fixed charges assumed for Diesel plant on account of same life being assumed for both steam and Diesel plants.
7. Fractional-load economy on turbine plant is erroneous on account of selecting units on the basis of 100 per cent. load factor and then operating them at fractional loads, instead of, in the case of fractional load, assuming different size units which would result in better economy and lower first cost.

While the figures used by the writer for Diesel engine operating costs are very much higher than the author's, they cannot be considered extreme for they have been compared with published records of actual operating costs, and in each case lower figures have been assumed on account of the units being larger and representing the latest development in Diesel engine design. It must further be borne in mind that in a comparison of this kind one is comparing the well-known reliability and well-known cost of operation of a steam turbine plant with a type of plant which is entirely untried for plants of this large capacity; consequently, the operating costs are entirely a matter of conjecture and in using figures somewhat better than published records of such plants, one is going as far as it is safe to go.

Aside from the question of operating costs there are other considerations which are of vital importance in the selection of prime mover for a mine plant, where any shutdown of mine due to power-plant troubles would be very serious, hence the question of absolute reliability of operation, the ease of getting sufficient supply of skilled labor familiar with the operation of the type of prime mover selected, the ease or difficulty of making repairs or getting repair parts, the suitability of the plant for the use of other fuels beside oil, are of prime importance. All of these considerations will tend toward the adoption of the steam turbine plant

rather than the Diesel plant, even if the operating costs as estimated did show up favorably to the Diesel engine. All such questions are already answered in the case of a steam turbine plant, where they are entirely problematical in Diesel engines.

Considering all these vital points, the Diesel engine must show a tremendous margin on paper in operating costs over the steam plant before the risk is justified in mine power plants.

In conclusion, the writer would refer to the statement made by H. J. Freyn, the well-known authority on Internal Combustion Engines, appearing in the *Transactions of the American Society of Mechanical Engineers* in 1911, as follows:[1]

"Since the Diesel engine owes its existence primarily, in fact almost exclusively, to its unsurpassed fuel economy, it is not surprising that its advent and development have had a singular economic significance in Germany and France and on the European continent in general, where fuel is very expensive and the standard of manufacture high, whereas in England and more particularly in America, where excellent cheap fuels abound, while skilled labor is scarce and dear, there is manifestly not the same inducement for the introduction of very costly although thermally highly economic machinery."

Also—

"Broadly speaking the outlook for the oil engine of medium size in this country seems good, perhaps not so much in the immediate future as later on after more knowledge of the advantages of the oil engine is disseminated and its good features are demonstrated; but over-enthusiasm should be carefully guarded against, since, as mentioned before, economic conditions in this country are materially different from those abroad, where the oil engine seems to have entered upon a triumphal career."

## Manganese Ores of Russia, India, Brazil and Chile

Discussion of the paper of E. C. HARDER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 113, May, 1916, pp. 761 to 798.

HERBERT K. SCOTT, London, England (communication to the Secretary*).—I am acquainted with the manganese-ore deposits of Russia, India and Brazil, and agree generally with the statements contained in the paper regarding those deposits. The information given by Mr. Harder regarding the Chilean deposits is very welcome.

When examining the Caucasian deposits, I endeavored to determine the quantity of ore originally contained in the deposit, and by deduction, the amount remaining for extraction. These calculations were rendered possible by the great extent of the outcrop and the extensive work that had been carried on in the deposit.

In making the first estimate, the quantity of manganese ore and associated sterile material in the bed was measured in a large number of

[1] Vol. 33, pp. 921 to 923.

* Received Aug. 16, 1916.

places, and the total quantity of ore originally contained in the deposit was calculated equal to 57,000,000 tons.

With regard to the second figure, it was found that only about 15 per cent. of the mineral contained in the deposit was exported owing to (*a*) the crude pillar and stall method of mining, (*b*) the large quantity of gangue associated with the ore, (*c*) the friable character of the mineral, (*d*) its low market value, and the exigencies of the purchaser.

In recent years, beginning in 1900, and more particularly since 1905, the introduction of longwall working and the stowing in the goaf of the associated sandstone, as well as the construction of numerous washing plants, has resulted in a larger proportion of the orebody finding its way to consumers. Already in 1906, workings were being reopened and pillars and poor ore withdrawn which had been hitherto abandoned, so that in all probability a large part of the ore in the old workings will be eventually recovered, although the measure in which this is done will depend upon the market value of the mineral at the time.

With longwall working 90 per cent. of the ore can be obtained, but for the purpose of calculation 75 per cent. may be estimated as likely to be realized by reason of the amount of crushed ground. Of the ore available, about 50 per cent. will be washed, with a loss of 33 per cent., so that the total quantity of ore likely to be extracted from these deposits will be as follows:

| | | |
|---|---|---|
| Mineral originally in deposit | 57,000,000 tons. | |
| 25 per cent. loss in working | 14,250,000 tons. | |
| | 42,750,000 tons. | |
| Lump ore 50 per cent. of above | | 21,375,000 tons. |
| Washable ore 50 per cent. of above | 21,375,000 tons. | |
| Less loss of 33 per cent. in treatment | 7,125,000 tons. | 14,250,000 tons. |
| | | 35,625,000 tons. |
| Quantity excavated to end of 1914 | | 13,500,000 tons. |
| Quantity of ore available for extraction | | 22,125,000 tons. |

This figure differs appreciably from that of 110,000,000 tons on page 768 of the paper.

Assuming that the value of the ore permits 1,000,000 tons per annum to be marketed after peace is declared, the deposits should be able to furnish that quantity of ore for over 20 years.

In addition to the minerals, pyrolusite, psilomelane and wad mentioned by the author, an appreciable quantity of a dull reddish mineral was mined, which although originally rejected by reason of not being black, was subsequently found to be of better quality than the general run of ore. A complete analysis of some of the first-quality stuff indicated that it was probably manganite ($Mn_2O_3.H_2O$), and as the complete analysis may be of interest, it is given herewith:

| | Per Cent. |
|---|---|
| Silica | 3.02 |
| Manganese peroxide | 48.02 |
| Manganese monoxide | 36.00 |
| Ferric oxide | 0.64 |
| Alumina | 0.91 |
| Baryta | 0.84 |
| Lime | 1.05 |
| Magnesia | 0.22 |
| Phosphoric acid | 0.337 |
| Arsenic acid | nil |
| Sulphuric acid | 0.542 |
| Oxide of copper | 0.03 |
| Oxide of lead | nil |
| Oxide of zinc | trace |
| Combined water | 8.40 |
| | 100.009 |
| Manganese (metal) | 58.25 |
| Iron (metal) | 0.45 |
| Phosphorus | 0.147 |
| Sulphur | 0.217 |

I visited the deposits of the Nicopol district in 1907, and found that manganese ore had been proved to exist over a large area. No calculation even reasonably near the truth can, however, be made regarding the quantity of mineral likely to be contained in the area underlain by manganese ore, inasmuch as the outcrops of the ore are limited in extent, and the details of the thickness and composition of the bed over the area have not, so far as I know, been obtained on a sufficiently comprehensive scale. I am convinced, however, that the quantity of mineral available is very large.

The totals given by Mr. Harder regarding the area of the deposit and the mineral available, apparently taken from the Bryschlag-Krusch-Vogt treatise, represent, in my opinion, but a fraction of the correct figures, and further do not appear to be in correct relation with each other, for an area of 20 sq. km. underlain by a manganese-ore bed 1 to 1.5 m. in thickness would not give a total quantity of mineral available, of 7,500,000 tons. Dr. N. Sokolow,[1] who studied these deposits in 1901, shows the manganese-ore area as exceeding 200 sq. km. in extent, and in one deposit seen by me, with an area of over 50 sq. km. in which many test pits and bore holes had been made, the quantity of ore available, estimated by a Russian engineer of standing, was many times greater than the total given by Mr. Harder for the whole zone.

Generally, the mineral excavated consists of 20 per cent. lumpy and

[1] Dr. N. Sokolow: Die Manganerzlager des Gouvernements Jekaterinoslaw, *Russia, Memoires du Comité Géologique*, vol. 18, No. 2 (1901).

80 per cent. small ore. The establishment of washing plants is general and approximately 50 per cent. is lost in treatment.

At one property of which "the run of mine" contained 34 per cent. Mn, the whole of the output was washed, giving the following results:

| | Per Cent. | |
|---|---|---|
| First grade | 50.00 | Manganese (metal). |
| | 8.00 | Silica |
| | 10.00 | Moisture |
| | 0.16 | Phosphorus |
| Second grade | 40.00 | Manganese (metal) |
| | 28.00 | Silica |
| | 10.00 | Moisture |
| Tailing | 20.00 | Manganese (metal) |
| | 38.00 | Silica |

Regarding the Indian deposits, I consider Dr. Fermor's comprehensive treatise from which the author quotes as the best statement regarding manganese ores hitherto published.

The quantity of ore likely to be contained in these deposits is great, particularly in those of the Central Provinces which are so large and numerous, with in many cases enormous outcrops. If Dr. Fermor's theory regarding the genesis of the deposits of the Gondite series be correct, and the ore continues in depth as at surface, the quantity of mineral available will be enormous.

The Sandur and Mysore State deposits of India classed as lateritoid by Dr. Fermor have much in common with those of Rodriga Silva and Ouro Preto in Brazil, mentioned by the author on page 788.

This class of deposit has often resulted, both in Brazil and India, in financial loss, and so possesses interest mostly of a negative character. While the outcrops of these deposits are generally striking in appearance and seem to contain large quantities of merchantable ore, exploration generally proves them to be superficial in character, the manganese ore giving place at a shallow depth to iron ore, and subsequently to the original rock.

The precise origin of the manganese and iron in these deposits, as in all those of a lateritic character, is obscure, but it is generally agreed that a large part of the material is derived from the underlying rock by the action of ground waters, although Dr. Fermor, in order to explain the formation of the larger deposits, suggests that some of the manganese and iron has come from surrounding rocks. Bryschlag[2] points out that in oxidation precipitation the iron is thrown down before the manganese, which explains the presence of the manganese ore on the surface with iron beneath.

[2] Bryschlag, Vogt and Krusch: *Deposits of the Useful Minerals and Rocks*, translated by S. J. Truscott, vol. 2, p. 852 (1916).

Analyses of Brazilian and Indian types of this class of ore, in the dry, are given below:

| | India | | Brasil | |
|---|---|---|---|---|
| | Sandur State, Per Cent. | Mysore State, Per Cent. | Botafogo, Per Cent. | Tres Cruzes, Per Cent. |
| Manganese (metal) | 40.92 | 38.24 | 41.23 | 37.25 |
| Iron (metal) | 14.95 | 16.85 | 16.23 | 19.42 |
| Silica | 1.50 | 10.40 | 3.11 | 5.40 |
| Phosphorus | 0.002 | 0.184 | 0.193 | 0.102 |

With regard to the Miguel Burnier deposit, the late A. O. Derby held that the manganese was derived from a limestone and suggested that the associated earthy ores, as he termed them, were residual from limestone. Certainly the limestone which did not show at grass and was some 15 m. from the deposit about 50 m. below surface; at a depth of 150 m. was in contact with the manganese ore bed, which was much thinner, and a similar condition was observed at Rodeio.

The extraordinary development of the Morro da Mina mine is an example, somewhat rare, of a property proving to be much more valuable than suggested by surface indications. This area remained for some years undeveloped after manganese mining was initiated in Brazil, principally because the lateritic covering of the hill consisted in great part of a partly altered spessartite rock, a siliceous manganese ore which was also found in an exploratory crosscut some 50 m. in length, driven into the hill near its base, at some remote period. In 1900, 10 years after the deposit was discovered, crosscuts were made on the hill in several places at a depth of some meters, below the lateritic cap, and the huge lenses, since developed, were discovered. Derby suggested that they were the oxidized portions of a manganese carbonate similar to that which produced the deposit of Piquery, and which differed from the more siliceous and resistant mineral-containing spessartite.

Had the explanatory crosscut at the base of the hill been continued for several meters, it would have cut one of the lenses at present being so profitably worked.

I saw the Bahia deposits some years ago, and at that time the Onha property was the principal producer. Indeed, more mineral has been quarried from it than from the Pedras Pretas property mentioned by Mr. Harder. Up to the end of 1908, I estimated that about 70,000 tons of ore had been got from these deposits and I have not heard that this total has since been appreciably increased. The Onha deposit consisted of a vertical lense, 150 m. in length, and a maximum width of 25 m. It was composed of a mixture of merchantable ore and a partly altered man-

ganese garnet rock. In depth this latter appeared to be increasing in quantity and the better ore was more difficult to obtain.

The war prevented me from contributing to the discussion on Mr. Harder's paper on The Iron Industry of Brazil, read in October, 1914, but I would ask now to be permitted to express my appreciation of the excellent work of Mr. Harder and other American engineers in the study of these deposits.

I do not, however, agree with the author when he asserts that the massive iron ores were laid down as found today and practically ignores the possibility of the massive ore having been formed by alteration from the general average of the iron formation.

No one visiting the iron-ore district can fail to notice that great changes must have taken place in the character and composition of the rock, and which are indeed still going on, although in an infinitesimal degree compared with former times.

This is recognized to some extent by the author, and his statement that the laminated ore is altered to a greater depth by surface agencies, elimination of silica, hydration, etc., than the hard ore, suggests the possibility of the massive ore having been formed in the same way, and afterward suffering dehydration.

So extensive and numerous, however, are these deposits that it is obvious to anyone seeing them that they contain enormous quantities of ore. Hence the question of their origin has little economic interest for the present generation.

Notwithstanding the evidence of a large number of analyses of samples, almost all of which are probably from outcrops, I am of opinion that these ores will be found to contain, when exported in quantities, less iron owing to hydration, and more phosphorus than is generally supposed, and in this resemble the Chilean ores, of which Mr. Harder says that a large proportion will be found to be of non-Bessemer grade.

With regard to the transport of the ore, I believe it could be carried on the Central Railway, if the line were improved and an endeavor made to handle it.

I do not consider that the Brazilian Government will assist the iron-ore export business (and it will require sympathetic treatment) to the exclusion of the establishment of steel manufacture in the country, even though as yet no coal suitable for coke manufacture has been found in Brazil.

The old rule that ore should be taken to fuel rather than the reverse is not now strictly applied anywhere—for have not iron works been constructed at Duluth to utilize empty Lake ore boats as fuel-carriers, and, further, has not the Gary plant been constructed between the ore and fuel, and following the industrial centers moving west?

Contemporaneously with the initiation of the export of iron ore from

Brazil, a commencement should be made with steel manufacture in the country. It would be easy to introduce the fuel necessary as return freight in iron-ore boats. (The success of steel works in Japan, Canada, British India and Australia will act as an incentive to Brazilians insisting that some steel should be manufactured in the country when the iron ore exports commence.)

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## Principles of Natural Gas Leasehold Valuation

Discussion of the paper of SAMUEL S. WYER, presented at the Arizona Meeting September, 1916, and printed in *Bulletin* No. 112, April, 1916, pp. 147 to 160.

F. G. CLAPP, New York, N. Y. (communication to the Secretary*).—I assume that where this valuable paper states, near its end, that "it is not possible to establish a market price," the author means that no particular price which will fit all conditions and fields can be given. I am not in agreement with Mr. Wyer, however, that the actual value of natural gas leaseholds is "largely a matter of opinion." I would even be willing to express myself very strongly as believing that if we, as experts, cannot give anything closer than a mere opinion, our testimony or reports are not of great value to our clients. I believe, moreover, that by detailed geological examinations and with our latest knowledge of geological structure, principles of natural gas occurrence, etc., we can approach just as near a certainty in natural gas valuations as we can in oil valuations, and much more so than in the valuation of many metal prospects, in which only the upper few feet of the vein has been opened. In other words, we are getting nearer and nearer every year to bringing geology as applied to oil and gas development into the class of an exact science.

Mr. Wyer's paper gives an excellent outline for methods of leasehold valuation. Being presumably intended as an outline only, it can be expanded indefinitely by a detailed discussion of the methods of valuation for the four different classes of leaseholds which he mentions: (*A*) "Producing;" (*B*) "Protective;" (*C*) "Reserve;" and (*D*) "Prospective." This classification is probably as good as can be made in practice.

In the producing leaseholds in particular it is easy to establish a market price at which these leaseholds should be bought or sold. The question would arise in any case whether or not the leases are connected with a pipe line or the distance from the pipe line. Assuming that a pipe line already taps the property and that gas is already being sold for a certain figure, this price must be multiplied by the number of thousand cubic feet still "in sight" in the ground, from which is to be deducted (*a*) bonuses, (*b*) rentals, (*c*) royalties, (*d*) cost of drilling, (*e*) taxes and

* Received Sept. 18, 1916.

insurance, (*f*) overhead charges, etc., taking into account, of course, the number of years during which the property has produced and will probably produce, and paying particular attention to the decline curve for the particular wells and the property as a whole. It is also quite necessary to consider, in producing properties, whether these have been thoroughly drilled, or whether there is room for additional wells. The distance apart of the wells and the question of whether or not they have tapped the deepest known sands is important; also the pressure of the gas, its rate of decline, and whether or not it must be pumped. All these and many more questions must be considered in valuing a lease; and when these facts are known, it is not a difficult matter for a natural gas geological engineer to place a definite figure on the property.

The real "protective leases" might be grouped with the "reserve leases," since if they contain natural gas they constitute part of the reserve supply; but if they do not contain gas, they are not necessary as a protection, and are a liability rather than an asset. Consequently, for practical purposes of valuation, they would probably be considered on nearly equal terms with the reserve leases. The value of these two groups of leases is to be figured on the basis of the number of successful wells which can be assumed for them; determined, *first*, from the positions of the reserve leases relative to the producing property, and *secondly*, on the basis of the continuity of the geological structure which has caused the accumulation of the gas.

Another factor which must be taken into account in valuing the protective and reserve leases is the degree of competition in drilling, which, if keen, will cause a serious decline of the pressure and volume of the gas. This is particularly true in such fields as the Hogshooter field in Oklahoma, the Mexia field in Texas, and certain other Oklahoma fields where a number of different gas companies were in the field, all of which picked up leases, many of them of small size, drilling them rapidly and sometimes inefficiently, exhausting the gas as fast as possible, causing a decline in pressure, with resulting influx of salt water into the sand along the lower level of the gas, and finally flooding out the entire field. These fields could have been operated for years, and their value would have been great, if possible to do so without competition; but conditions were such that an engineer on the spot could say definitely that none of the properties were of any great value, but that, on the other hand, they constituted a losing proposition. Such a field, however, if considered as a whole, would have had great value, since all natural gas engineers and geologists will unite in the conviction that monopoly is the proper thing in the natural gas business, and only by monopoly can large properties or fields be operated with financial success and economically for the community.

We therefore see, in *producing*, *protective* and *reserve* leases, that it is possible for an engineer to calculate very definitely the proper value

which should be placed upon them in order to render their purchase or sale profitable. In the case of the leases which Mr. Wyer classes as *prospective* leaseholds, the difficulty is a little greater, perhaps, since these properties are frequently situated many miles from known gas production; and to a person unfamiliar with natural gas geology, there might be no basis on which to judge their prospective success and consequent value. This is where the geological engineer comes in, and no doubt exists that our science has now reached the standpoint where we can predict the chances in such territory with a reasonable degree of success. We cannot, of course, say in any particular field, as yet untested, whether it will *certainly* contain gas; because sands may be absent, even though the correct geological structure be present; but we can say, having a group of fields or leases, about what proportion of these will be successful and what value should be placed upon them; this being fully as definite and safe a proposition as to place a value on a particular security of an industrial corporation before the latter has started its machinery. The values in such cases are based on the actual net profits "in sight," with the addition of a certain percentage for prospective future business. This additional percentage is an instance parallel to the prospective outlying territory as yet undeveloped by a natural gas company.

We see, therefore, that natural gas valuation, if not already an exact science, is approaching this limit, and that there is no difficulty, at least in the case of a particular property or company, in fixing a valuation, if the properties be studied in the necessary detail. The sellers or purchasers may not always agree on this valuation, any more than they will in a gold mine, war stock or any other commercial proposition; but, nevertheless, it is possible for each side to fix the figure which the property is worth for the purpose desired.

One of the most important questions in valuing a natural gas company, and one to which very little attention has been paid, is that of salt water in the sands. This is sometimes a serious matter, especially in fields where drilling has been close and where the sands are rapidly depleted owing to the heavy draught of gas. A field where the sand is saturated with water below the natural gas level is quite a different proposition commercially, and one not nearly so attractive, as is a field where the sands are dry. In this respect, the natural gas business differs from the oil business.

## Tungsten-Molybdenum Equilibrium Diagram and System of Crystallization

Discussion of the paper of Zay Jeffries, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1225 to 1236.

A. G. Worthing, Nela Park, Cleveland, Ohio (communication to the Secretary*).—The paper by Mr. Jeffries is very interesting. He is to be commended for his pioneer work in the study of equilibrium systems at the very high temperatures which one obtains with tungsten and molybdenum.

There is one statement, however, to which I must take exception. On page 1227, a statement is made which would seem to indicate that the Nela Research Laboratory joined Dr. Langmuir in the recommendation of 3,300°C. as the most probable melting point of tungsten. This is not the case. In the paper referred to by Mr. Jeffries, I have given 3,630°K. or 3,357°C. as the melting-point temperature for tungsten. Confirming observations by other workers in our laboratory justify the belief that this latter value is correct to within a small part of the discrepancy between the two values quoted.

Mr. Jeffries recognizes the possible introduction of errors in his calibration diagram in which he has plotted temperature as a function of the per cent. of tungsten fusion wattage. A different method of procedure which has merit would be the actual determining of points for the curve between the melting points of molybdenum and tungsten with the aid of a tungsten wire operated at intervening temperatures. It is perfectly possible by means of optical pyrometry, when once a temperature calibration has been established, to determine within a few degrees at what temperature such a wire is being operated. There are two or three such calibrations to select from. Of these the writer has considerable confidence in his own which is reported in abstract in the paper already referred to. Whichever one Mr. Jeffries might select, the procedure would seem to be better than that actually used. It would be interesting to note in this connection whether or not, with the use of such a calibration, the melting-point determination for molybdenum would consistently fall on the curve obtained with the tungsten wire.

In spite of our criticism and suggestion, we wish to express our high appreciation of this work.

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* Received Sept. 14, 1916.

## An Explanation of the Flotation Process

Discussion of the paper of A. F. Taggart and F. E. Beach, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1373 to 1386.

Oliver C. Ralston, Salt Lake City, Utah (communication to the Secretary*).—This paper has appealed to me as being one of the most lucid, well-connected and complete papers on this subject which has been published. To be sure, the concepts called upon in an explanation of the flotation process by these authors are mostly old ones which have been more or less discussed in previous literature, but they have nevertheless been assembled in a most constructive and suggestive manner. We find a new contribution to flotation theory in the effect of fine particles increasing the viscosity of the interfacial film between a liquid and a gas. This phenomenon has been variously hinted at but never definitely stated in accepted scientific terms.

The discussion of "contact angles" is very clear and will doubtless be welcomed by those who have not had the opportunity to investigate the literature on this subject, but it is unfortunate that the "hysteresis of the contact angle," observed by Sulman and reported some years ago, is not mentioned or explained.

A few other points need discussion in order to call attention to the fact that, while this paper is the best discussion of the subject which has yet appeared, all of the truth has not yet been told.

On page 1385, in discussing the reasons why heating a pulp often allows the recovery of a higher-grade froth concentrate, four different factors are mentioned. One of these factors deemed probable by the authors is that there is an "increased number of air bubbles formed by the air released from solution." While it is commonly known that heating a solution expels the dissolved air or other gases, to one who has seen the great volumes of air that enter the pulp in a flotation machine it is hard to believe that the pitifully small number of additional air bubbles released by heating the solution can have such a good effect. I would call attention to the following more probable explanation of this fact: the viscosity of water at 80°C. is less than one-fifth of the viscosity at 0°C. In other words, on heating an ore pulp the fluidity of the water in the pulp is very greatly increased and much less water containing entrained gangue will be carried along with the froth during flotation. In other words, a cleaner concentrate is obtained. The surface tension and other physical properties of water are not altered in anywhere near the same degree by change of temperature, and I would

* Received September, 1916.

suggest that the viscoscity effect is probably responsible for the major portion of the improvement.

I notice that in their discussion of the Potter-Delprat process the authors claim to have observed microscopically the condition of the solid particles in the froth, and find them completely "within the films and at no point in contact with gas." The manager of one of the large Australian companies using flotation told me that two physical chemists in his employ consumed about 6 months' time in an attempt to definitely find out whether the mineral particles were completely submerged in the film or whether they were at some point in contact with gas. In view of their failure to reach a decision, I am inclined to ask the present authors if they are certain of this observation.

Further, we notice that electrical forces are not called into question in this discussion of flotation, although the authors acknowledge that the potential surface energy in the contact surfaces between dissimilar substances might well include electrical forces which might be called on in explanation of certain aspects of flotation. However, they feel that "*Once in contact* the preferential adhesion . . . to a sulphide surface in the presence of water is sufficient to account for the persistent attachment of the sulphides to the bubble films." I feel that they are justified in this opinion, but having seen particles migrate under the microscope when in an electrical field, and knowing that the various kinds of particles and bubbles carry electrical charges, may I not suggest that the electrical charges assist the various finely divided phases to *get in contact?* The authors consider that mechanical agitation, etc., is sufficient to get the various phases into contact, but the acceptance of these electrical phenomena in no way damages the theory which they have presented and merely adds another contributing factor to the theory.

Finally, the authors state that they have not appealed to colloidal phenomena in this discussion. This statement, in the face of the fact that the largest single section of the paper is headed "adsorption," and that most of the forces called upon are those of "*capillarity,*" is hard to understand. While I agree with the authors almost entirely, in so far as they go, I find that their paper is filled with something very closely akin to colloid chemistry. The study of capillary forces comes under the head of physics and the study of the effect of chemicals on these capillary forces comes under chemical physics or physical chemistry. Now the physical chemistry dealing with capillary phenomena is known as colloid chemistry, and Freundlich even went so far as to entitle his text on colloids, *Kapillar Chemie.* So it can be seen that while I am delighted with the work these authors have done in formulating their theory, I am disappointed in the way they have attempted to name it. I would say that it can be accepted as part of a *colloidal theory* of flotation.

## The Flotation of Minerals

Discussion of the paper of ROBERT J. ANDERSON, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1119 to 1136.

OLIVER C. RALSTON, Salt Lake City, Utah (communication to the Secretary*).—The literature on the theory of flotation has been enriched, of late, by the views of a number of excellent mining engineers who unfortunately were tyros in physical chemistry and physics. Hence the obscurity and mystery with which the process is supposed to be surrounded.

The present paper goes far toward explaining some of the ideas which have seemed obscure or which have been poorly expounded by the men who advanced them, but there are a number of places where the author's explanations are unsatisfactory, or where he has fallen into the same mistakes made by the original propounders of the theories involved.

While recognizing the value of constructive compilation, it is nevertheless disappointing to find a paper reviewing the recent theories of flotation which presents nothing new—not even a new viewpoint.

At the bottom of page 1121, there are stated to be present the following "phases" in flotation: "solid-liquid (ore-water), solid-liquid (ore-oil), solid-gas (ore-air)," etc. The writer could not possibly have meant to apply the word "phases" to the ideas involved. Each of the *pairs of phases* mentioned has an interface in which exists the interfacial tension that he is talking about.

On page 1127, in discussing electrified bubbles, it is stated that "the bubbles in flotation are simply air spaces contained by a mantle of oil or of water and there is, therefore, nothing within to bear the charge. In case it would carry a charge, which would only be possible by the presence of contained ionized gases or water vapor, the charge would be speedily dissipated by contact with the interfacial boundary. Then in order for a bubble to carry a charge it must be protected by a dielectric film." It can be seen from this series of arguments that the author has fallen into the same error that has confronted a number of exponents of an "electrostatic theory" of flotation. The theory of Bains,[1] which is objected to by Fahrenwald,[2] holds that the friction of the liquid and solid and gaseous masses on each other generates static electric charges and that the function of the oils used in flotation is to insulate these charged particles. Fahrenwald's objections to this theory are sound. However, all these writers have failed to consider the mechanism of attachment of electric charges to

* Received September, 1916.

[1] T. M. Bains, Jr.: Electrical Theory of Flotation, *Mining and Scientific Press*, vol. 111, pp. 824 and 883 (Nov.-Dec., 1915).

[2] F. A. Fahrenwald: Electro-statics of Flotation, *Op. cit.*, vol. 112, p. 375 (March, 1916).

colloidal particles. They apparently have not understood how suspended particles, including air bubbles, can carry electric charges when suspended in a conductive solution. I have already called attention to the fact[3] that the *film* of liquid in contact with some other phase (gas, liquid or solid) has different properties from those of *bulk* water (or liquid). Mr. Anderson has been kind enough to quote some of the properties of this film, such as its thickness, density and tension, but has failed to remember that I also gave the difference in potential between its outer face and the bulk liquid, even when that liquid is highly conductive. Powis[4] has very recently expanded on the conception of charged colloidal particles and the theories of colloid chemists as to how such particles can hold electrical charges. Briefly, the charges are due to absorbed ions in the interfacial film. Any electrolyte in the solution is more or less ionized, and in case one of the ions is absorbed in the film more than the other, an apparent electrostatic charge is the result. In fact, the electric charge could not exist if the solution in question were not ionized, and hence conductive. It is a strange fact that the conductivity of the solution places serious obstacles in the way of the theory of Bains while conductivity (or ionization) is necessary to explain electric charges on the particles as viewed from the colloid-chemical or "capillary-chemical" standpoint.

On account of such serious misunderstandings as those above mentioned, it would seem that there is still room for the writing of a rigid critique on the theory of flotation.

As Mr. Anderson says, "in the line of development of flotation theory there has been steady progession of thought and a remarkable increase of knowledge in the past few years." That is just the trouble—the progression has been mainly in thought and too little in laboratory investigation. Too many have been willing to contribute *thoughts* to the technical press and too few have busied themselves with experimental measurements to prove the theories proposed.

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[3] O. C. Ralston: Why Do Minerals Float? *Op. cit.*, vol. 111, p. 623 (October, 1915).

[4] F. Powis: Transference of Electricity by Colloidal Particles, *Transactions of the Faraday Society*, vol. 11, p. 160 (April, 1916).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Erosion of Guns—The Hardening of the Surface

BY HENRY FAY,* PH. D., D. SC., CAMBRIDGE, MASS.

(New York Meeting, February, 1917)

THE erosion of guns is a complex problem which can be solved only by a detailed study of all the factors involved. In the present paper it is proposed to submit the results of observations and experiments which have extended over several years. Certain facts have been established and preliminary reports have been published.[1] Conclusions have been reached since then which, it is believed, are worth presenting for the purpose of promoting discussion, and with the hope that ultimately the whole truth will be known. The particular phase of the problem which will be presented here is the hardening of the inner surface of the gun tube.

It has been known for a long time that after firing a large caliber gun for some time, the surface of the metal becomes hard and brittle, cracks, and wears away.[2] The character of this phenomenon may be learned by an examination of some results which are typical, obtained from a detailed study of a 12-in. gun. The gun was trepanned and rings representing various sections of the gun were cut out for examination. The first ring represented the metal at the muzzle end, and the other rings were removed at approximately 120, 240, 325 and 337 in., respectively, toward the breech end. The appearance of the surface of the metal in the different sections is shown in Fig. 1. The greatest amount of wear and maximum amount of hard surface layer was found in section marked *E*, which is nearest the powder chamber and at the beginning of the rifling; the maximum amount of cracking in *D*; the maximum amount of copper deposition in *C*. Section *B* showed heat cracks on both lands and grooves, and section *A* showed heat cracks *on the driving edge of the lands only*. There was much flow of metal in both sections *E* and *D*, and there was progressively less effect noticeable toward the muzzle end. The polished cross-section of these same pieces is shown in Fig. 2. The

* Professor of Analytical Chemistry, Massachusetts Institute of Technology.

[1] Tests of Metals, Watertown Arsenal, 1913 and 1914.

[2] This action is not confined to large caliber guns exclusively but will occur also in smaller guns, although, as will be shown, the action is much accelerated with the increase in the weight of the projectile.

extent of the erosion is greatest nearest the powder chamber, and diminishes toward the muzzle end.

The depth of heat crack varies irregularly in each section, and somewhat regularly in sections proceeding from *A* to *E*, increasing in depth toward the powder chamber. The number of cracks does not, however, vary in the same way. In section *E*, where the greatest erosion has taken place, the smallest number of cracks appear.

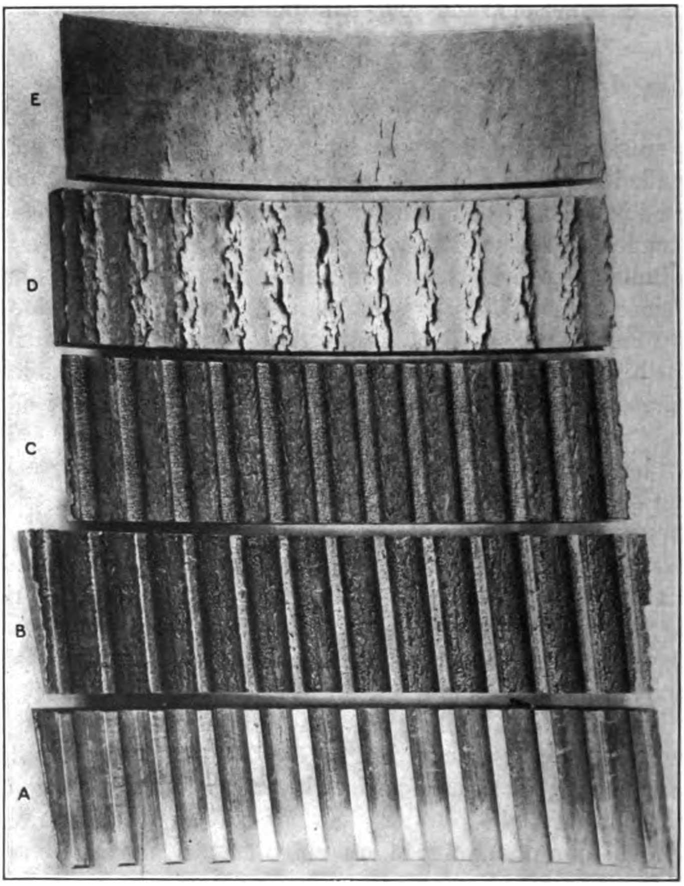


FIG. 1.

The greatest depth of crack is found in section *D*, with an average depth of 0.038 in. The width of crack in this section varies between 0.001 and 0.015 in. In section *E* the depth varies between 0.024 and 0.039 in. In section *C* the average depth of crack is only a trifle less than in *D*, but the width is not nearly so great. The cracks diminish progressively in depth and width in sections *B* and *A*, the latter section showing practically no cracks on the face normal to the axis of the gun. On a

face parallel to the axis of the gun and cut along the driving edge, section *A* shows some clean, sharp cracks about 0.01 in. in depth.

Sections *B*, *C*, *D* and *E* show a hard layer of metal on the face cut normal to the axis. All five sections show the hard layer on the face cut parallel to and along the driving edge. The depth of the layer of hard metal on the face normal to the axis varies as follows: *A*, 0.0000 in.; *B*, 0.0004 in.; *C*, 0.0008 in.; *D*, 0.0013 in.; *E*, 0.0015 in.

The cracks are undoubtedly due to the unequal expansion and contraction between the hard and soft layers. When subjected to strain

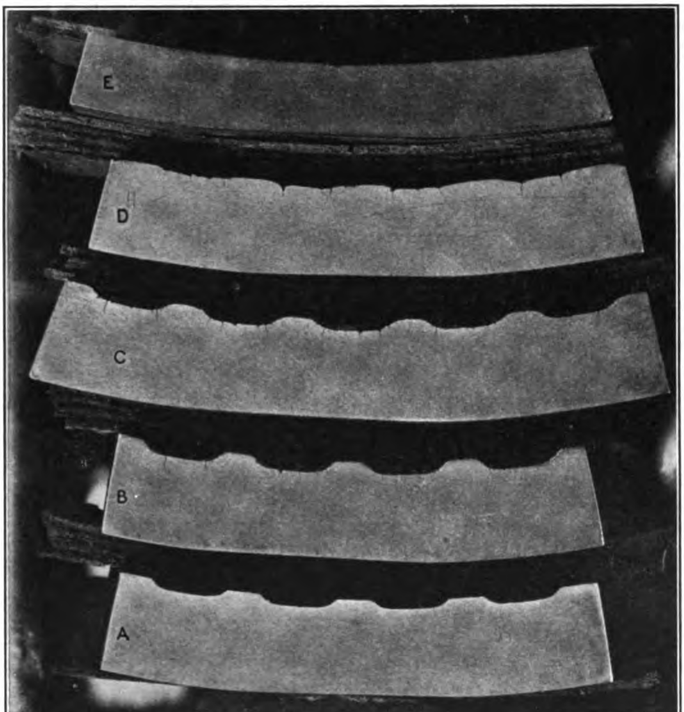


FIG. 2.

the hard, brittle surface develops cracks, and the cracks penetrate into the soft metal. The section showing the greatest thickness of hard layer ought to show the greatest depth of crack, but other influences counteract this effect.

The microscopic appearance of a surface showing cracks is shown in Fig. 3, and in Fig. 4, the latter a characteristic structure always found at the junction of a land and groove. The interpretation of these photographs will be considered in detail after discussing the cause of the formation of the hard layer. In addition to the hardening which takes place in the gun tube proper, hardening and cracking of the surface takes

FIG. 3.—× 50

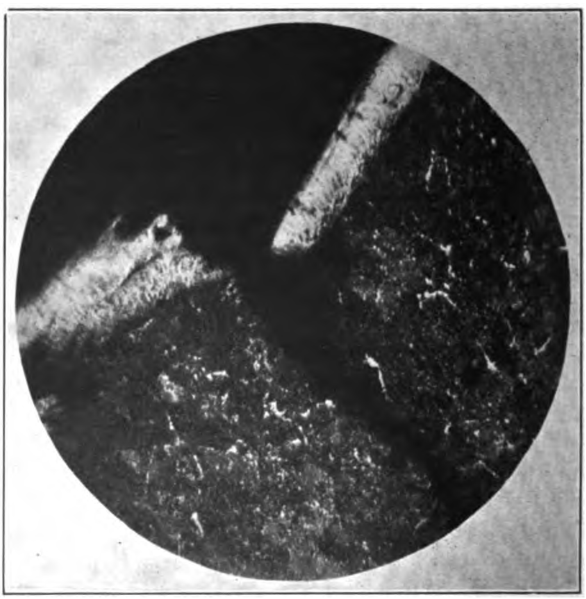

FIG. 4.—× 50.

place on the pressure plugs which are inserted in guns for the purpose of measuring maximum powder pressures. The pressure plug shows hardness around the edge of the central hole, around the edge of the plug itself, and shows the effect of hardening in the form of cracks all over the surface, but particularly in the vicinity of letters which have been stamped on the surface. Any theory which may be offered to explain the hardening of the gun tube proper must also explain the hardening and cracking of the pressure plug.

While these results are in general typical of what takes place, there are variations in individual cases. For instance, the metal on the driving edge may not have been affected as it was in the 12-in. gun referred to, but it is merely a matter of time until the conditions as described are obtained. The action is a progressive one and is dependent upon the number of times the gun is fired.

### *The Cause of Hardening*

Hardening may be produced by three well-defined methods: (1) cementation; (2) heat treatment; (3) mechanical deformation or cold work.

*Cementation.*—The principal products of combustion of the powder are carbon dioxide, carbon monoxide, water, and nitrogen. The equilibrium phenomena are very complex and will vary with the composition of the powder, the temperature and pressure. It is conceivable, however, that large amounts of carbon monoxide may be formed, and under the conditions of high temperature and pressure cementation could take place. Giolitti[3] has shown carbon monoxide to be an ideal carburizing compound, and on repeated firings small increases of carbon to the surface might take place. This would produce a hard, brittle layer on the surface richer in carbon than the original metal.

*Heat Treatment.*—When a piece of steel is heated above its critical temperature, $Ac_1$, and then suddenly quenched, it is hardened, the depth of hardening being dependent upon the temperature of heating and the rate of cooling. The temperature of combustion of the powder is sufficiently high to heat a skin of the metal above the $Ac_1$ point, and hardness would result if the surface were cooled with sufficient rapidity. The relatively low temperature of the large mass of steel would produce a rapid loss of heat.

*Mechanical Deformation or Cold Work.*—It is a well-known fact that when any ductile metal is subjected to mechanical deformation it becomes hard and brittle, and if the work is carried too far, cracking will result. Even the most ductile metals, such as gold and platinum, lose their plasticity and become hard and brittle when subjected to drawing, hammering, etc. In wire drawing it becomes necessary to anneal the metal

[3] *Journal of the Iron and Steel Institute*, vol. 84, p. 307 (No. 2, 1911).

frequently in order to restore plasticity and prevent it from cracking. This phenomenon is common to both ductile and brittle metals.

The mechanism of this action has been studied in detail by Beilby[4] and his conclusions have been confirmed and accepted by many other investigators. According to him, a surface skin may be built up by mechanical movement which gives unmistakable evidence that the surface must have passed through a state in which it possessed the perfect mobility of a liquid. This surface possesses distinctive properties which differentiate it from the surface beneath it. Hardening thus results from the formation at all the internal surfaces of slip or shear of mobile layers similar to those produced on the surface by mechanical movement. These layers retain their mobility for a brief period only, and then solidify in a vitreous amorphous state cementing together all of the surfaces of slip or shear throughout the mass.

Such a surface produced by polishing, burnishing, drawing, hammering, or cold work of any kind, possesses the property of hardness and brittleness. Metal in this condition may be restored to its natural crystalline, ductile state by annealing.

Having considered the manner in which hardness may be produced, it becomes necessary to examine the different methods in detail, and to see by which method the facts can best be explained. It would seem at first sight to be rather difficult to prove whether or not cementation had taken place, as the maximum thickness of layer which had been hardened amounted to only 0.0015 in. It was obviously impossible to cut off this layer for analysis, not only on account of the thinness, but also on account of the hardness.

Microscopically the appearanee of this surface suggested cementite, as it remained bright after etching, as shown in Figs. 3 and 4, and, furthermore, in no case was there any evidence, even under high power, of martensite, which would also remain brighter than the original structure. Etching with boiling sodium picrate, which darkens cementite, gave negative results. The hardness of the layer and its appearance would indicate free cementite, but this test would, therefore, appear to be conclusive that the surface cannot contain any free or excess cementite. The test was carried out always with a control piece which contained about 1.25 per cent. of carbon, and therefore known to contain free cementite. Under all conditions the control gave positive results. There is some doubt as to whether or not a small amount of cementation takes place, and experiments are in progress which, it is hoped, will prove this point definitely.

While it seems fairly clear that the hard surface layer is not free cementite, judging from the sodium picrate test as evidence, it would have been difficult on this basis to explain the fact that the hard layer

[4] *Journal of the Institute of Metals*, vol. 6, p. 5 (No. 2, 1911).

occurs only on the driving edges in the muzzle end of the gun. Selective formation would not seem probable.

That hardness might be produced by raising the temperature above $Ac_1$ and suddenly cooling seems entirely possible, and most of the facts can be explained on this basis. The temperature within the gun at the time of the explosion is far above $Ac_1$, in fact may be above the melting point of the metal. A thin skin of metal is heated for a brief period during the explosion, and before the heat can penetrate any distance, it is cooled by the mass of metal back of this layer. Such a hardening ought to produce the microscopic constituent martensite, but this structure in its characteristic needle-like form is never found, although troostite

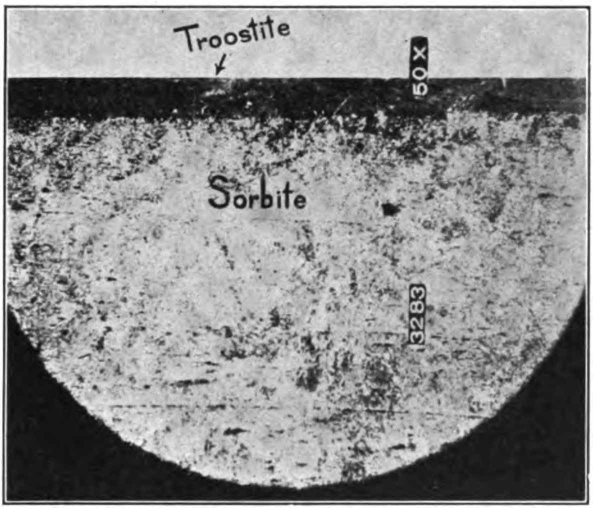


FIG. 5.—× 50.

is frequently found. Rosenhain[5] and Belaiew[5] have reported martensite in the hardened surface of the rifling of a gun, but so far in the examination of many gun fragments, and in trepanned guns, the characteristic martensitic structure has been missing. In some experimental steels which had been subjected to erosion tests by allowing the gas of an ignited charge of powder to escape through a tube of the metal, there was found not only well-characterized martensite but also troostite adjacent to it. The essential difference in this case is that there was no projectile to produce mechanical deformation on the surface.

That the hard surface is actually composed of martensitic material is proved by the fact that on tempering it passes over to characteristic troostite. Samples were cut out of the ring from section $E$, and heated

[5] Rosenhain, Belaiew: International Association for Testing Materials, New York, 1912, *Proceedings*, vol. 2, section A, p. 127.

for 15 min. to 300°, 400°, 500°, 600° and 700°. The specimen heated to 300° showed troostite development most markedly and unmistakably, as shown in Fig. 5. The 400° and 500° samples also showed troostite, but less well defined, the transformation having progressed further, and there was almost complete reversion in the specimens heated to 600° and 700°. This experiment seemed to prove conclusively that the hard surface was martensite, but threw no light upon the fact that in the muzzle end of the gun the surface was hardened *on the driving edge only.*

It was thought for a time that mechanical deformation or cold work produced by the rotating projectile would account for all of the phenom-

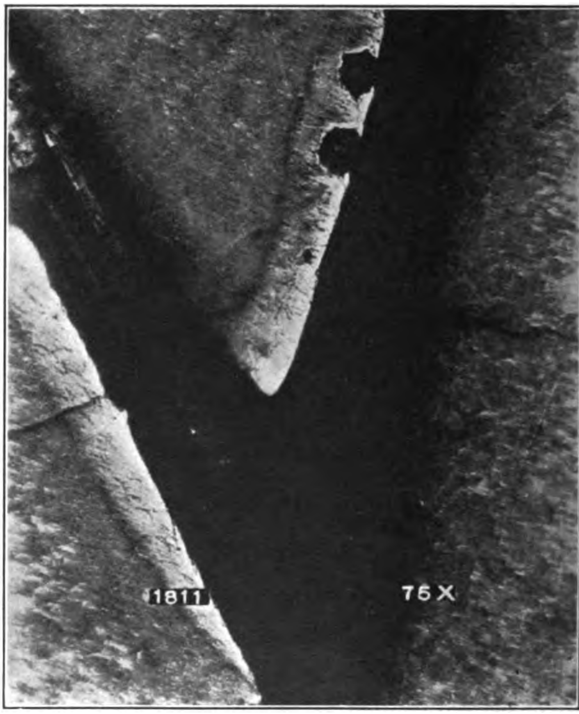


FIG. 6.—X 75.

ena, but this idea was negatived by (1) the martensite→troostite conversion; (2) by the fact that the powder chamber itself, untouched by the rotating projectile, showed hardness; (3) and by the fact that the pressure plug showed hardness.

That mechanical deformation has much to do with the production of hardness is shown by the selective hardening on the driving edge of the land on the muzzle section, and by some experiments which were conducted on pressure plugs.

Knowing that pressure plugs harden in the same way as the surface of

the gun tube itself, it was decided to experiment with pressure plugs, and to follow the development of the hard surface layer in its various phases. For this purpose, a plug having approximately the same composition as gun steel was prepared for service. The metal was carefully analyzed, it was heat treated, and photographs were made of the fine sorbitic structure. The surface as a whole was uniform. The prescribed lettering was stamped on the surface. It was then placed in service, and after having been fired until it showed signs of heat cracking, was returned for microscopical examination. The face was polished and etched, and the

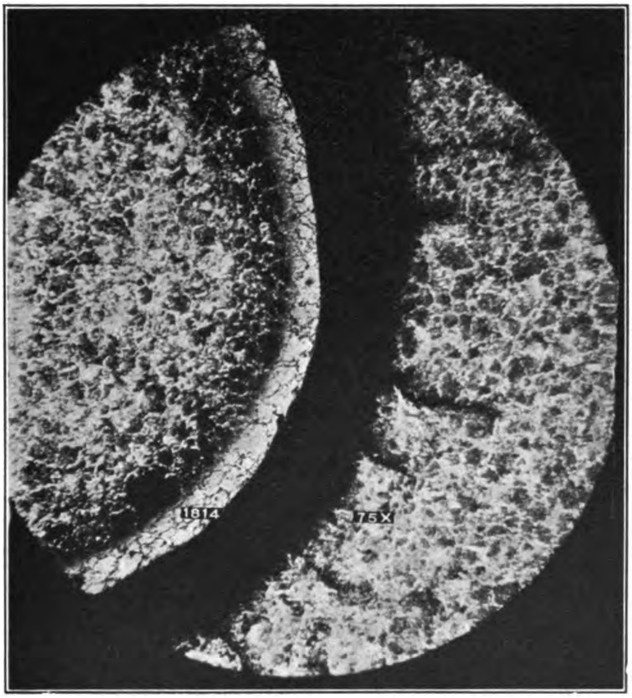


Fig. 7.—× 50.

structure seemed to be normal, but it was seen on detailed examination that the hard surface had begun to appear on the edge of the center hole and on some of the letters. Examination showed that this development had taken place at those points where the *maximum amount of cold work had been applied in stamping the letters.* This is shown in Figs. 6, 7 and 8, representing the letters A, O, and a period respectively. In the letter A and in the period the die was apparently not held in a truly vertical position and one side of the letter was cold-worked more than the other. Selective conversion of soft to hard metal has taken place at these points. This is shown in the lighter area of Fig. 6, and also of Figs. 7 and 8,

the light area in each case being the hard area. Observations of a similar character were made on almost every letter on the face of the pressure

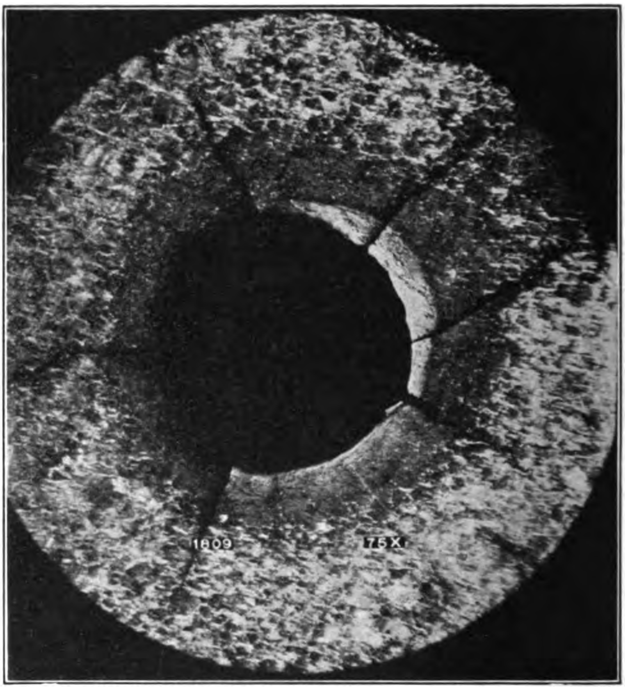


FIG. 8.—× 50.

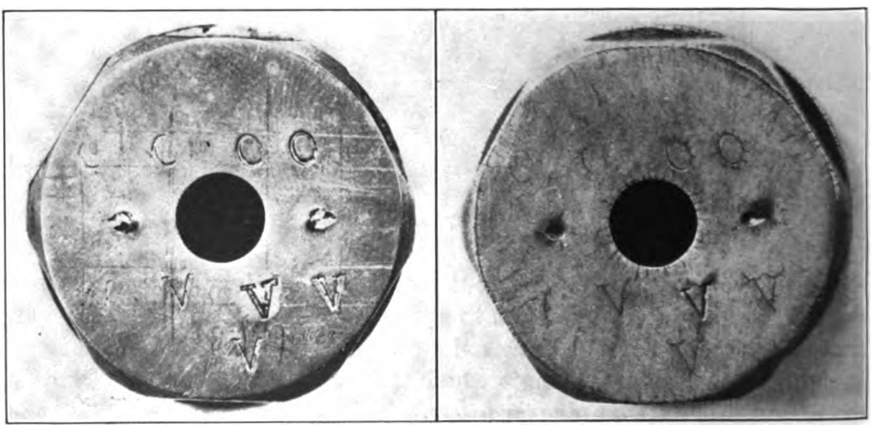

FIG. 9.—PRESSURE PLUG BEFORE HAVING BEEN PLACED IN SERVICE AND THE SAME PLUG AFTER HAVING BEEN IN A GUN WHICH HAD BEEN FIRED 100 ROUNDS.

The surface of the metal (right) has been worn away by some volatilization of the metal and by the polishing necessary to produce the structure.

plug, each showing partial conversion from soft to hard structure at the point of maximum cold work.

This evidence seemed to favor the view that cold work was an important factor in the martensitization[6] of the metal.

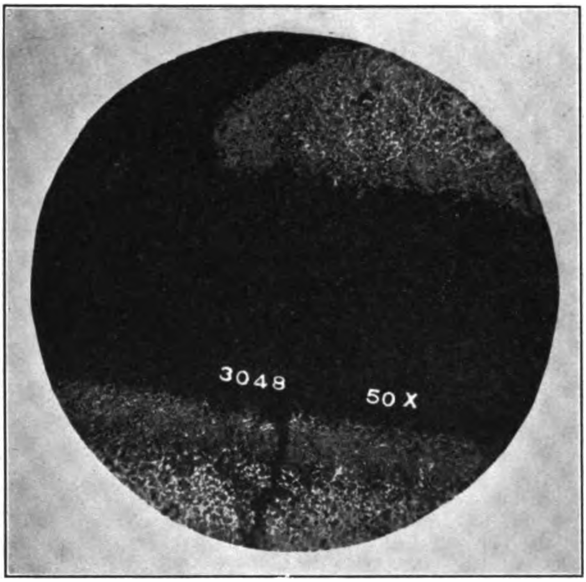


Fig. 10.—Letter V Impressed at 5000 Lb. × 50.

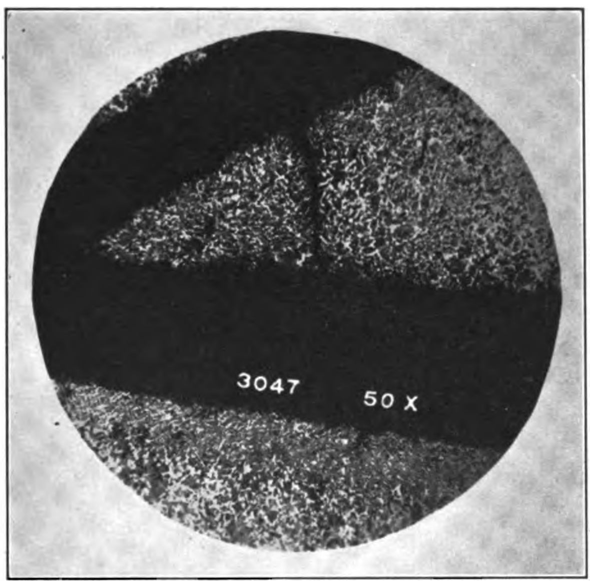


Fig. 11.—Letter V Impressed at 2000 Lb. × 50.

[6] The process of producing hardness by converting the free iron carbide into solid solution of iron carbide in iron. This is commonly done by quenching the steel from above $Ac_1$.

In order to test this view systematically, a second pressure plug was prepared and the letters V and O were stamped on the face of the plug

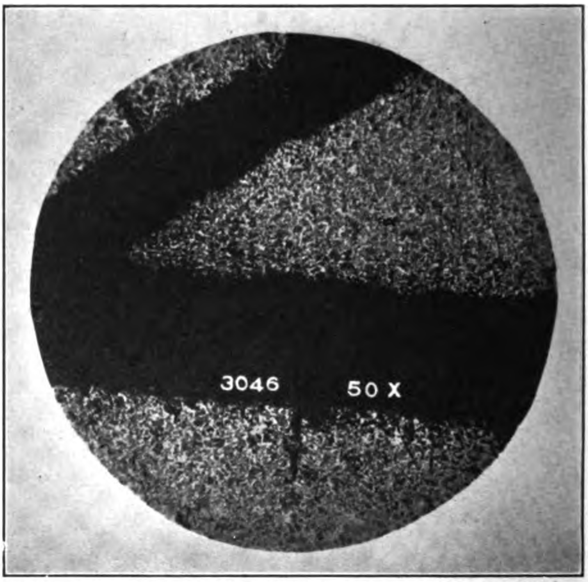


FIG. 12.—LETTER V IMPRESSED AT 1500 LB. × 50.

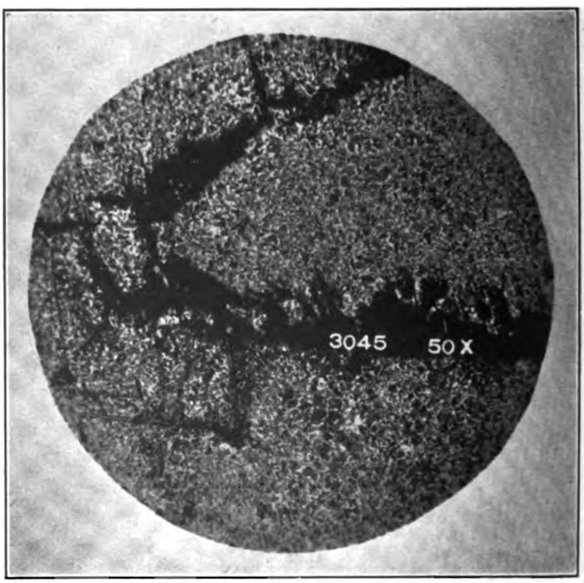


FIG. 13.—LETTER V IMPRESSED AT 1000 LB. × 50.

at varying pressures. This was done by holding the stamping die in one jaw of a tensile testing machine and the plug in the other jaw. The

machine was now put into compression and the letters stamped at varying pressures, from 500 to 2,000 lb., as shown in Fig. 9. By mistake, one of the letters V was stamped at 5,000 lb. instead of 2,000 as originally intended.

The pressure plug was then put in service and the gun was fired 100 rounds, when the plug was returned for examination. The original sorbitic condition of the metal was found to have changed to the troostitic condition at those points of maximum cold work and to have remained practically unchanged at other points. Thus the first step in the change: sorbite→troostite→martensite had been accomplished. Figs. 10, 11, 12 and 13, show the letter V at 5,000, 2,000, 1,500 and 1,000 lb., respectively.

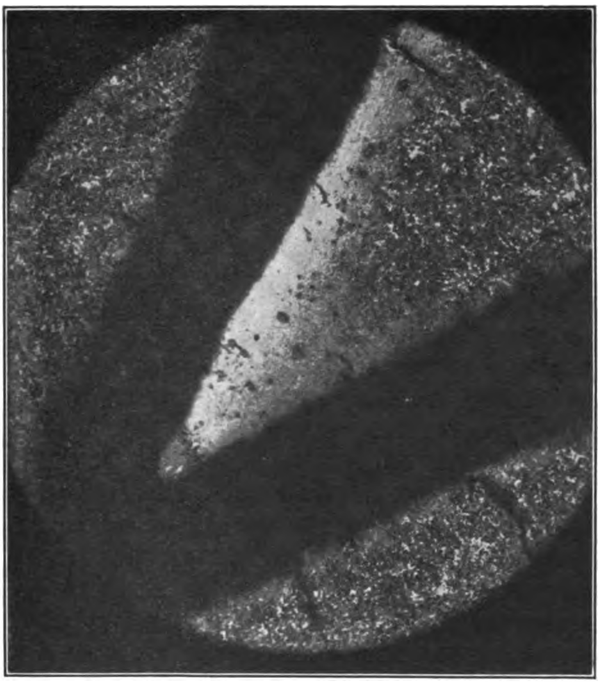

Fig. 14.—Letter V Same as Shown in Fig. 10, but after Firing 200 Rounds. × 50.

In Fig. 10, the troostitic character of the metal is clearly shown on the edge and apex of the letter V. The letter impressed at 2,000 lb. also shows the change, but not advanced so far as in the letter of greater pressure. At 1,500 lb. the change has begun but the advance has not been far, and at 1,000 lb. pressure the change is not noticeable.

In the same plug similar changes were noticed on the edge of the central hole, on the outer edge of the plug, and on two periods stamped at an angle of 45°. The period was strongly cold-worked on one side and the change was well-marked.

The plug was again placed in service for 100 rounds and then returned for examination. The result is shown in Fig. 14, which is the letter V impressed at 5,000 lb. It will be noticed that the transformation has proceeded to its final stage, viz., the light etching structure which had previously been shown to be martensite.[7]

These experiments seem to prove that the hardness of the surface of a gun tube as well as the pressure plug is due to a combination of mechanical deformation and a process of martensitization. This process requires time, and the development takes place in stages. Even in the early stages there was evidence of heat cracking before the change of structure had taken place. When the structure begins to change it first shows as troostite, and then develops into martensite, but the latter is amorphous[8] in form. Temperature and work are also important factors. The transformation takes place with the greatest rapidity where the work has been most severe.

These factors counterbalance each other in the gun to a certain extent. In the powder chamber there is the maximum temperature, but the minimum amount of work. The only work which has been done upon the metal here is the original machining of the gun. In section *E* where the rifling begins, the maximum amount of cold work is done. The rotating band is undeformed and the whole surface bears on the rifling. This is also true of section *D*, but in each section toward the muzzle there is a decreasing amount of work due to the grooving of the rotating band until at the muzzle end the driving edge alone receives work.

The reason that a cold-worked metal is more susceptible to the sorbite→troostite→martensite change when exposed to firing is not clear. There may, however, be two factors at work: 1. Beilby has shown that mechanical deformation of any kind produces an amorphous state in the metal, and it is entirely possible that the amorphous metal may be more sensitive to the reaction, *i.e.*, it may have greater solvent power for iron carbide. He has shown, and many others have confirmed the observation, that such metal has a greater solution pressure and it is possible that it may have greater dissolving power for iron carbide.

2. Spring has shown that certain reactions, which ordinarily take place when under the influence of heat, can be made to take place at the ordinary temperature when under the influence of pressure. Thus he caused dry sodium carbonate to react with dry barium sulphate with the formation of sodium sulphate and barium carbonate. He also caused

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[7] There seems to be a difference in the degree of magnification of Figs. 10 and 14, but this is due to a wearing away of the metal by volatilization and polishing, thus exposing the deeper and narrower part of the groove of the letter.

[8] Capable of being easily converted into troostite, but showing none of the characteristic needle structure.

powdered zinc to alloy with finely divided copper to form brass. In each case precautions were taken to eliminate any heat effects.

In a gun the original metal is sorbitic and the iron carbide is therefore mostly in the free form, only a minimum amount being held in solid solution. When subjected to the pressure of cold work, or the pressure of the explosion at a high temperature, solution may take place, each round of firing adding to the amount which goes into solution. This would lead naturally to the formation, first, of troostite, and when the solution is complete, of martensite. These steps are actually observed.

Either hypothesis—solution in amorphous metal, or solution by pressure—may explain the phenomena observed, but the latter seems the more likely. This would more adequately explain the fact that in erosion tests of steels of various composition, those steels high in nickel or manganese show the greatest amount of martensite formation. In these steels the $Ar_1$ point is lowered slightly, thus increasing the tendency for the carbide carbon to be held in solid solution form. Thus, in one case, a 3.50 per cent. nickel steel showed a martensite surface 0.05 in. in depth, while under exactly similar conditions a plain carbon steel showed only 0.004 in. Nickel and manganese lower the $Ac_1$ point, and hence make it easier for the sorbite→troostite→martensite transformation.

Experiments are now in progress with alloy-steel pressure plugs to see whether or not the hard surface develops more rapidly with them than with plain carbon steels. It is highly probable that those steels having the highest $Ac_1$ point will show hardening the least rapidly. A convenient means of testing this is to make up a series of steels into the form of small screws, stamp some letter at 5,000 lb. pressure, place them in the mushroom head of the breech block, and make observations after firing.

The preceding work was carried on at the Watertown Arsenal and the writer wishes to express his appreciation of Colonel Wheeler's interest and his courtesy in allowing the data to be published.

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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

# Problems Connected with the Recovery of Petroleum from Unconsolidated Sands

BY WILLIAM H. KOBBÉ*, ST. LOUIS, MO.

(New York Meeting, February, 1917)

## I. INTRODUCTION

The word recovery as used in this paper is applied in its broader sense and not limited to wells producing from horizons of unconsolidated sands. Certain problems connected with the winning of petroleum from such horizons present themselves while drilling is in progress and should be considered before drilling is commenced wherever unconsolidated sands are expected. Maximum recovery depends not alone upon efficient pumping methods but also upon the selection of a proper drilling system and the completion of a well especially adapted to the extraction of these oil-bearing sands.

## II. DRILLING

### Determining Factors in Selection of Method

The selection of a drilling method for the development of a region in which the oil reservoir consists of unconsolidated sands is based on the following considerations: (1) Character of the formations overlying the reservoir; (2) thickness of the oil-bearing stratum or strata; (3) thickness and character of water or gas sands; (4) gas pressure to be encountered or expected; (5) total depth of well.

#### 1. *Character of the Formations Overlying the Reservoir*

The strata overlying the reservoir may consist of beds of shales, clays, sands and other soft materials or they may be slates, limestones and sandstones consolidated to all degrees of hardness. Very often the overlying strata are predominantly of soft materials but interbedded with harder formations. Likewise, soft strata are encountered at different horizons in an overburden consisting almost entirely of the harder sedimentaries. The physical character of this overburden is frequently

* Petroleum Engineer, Pierce Oil Corporation.

complicated by the occurrence of quicksands, gypsum, rock salt or iron pyrites.

In deciding upon the surface equipment best adapted for the penetration of these various formations composing the overburden, the governing factor is based upon evidence or expectations that the strata are: (*a*) predominantly soft, (*b*) predominantly hard, or (*c*) about equally divided.

If it is known that the overburden consists of predominantly soft and cavey material such as sands, clays, soft shales, quicksands, beds of gypsum, etc., a rotary drilling rig of a size and type commensurate with the depth would be the most efficient method of overcoming such conditions.

On the other hand, if it is known that the overburden consists of predominantly hard strata such as limestones, sandstones, hard shales and slates, cable-tool equipment would be employed and a standard rig, with or without calf wheels, erected.

In many parts of the world it is impossible to classify the overburden either as predominantly soft or as predominantly hard. It is made up of many strata of each class. Such conditions are typical of the deep and difficult drilling in certain parts of California, for example at Fullerton and in portions of the Midway district. The only efficient method of drilling wells through 3,000 or 4,000 ft. of these mixed strata is the combination system employing both cable and rotary on the same rig. A heavy combination equipment, with a 24 by 112-ft. derrick, iron crown block, latest type of draw works and large rotary, frequently costs as much as a completed well located in the comparatively shallow portions of the Mid-continent or Eastern fields.

### 2. *Thickness of the Oil-bearing Stratum or Strata*

The thickness of the unconsolidated pay sand that must be penetrated is an important factor in the selection of the drilling system. One hundred feet of loose oil sand carrying even a moderate gas pressure often necessitates the use of rotary equipment notwithstanding the fact that a decision based solely upon the physical character of the overburden would select cable tools for the work. For example, in the extreme northern portion of the Midway field in the San Joaquin Valley, California, there is 700 ft. of predominantly hard cable-tool overburden which must be penetrated in order to reach an unconsolidated oil sand 280 to 300 ft. thick. Experience has shown that the rotary drilling method completes a well in that district in half the time and at less cost than cable tools. This for the reason that the former system averages more hole per day from top to bottom than the latter although more time is required by the rotary to reach the sand. The cable tools have no difficulty "making hole" through the 700 ft. of predominantly hard

overburden but encounter the greatest difficulty in making headway in the oil sand. In fact, the heaving nature of these sands causes innumerable fishing jobs and frozen strings of casing when an attempt is made to overcome them with cable equipment. After fighting them for weeks, if the standard driller has 200 ft. of pay to his credit he is doing well, whereas the same driller on a rotary rig can wash through the 300 ft. of sand in one tour of 12 hr.

These conditions indicate the important bearing the thickness of the unconsolidated pay sand has upon the problem of selecting a drilling system.

A few miles south of the area just described the oil sand is only 10 ft. thick and underlying similar formations of hard strata. Here the cable tools are far superior to the rotary because they can readily overcome the thin bed of pay sand and are better adapted to drilling through the overburden.

Therefore, it is seen that the mere thickness of unconsolidated oil sand may be the controlling factor in classifying the physical nature of the territory to be drilled.

### 3. *Thickness and Character of Water or Gas Sands*

Cable tools are superior to the rotary for prospecting new territory and for determining the exact depth of water- or gas-bearing horizons. It sometimes happens that an area or pool has been inefficiently developed and exploited; that available well records are unreliable, or that the lenticular nature of the sands causes great uncertainty regarding the exact depth of water strata. Under such conditions it is advisable to employ cable tools until all necessary data have been obtained, when the drilling method may be advantageously changed to the rotary. Conditions such as these at one time existed in portions of the Burkburnett field in northern Texas and although it was known that the territory was particularly adapted to rotary drilling it was necessary to "feel out" from proven areas with cable tools on account of the "spotted" nature of the oil and water sands.

### 4. *Gas Pressure to be Encountered or Expected*

Gas pressure is a factor of the utmost importance in the selection of a drilling system. Other things being equal, the greater the pressure the greater its importance; not only from the mechanical standpoint but to serve the ends of conservation. This pressure may occur above the oil reservoir, within the reservoir, or both. When it is known to occur above the oil sands it may be cased off with cable tools and conserved between strings of casing by the use of a packing spider, a special

device which has met with great success in the deep and high-pressured territory of southern California.

Where high gas pressures exist in formations difficult for a rotary to overcome it frequently becomes necessary to utilize special methods in conjunction with the cable system to properly cope with the situation. For example, the mud-laden fluid method and circulator systems may be called to the aid of the cable tools in passing through strata under high gas pressure.

The rotary system, however, is the ideal method of overcoming gas sands and should be employed wherever it is possible to drill by this method. In portions of the north Midway field the pressure within the unconsolidated and very loose oil pay is sufficient to heave the sands 200 to 300 ft. up in the casing of cable-tool wells unless special safeguards are introduced, whereas the rotary with its column of mud slip penetrates these sands with the greatest ease and dispatch.

### 5. *Total Depth of Well*

This is naturally an important factor in the selection of a drilling system, whether the oil sands to be developed are unconsolidated and loose or hard and compact. No attempt will be made in this paper to describe the many different types of rigs and methods of drilling. As has already been pointed out, the character and thickness of the oil sand often modifies the selection of any particular drilling method but the same sand at 1,000 ft. requires lighter equipment and possibly a different system than if it occurred at 4,000 ft. The former may be easily reached with cable tools and a 20 by 84-ft. standard rig while the latter may require the heaviest rotary or combination equipment with a 24 by 112-ft. or even 120-ft. rig, 7½-in. Ideal rig irons, calf wheel with sprocket drive, etc.

## Wildcat Territory

In the foregoing paragraphs it was assumed that developments of unconsolidated oil sands was to be undertaken in proven territory–that is, in a region already known and where accurate well logs were available for study and correlation. This is a very frequently not the case. Oftentimes little or nothing is known concerning underlying strata and unless geological evidence enables the construction of a fairly accurate columnar section of the region the selection of a drilling system is practically limited to cable tools. As has already been stated, this is the only method of pracedure in unknown or wildcat districts because the cable system is peculiary adapted to prospecting and the accurate determination of oil-, water- and gas-bearing strata.

Johnson and Huntley[1] make a good comparison of the advantages and disadvantages of the cable and rotary systems which indicates this superiority of the former for work in wildcat districts.

*Comparison of Drilling Systems*

*Cable System.—Advantages.*—

1. Less first cost of tools and rig.
2. Lower labor cost per day.
3. Less water necessary.
4. Can drill in the hardest rock.
5. More drillers available in some fields, although this is becoming less true.
6. Gives more information as to the formations passed through, and is thus better for prospecting.
7. Less cost per foot for relatively shallow wells.

*Disadvantages.*—

1. Longer drilling time.
2. Much slower when under-reaming is necessary.
3. Danger of delays and fishing troubles in soft strata.
4. When many water sands, hard to carry large hole to deep pay.
5. Greater cost per foot for moderately deep wells.
6. More casing necessary to handle caves and water sands.
7. Liability of getting crooked hole in soft formations.
8. Harder to control heavy pressures and more likelihood of "blowouts."

*Rotary System.—Advantages.*—

Faster drilling in soft strata.

2. Less trouble from caving and water sands.
3. Less casing used in soft formations with water and gas sands.
4. Straighter hole in deep drilling in soft formations.
5. Can handle alternate hard and soft formations, with less danger of accidents than with cable tools. This is made possible by the new bits and heavier rotary machines.
6. Can carry a large hole deeper.
7. When "drilling in," easier to control high gas pressure and prevent blowouts.

*Disadvantages.*—

1. Very slow in hard strata.
2. Greater daily labor cost.

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[1] R. H. Johnson and L. G. Huntley: *Principles of Oil and Gas Production*, pp. 118–119. New York, Wiley (1916).

3. Limited trained labor supply in some fields.
4. Greater cost per foot for shallow wells.
5. Does not show up smaller oil and gas pays, and important reservoirs may be passed through in prospecting.
6. More water necessary, a drawback in arid regions.

*Cable System Generally Used in Wildcatting*

Good practice demands that cable tools be used on any test well in an unknown or wildcat district. If the approximate depth of the stratum which is thought to be oil-bearing is known, or if the depth is decided by contract with the landowners, a standard or calf-wheel rig is erected of a size and type commensurate with that depth and drilling commenced with cable tools. If the territory proves to be suitable for cable-tool work and stands up, the well progresses satisfactorily and rapidly as a rule but if caving and unconsolidated sands are encountered, complications surely follow. These unforeseen contingencies will be treated briefly.

A test well passes through 1,500 ft. of hard formations and enters a stratum of unconsolidated oil-bearing sand under pressure. The 10-in. casing is landed at 1,450 ft. as a water string to shut off several water-bearing strata encountered at higher levels. The 8¼-in. casing put in after landing the 10-in. becomes "logy" and finally "freezes" when 40 ft. of the sand has been drilled.

The foregoing case is typical and one taken from actual practice. It is essential to keep the casing "free" when carrying hole through unconsolidated sands, whether pay sands or not. Sometimes 10 ft. of sand will freeze casing and in other cases with careful management in experienced hands 150 to 250 ft. of heaving oil sand may be overcome with cable tools.

In the example just mentioned, the 8¼-in. casing is frozen. The first thing to be done is to attempt freeing it. Alternate pulling and driving may accomplish this. The power of a calf wheel applied through seven lines to the heavy casing block is tremendous and an expert driller knows just the amount of stretch the casing will withstand before parting. If it comes an inch or two the drive clamps are adjusted on the square of the stem and the casing driven back to its former position, when it is again pulled—possibly yielding an additional inch. The principal thing is to cause it to move even slightly, as it is sure ultimately to become free by this method. It is well to apply the casing tongs after each pull or two and set up the entire string a trifle. This tightens any threads that may have become loose.

It frequently happens that the alternate pulling and driving method proves futile. The casing refuses to move an inch. In this event other methods are at the command of the engineer. One that is often success-

ful and is not commonly known is to relieve the static pressure within the casing, it being presupposed that the hole is full or nearly full of fluid, and allow the sand to heave within the drilling string. It is seldom necessary to bail more than 300 or 400 ft. of fluid from the well in order to so disturb the balance of pressures that a sudden upheaval of the oil sand takes place. Following this upheaval the driller tries the casing and if it is free, which is nearly always the case, the hole is cleaned out to bottom and drilling resumed.

It sometimes happens on account of insufficient pressure in the sand, or from other causes, that the sand fails to heave when the static pressure is reduced. In that event a casing spear, preferably of the trip type, is run to bottom on the tools and engaged near the shoe, with long stroke jars and sinker bar or stem giving necessary impact to the jars. A strong pull is then taken with hydraulic jacks applied to the casing by means of a spider, while jarring is commenced on the tools. This method will free exceedingly tight casing. A nail may often be driven out when it cannot be pulled. The jarring of casing is exactly similar.

If this method fails to free the casing, a spear may be run on a string of the next smaller size casing and a hold secured near the bottom of the frozen pipe. A heavy pull is then taken on both strings of casing while jarring is commenced with the tools which carry long stroke jars and a casing spear as described in the previous method. In fact, the two methods differ only in that a much stronger pull can be taken on the two strings of casing than on the one; in other words, the pull is doubled.

In rare cases it happens that the unconsolidated sand holds the casing in such a grip that even this tremendous pulling force and the heavy jarring, which may be likened to the blows of a steam hammer, fail to free it. As a last resort the casing may be split in several places to allow the binding sand to run into the well. To accomplish this a casing ripper is run in on the tools and several long gashes cut in the casing at a point where it is thought the maximum "friction" exists. If this results in the entry of sand it will nearly always free the string.

In the event that all these attempts fail and it is impossible to free the casing, two methods of overcoming the difficulty remain. A four-wheel casing cutter may be run in on 3-in tubing and the casing cut at a point just above where it is frozen. The upper portion of the string is then removed from the well, a new shoe adjusted and the joints imbedded in the sand "side-tracked" or drilled past. This is slow and difficult work and not always successful. Some operators shoot the casing instead of cutting it. A small charge of dynamite is exploded at the point where the casing is frozen, but this results in jagged ends at the place of rupture, making the casing more difficult to side-track. It is a quick and inexpensive method but cannot be considered good practice in many cases.

If it is not desired to cut or shoot the frozen string, the only thing to be done is to abandon it and case with the next smaller size pipe. This is permissible providing the frozen string is of such a size, say 8¼ or 6⅝ in., that the well may be completed to the required depth before "pointing out."

Loose oil sand under pressure often heaves in a cable-tool drilling well when the bit drills through the cover rock, and unless the driller is warned in time the tools are buried for many feet by the sand. In certain of the California fields the oil sand has often buried a string of tools 300 ft. or more and frequently led to a bad fishing job. Whenever such sands are anticipated the experienced driller "carries" the hole full of water or thin mud slip in order to counterbalance the rock pressure. The jars and sinker bar are also used on the tools so that in case of a sand heave they may be jarred free. It is sometimes necessary to jar the tools through 100 ft. or more of oil sand, which requires possibly 12 to 24 hr. If the wire drilling line parts a bad fishing job is apt to result and it may be necessary to pull the casing (if it is free) in order to recover the tools. Or if it is frozen a "fishing string" is run in, after cleaning out to the top of the lost tools, and an attempt made to recover them with a socket or other special fishing tool. This is simply an example of one of the many fishing jobs of endless variety that may occur from the heaving of unconsolidated oil sand. These same remarks apply with equal force to conditions arising from the penetration of any unconsolidated sand or other material of a cavey nature which may freeze casing or bury cable tools.

The foregoing problems connected with the drilling of wells with cable tools in unconsolidated sands are, of course, not met with when the rotary system is employed, but it must be remembered that wildcat territory, which demands the use of cable tools, is being considered. The difficulties encountered were unforeseen and unexpected. After the cable tools have proved the district, future exploitation may be carried on by the rotary system.

### *Rules Governing the Drilling of Deep Test Wells*

A few general rules and cautions governing the drilling of deep test wells in wildcat territory where unconsolidated sands are expected, follow:

1. Allow a wide margin above the calculated cost of the well.
2. Commence drilling with a sufficiently large hole—20 to 22 in. for a deep test.
3. Do not expect a light rig for shallow territory to drill a deep test.
4. In remote districts have all the casing on the ground and an adequate supply of small tools, pipe and fittings.
5. Have a rope grab, combination socket and other frequently needed fishing tools available for instant use.

6. After selecting the site for the rig, have a cellar dug about 8 by 10 ft. by at least 15 or 20 ft. deep. The depth is important.

7. Provide an adequate supply of the best obtainable water for steaming purposes.

8. Anticipate heavy oil or gas pressure and have a control casing head on hand if occasion arises for its use to cap the well.

9. Locate a deposit of fine clay for use in mixing mud slip to shut off pressures or to use while drilling in unconsolidated sands that refuse to bail readily.

10. If the well is to be drilled by a contractor at so much per foot, do not expect sands tested, water shut off, and logs kept as carefully and efficiently as would otherwise be done. Many important features of the work are slighted and the time required for their proper accomplishment sacrificed in the effort to make the maximum amount of hole in the minimum time.

11. Carry every string of casing to the maximum depth possible and never abandon the effort until every means at the command of the operator has been exhausted in the attempt.

12. The management of the work, selection of methods, etc., should not be left to the drillers but placed in the hands of a competent superintendent.

13. Keep a most careful and accurate log of the well, preferably in several copies in order that if one is lost the record may be preserved.

Taking up some of these points in more detail: In allowing for the cost of a test well in wildcat territory a safe rule to follow is to figure the actual cost as accurately as possible and multiply this figure by two. How many wells are reported as "abandoned for lack of funds?" It is safe to say that unconsolidated sands or other cavey materials are responsible for a large percentage of the failures and for most of the difficulties, discouragements and delays in the drilling of test wells. Many wells that "ought not to cost $15,000" actually require the expenditure of $25,000 or $30,000 for their completion. Much larger amounts are not at all unusual and it is well to keep in mind that figuring the cost of a test well in a wildcat district is a good deal like provisioning an exploring party to penetrate unknown wilds—emergencies must be anticipated in both cases.

The importance of commencing the well or "spudding in" with a sufficiently large hole cannot be over-estimated. In order to reach the desired depth, whether 2,500 or 4,000 ft., it is imperative to make the size of the hole at the surface commensurate with the depth. In the soft Tertiary formations of California and in the Baku fields of Russia it is found advantageous to use large-diameter stovepipe casing for the first few hundred feet, especially where beds of boulders are encountered. This type of casing is made by riveting sheet iron or steel of 14 to 10 U. S.

gage and may be purchased in lengths of 2 ft. or multiples of 2 up to 20 ft., the shorter lengths being used when progress with the string becomes slow. In the Russian fields, wells are commenced with holes of extremely large diameter, not on account of any extraordinary depth but in order to complete a well in the pay sand of sufficient diameter to permit the use of large-sized bailers, the method used in that country to bring the oil to the surface, because the excessive amount of unconsolidated sand precludes the use of pumps. The usual sizes of the California stovepipe casing are 16 in., 18 in. and 20 in. and if 500 to 700 ft. of the 18-in. size can be put in a well the operator has a good beginning for a deep test in unconsolidated formations. A great advantage possessed by this type of casing is the absence of collars with a consequent reduction in friction, making it especially suited for driving through strata of unconsolidated sands. Its use will probably extend to Oklahoma and the Mid-continent fields as development extends westward into deeper territory possessing all the problems arising from the presence of unconsolidated sands or other strata of loose materials. Many test wells have been and are being drilled in the Permian Red Beds of western Oklahoma and the occurrence of deep deposits of unconsolidated materials in this region is gradually bringing about improved drilling methods and the introduction of more efficient mechanical devices. For the same reason, operators accustomed to the hard strata of the Eastern fields where casing problems are of small importance are realizing the necessity of large-diameter holes and the use of heavy-weight pipe with improved methods of handling it. Commencing a test well with too small a diameter means that it "points out" before it "tests" anything. In Pottawatomie County, Oklahoma, a typical wildcat section, are many abandoned "test" wells that pointed out before a depth of 2,500 ft. was reached, although it was known that if oil occurs it would be found at 3,000 ft. or more. These failures were all due to the fact that the holes were commenced with diameters too small and that the presence of unconsolidated sands was not taken into account. These sands necessitated the use of a greater number of strings of casing which in turn caused the well to point out sooner than would have been the case had it been commenced with a hole of larger diameter. "It may be said in conclusion that in drilling in wildcat territory where the number of water and gas sands is unknown and the depth of the oil sand uncertain, and therefore the ultimate depth of the hole cannot be known, one must start with a larger size of casing than that which will probably be used for later wells when the field is developed. Pioneer wells have been drilled in some pools, which failed to discover oil because the hole was so small it could not be carried deep enough to penetrate the sand which was later discovered to be the main oil pay, perhaps only a few feet beyond where the early well stopped drilling. Wells in the Calgary dis-

trict in Alberta are started with 18-in. casing. In California even larger casing has been used."[2]

Probably the best type of rig for handling heavy strings of casing in unconsolidated sands is the California or calf-wheel with sprocket drive and extra heavy rig irons. A light standard rig such as is commonly used in the Eastern fields and parts of Oklahoma is sufficiently strong to handle light-weight strings of casing through consolidated formations but is totally inadequate for the heavy pulls required to free long strings of California pipe in contact with several hundred feet of unconsolidated sands. For the heaviest work, iron or oak-calf and bull-wheel shafts are required, and concrete foundations for the derrick. The usual practice of placing a few pieces of waste lumber under each derrick leg as foundations should not be countenanced, especially where the casing may have to be carried through cavey material with the consequent heavy strains which inevitably result in pulling the derrick out of plumb.

A feature connected with cable-tool drilling in unconsolidated sands which many operators, accustomed to hard formations, overlook, is the importance and advantage of an adequate cellar. When a string of casing is being carried through sands or other cavey material it has to be "worked" almost continually and sometimes freezes while one "screw" is being run. It is difficult or impossible to make any hole ahead of the shoe and frequently the casing is allowed to "follow;" that is, it is released from the spider slips and follows the drilling bit of its own weight. If it refuses to follow it is worked up and down with the casing block and calf wheel and sometimes spudded with a jerk-line or driven with the drive clamps and tools. In all of these operations a deep cellar is of the greatest advantage because it allows the driller to work the casing below the derrick floor until there is sufficient clearance for another joint to be screwed on. Casing averages about 20 ft. to the joint, hence a cellar of approximately that depth is most desirable, although 15 or 18 ft. is a great improvement on the inadequate and shallow pits commonly placed under derricks in regions where the problem of unconsolidated sands is unknown or under-estimated.

## Finishing a Well in Unconsolidated Sands

The process of finishing a well in an unconsolidated pay sand is quite different from the "bringing in" of a well tapping a hard sand. It sometimes requires weeks to penetrate a thick stratum of loose pay sand with cable tools, whereas the "drilling in" and "shooting" of a hard sand is a much simpler process. Where the sand, whether loose or hard, is under great pressure and the well "comes in" as a gusher or flowing well

[2] R. H. Johnson and L. G. Huntley: *Principles of Oil and Gas Production*, p. 125. New York, Wiley (1916).

as soon as the sand is tapped, the results are exactly similar except that the well in the unconsolidated pay is apt to throw out great quantities of sand.

The ideal method of finishing a well through a thick stratum of loose oil sand is the rotary system. As mentioned in the first part of this paper, it is usual for a rotary to penetrate 300 ft. of unconsolidated oil pay in 12 hr. or less, and by thickening the circulating mud the pressures may be balanced to such a nicety that upon reaching the stratum underlying the sand the entire string of drill pipe may be withdrawn without the slightest danger of caving. In other words, a column of mud-laden fluid extends through the sand and is substituted in the physical balance of forces for the loose sand which previously occupied the same space. Although all the purposes of an open hole have been served, nature's balance is left undisturbed. This mud may be allowed to stand for several hours but the best practice is to prepare the "liner" or perforated casing in advance and have everything ready for "casing" immediately upon withdrawal of the drill pipe.

The liner may consist either of "shop-perforated" casing or one of the several types of screen pipe which have come into extensive use because of their greater efficiency in handling sand with a consequent increase in the amount of oil recovered.

It will be assumed that screen pipe is to be set and that the water string consists of 8¼-in casing landed in a hard stratum a few feet above the oil sand. Three hundred feet of 6⅝-in. screen pipe is carefully tallied and two blank joints of the same size made ready. A collar is screwed on the bottom of the last joint of screen pipe and a back-pressure valve and wooden wash plug placed therein. The 300 ft. of screen pipe with the two blank joints on top is then run into the hole to bottom on 3-in. tubing, this tubing resting on the wooden wash plug and being attached to the 6⅝-in. casing with a right- and left-thread nipple. The principal thing is to have the tubing extend through the screen pipe in order that the water may pass out of the bottom of the liner and not through the meshes of the screen. When the liner has reached bottom it is raised an inch or two and the well thoroughly washed by pumping clear water through the tubing or wash pipe for 24 hr., or until the "returns" cease to show any trace of mud. The wash pipe is then backed off by turning to the right and withdrawn. This leaves the liner clean, with no mud between the meshes and the oil sand, and the two joints of blank 6⅝ in. extending up into the 8¼ in. forming a 40-ft. lap. An adapter from 6⅝ in. to 8¼ in. is then lowered on the bailer or dropped through the water in the casing until it rests on top of the liner, making a smooth connection between it and the 8¼ in. The well is then tubed and when the water pressure is relieved begins to produce oil.

The proper setting of screen pipe with a rotary requires experience

and knowledge, especially as quick work is desirable and any accident may prove costly. Instead of using tubing, a heavy liner, whether of screen pipe or perforated casing, may be set with the rotary drill pipe, as the same method is used with both.

The finishing of a well with cable tools in a very thick stratum of unconsolidated oil pay is much more of a problem than that just described where a rotary is used. The ordinary procedure, especially in districts where screen pipe has never been tried or is unknown, is to carry the casing, of the size the hole is to be finished with, through the oil sand and then perforate it with a perforating machine. It frequently happens that the string freezes after penetrating the sand for 100 ft. or less, in which event it is usual to place an iron or wooden heaving plug in the bottom of the casing and then perforate. If the operator is fortunate enough to carry the casing through the pay it can be landed in the stratum underlying the sand, which obviates the use of a heaving plug, the only function of which is to prevent the sand heaving inside the casing and overwhelming the pump. Irrespective of the depth of sand attained, the use of the perforator is necessary with this method and is its greatest drawback, mainly on account of the uncertainty of the results therefrom. For mechanical reasons, variation in the thickness or toughness of the casing, and other factors beyond the control of the operator, the use of a perforating machine is frequently disappointing. If the well comes in as a small producer there is always the suspicion that the perforator failed to punch a sufficient number of holes, whereas if the casing "sands up" and the well requires constant "pulling" the reverse is true. Arnold and Garfias[3] mention these points in describing methods of oil recovery in California:

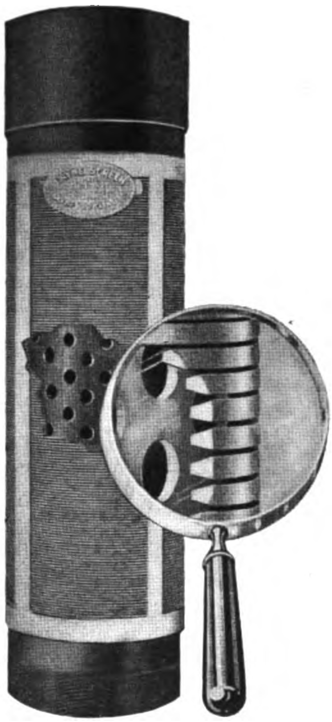
"The indiscriminate use of perforating machines has been the source of much trouble. In the Coalinga field one well about 1,300 ft. deep was perforated three times, and when the casing was removed it was found that, except for a few holes, the machine had only indented the casing. In other wells, owing to local brittleness of the casing, the perforator has removed large pieces of it; and in some deep wells, owing to the great weight of the column of tubing to which the perforator is attached, the perforator has cut into or strained the collars so as to cause collapse of the casing."

In the light of these facts and of past experience in drilling many wells in unconsolidated oil sands, it may be said *that every effort should be made to avoid bringing in the well by perforating the casing after it has been placed in the hole.* As a substitute for such perforating the operator may use screen pipe or shop-perforated casing. In finishing a cable-tool well with either of these two devices the problems involved are largely or entirely dependent upon the thickness of the unconsolidated sand. If this sand is a thin bed 50 ft. or less in thickness there is usually no difficulty in

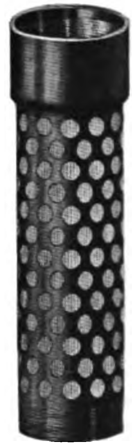
[3] R. Arnold and V. R. Garfias: Methods of Oil Recovery in California, *U. S. Bureau of Mines, Technical Paper* 70, pp. 9–10 (1914).

setting any type of screen or of shop-perforated casing, but where the sand is 250 to 300 ft. thick many obstacles may be encountered.

The ideal method of finishing a cable-tool well in a very thick stratum of loose pay sand is to carry, say, 8¼-in. casing *through* the pay and then set a 6⅝-in. liner, either screen or shop-perforated, preferably the former, inside this 8¼ in. and then remove the 8¼-in. casing, leaving the liner in contact with the oil sands. This is very difficult to accomplish, however, because it necessitates keeping the 8¼ in. free at all



After Layne & Bowler Corp.
FIG. 1.—SECTION OF WIRE-WOUND SCREEN CASING.



After McEvoy Wireless Well Strainer Co
FIG. 2.—SECTION OF WIRELESS SCREEN CASING.

times, and this is seldom possible. In fact, it may freeze beyond all power of freeing while the liner is being set.

It well may be asked, why not carry 8¼-in. screen or shop-perforated pipe through the sand in the first place and thus save the extra labor of setting a liner and gain the advantage of completing a well of larger diameter. The answer is that it is much more difficult to carry perforated than blank casing through unconsolidated sands and that all types of screen pipe possess the same objection. In addition, certain screens are not made to withstand such work and are easily damaged, necessitating the use of a conductor casing when setting.

Screen casing may be divided into two general classes: (a) the screening device on the outside of the casing, see Fig. 1; and (b) the screen forming a part of the casing wall, see Fig. 2. In the first type the screen generally consists of wire wound around perforated casing, while in the second type the screen is in the form of numerous buttons or slotted brass plugs carried in the walls of the casing and flush with the outside surface, or in slits cut directly in the casing wall itself. This latter type, sometimes known as "wireless" screen, has the advantage of withstanding hard usage, and many operators have carried it for considerable distances through unconsolidated sands. But in very thick strata of loose pay it is as difficult to carry as shop-perforated casing and entirely impossible where blank pipe encounters difficulties. The wire-wound screen casings are entirely unadapted for carrying through thick strata of unconsolidated sands and the attempts to finish a well in that way should never be made. The wire screen, however, possesses many advantages over the wireless, as will appear in later paragraphs, but in finishing a well with this type of screen it is necessary to set it either inside a conductor string or by means of a rotary, as otherwise the wires may become detached from the casing and cause endless complications and trouble.

In concluding the subject of the drilling and finishing of wells in unconsolidated sands, it is desired to emphasize the fact that most of the problems connected therewith arise from two general causes: (a) the development of unknown districts; and (b) the existence of such a combination of physical conditions in the strata that it is impossible to employ one system capable of overcoming them. For example, if a thick body of unconsolidated oil sand has an overburden of 1,200 ft. of hard compact formations the cable system demanded by the latter is confronted, upon entering the oil sand, by many of the problems already described. These conditions would not justify the installation of an expensive combination system and could best be overcome only by the exercise of good judgment and foresight backed with skill and experience. Even though possessing these qualifications in high degree, the seeker for oil must recognize the element of "luck" in many of his operations and be duly appreciative when it is in his favor and stoically patient when it is otherwise.

## III. EXTRACTING THE OIL AND SAND

The process of obtaining oil from the earth when that oil is contained in a loose unconsolidated sand is very different from the method employed to extract the recoverable portion of the oil content of a hard stratum. *With an unconsolidated sand the extracting of its oil content may take place above ground whereas with a hard oil-bearing stratum this is impossible.* This distinction has an important bearing upon all questions relating to

oil production. It is the factor deserving the greatest consideration in determining the efficiency of methods employed for the maximum recovery of oil from unconsolidated sands and is the basic principle of the statement that: *Other things being equal, the maximum recovery of oil from an unconsolidated sand is directly dependent upon the maximum recovery of the sand itself.*

The truth of this law is based upon two important considerations: (1) The removal of sand by a well causes a larger volume of sand, and therefore of oil, to move toward that well than would be the case if oil only were removed. (2) The efficient separation of sand and oil, and therefore a nearer approach to obtaining the recoverable oil content, is made when this separation takes place above ground.

## The Movement of Sand and Oil Toward a Well

The higher the viscosity and the lower the gas pressure within the oil reservoir the greater becomes the importance of creating and maintaining a movement of sand toward a producing well. This has been proved beyond all question in extensive operations in the north Midway district of California where the oil is of 14.5° Bé. gravity with a very low gas pressure.

"In the Balakhany oil district, where the oil is heavy, and the gas has nearly all escaped from innumerable wells, the concentration of the petroleum toward old producing areas is very marked, and whilst old bailing properties, which have for many years yielded a moderate payable production show little or no fall-off, new lands interspersed between these are almost valueless and give practically no output when new wells are sunk. This is additional testimony of the movement of oil towards spots where immense quantities of sand have been removed by bailing, creating areas of low density into which the petroleum, no longer assisted by a gas pressure, percolates by gravitation. In one case, a new well bored in the Balakhany district was a distinct failure, for stratum after stratum failed to yield even a mean production, but an old well which had been bailed for many years before abandonment, gave, when cleaned out, the extraordinary production of 2,500 poods a day for more than a month, and then only fell off by degrees to 400 poods daily."[4]

The fact that large producers come in without "making sand" evidences the existence of high gas pressure and insufficient means of entry for the sand to pass into the casing, but it is not proof that the large production is due to the absence of sand or that the output would not be increased if sand were expelled with the oil.

With proper mechanical arrangements a well tapping a thick stratum of unconsolidated pay produces a tremendous amount of sand. That it continues to do this for long periods is proof that a constant replenishment takes place near the casing, necessitating a movement of sand and

[4] A. B. Thompson: *The Oil Fields of Russia*, p. 51. London, Crosby Lockwood and Son (1908).

oil toward the well. This movement is frequently of such volume, or occurs so suddenly, that the casing penetrating the reservoir is damaged or completely severed from the upper and more rigid portion of the string. Several instances of such damage may be mentioned: In one case 250 ft. of 36-lb. 8¼-in. casing was twisted somewhat in the shape of the letter S and was removed from the well only with the greatest difficulty. In another instance several hundred feet of 6¼-in., 20-lb. casing was broken just below a collar and displaced to such an extent that no part of it was touched by the tools while repairing the damage. Several cases may be cited where casing has been so deflected or bent by sand movements that it was impossible to tube the well beyond the point of flexure. No type of casing is proof against this force of moving sand, and both screen and perforated pipe suffer, although it is believed the former less so than the latter. Likewise it appears to make but little difference whether the casing passes entirely through the sand and is anchored in an underlying stratum or penetrates for only half the distance.

## Methods of Extraction

In order to bring about a movement of sand with its oil content toward the well, it is necessary to provide means for its efficient removal with special casing devised to facilitate this process. If the sand cannot enter the casing, not only is the movement of oil retarded but the sand surrounding the casing is partially drained of its oil content, resulting in a constantly increasing body of "dead" sand opposed to the richer portions of the reservoir. Instances are known where a well, because of faulty perforations or inadequate screen openings, produced small quantities of clean oil free from sand for several years and when an attempt was made to substitute screen with proper-size mesh the sand had become so "dead" in the vicinity of the well that it lacked the "life" to enter the casing and continued to obstruct the path of better "pay." The same conditions apparently explain similar failures when a well was reperforated, but in such a case there is always the element of doubt that arises whenever a perforating machine is used. These cases occurred with a very loose oil sand containing an asphaltic crude of 14.5° Bé. gravity and were undoubtedly influenced by nearby wells having established strong drainage channels. In any event, the fact remains that they produced no sand and very little oil, and that these conditions could not be improved. Furthermore, there was no local variation in the reservoir or other controlling factors to account for this phenomenon.

On the other hand, when a well is properly equipped and produces great quantities of sand with the oil a large area of lower density is gradually established around the casing and, unless encroaching water is present in quantity, this removal of sand must inevitably lead to the for-

mation of a cavity. This cavity probably assumes the form of an inverted cone as sand continues to be removed while new sand with its oil flows by gravity down the slopes of this subterranean funnel and supplies the well with fresh material. The shape of the cavity is undoubtedly influenced by the dip of the strata, when marked, and by the disturbing effect of nearby wells. That a cavity forms, however, cannot be disputed. Sand slips, damaged casing, and collapse of the cover rock evidence such a cavity in addition to the impossibility of otherwise explaining the effect of removing the tremendous volume of sand that wells have been known to produce.

It would be interesting to know when the slopes of these cavities reach an angle of repose and experiments with saturated oil sand on the surface should prove most instructive. The fact that wells producing from unconsolidated pay sands gradually diminish in their output of sand may indicate the approaching stability of these underground slopes, and evidence presented in later paragraphs strengthens such belief.

Having attempted to show the necessity, not only of removing sand in order to recover the maximum quantity of oil, but the favorable results of such removal in cavity formation with its accompanying low pressures and "live slopes," the mechanical handling of this sand will now be described.

## Function of Screen Casing

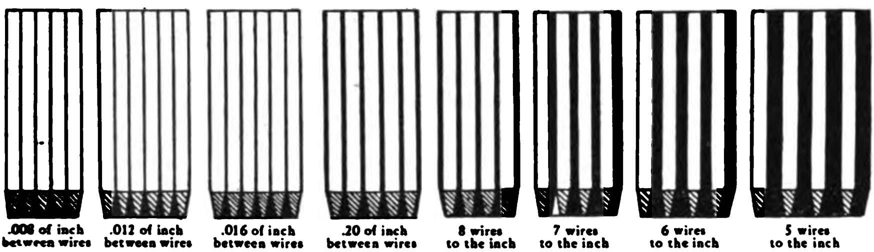
Screen casing of proper mesh serves its best usefulness in permitting the *regulated entry of the maximum amount of sand and oil* while excluding gravel and small stones which would "sand up" the pump by sticking the valves, or otherwise prevent its proper action. It probably serves another use in that it results in a more even distribution of the oil and sand channels radiating from the well than is the case with ordinary perforations. This even drainage of sand and oil equalizes the pressure in all directions around the casing, whereas the use of a perforating machine may result in the sand being drained from one side more than the other, thus creating a pressure against the casing. This unequal drainage due to faulty perforation is an argument for the use of screen pipe and a possible explanation of deflection and damage to oil strings.

A section of wire-wound screen casing is shown in Fig. 1. This type of screen utilizes "keystone wires" whose function is to prevent clogging of the mesh—the space between the wires increasing radially and thus permitting any substance that enters to pass through unhindered. An exceptionally large screening surface is exposed to the sand with this type in that the wires between the perforations are utilized as fully as those directly over the openings.

Fig. 2 shows a section of wireless or button screen which is especially

adapted for work with cable tools, as it will withstand hard usage and the driving and pulling often necessary with that system in drilling through unconsolidated sands.

The selection of the proper-sized mesh depends upon the physical character of the sand, viscosity of the oil and the gas pressure. The mesh should be of sufficient size to allow the maximum entry of sand and oil while excluding pebbles and small stones. This is best determined by practical experiment above ground. Other things being equal, a high-viscosity oil demands a coarser mesh than one of low viscosity. A screen made up of five wires to the inch (see Fig. 3) proved most efficient with a moderately fine sand carrying oil of 14° to 15° Bé. gravity. The erroneous belief that the function of screen casing is to prevent the entry of sand in oil wells, with the consequent avoidance of pumping troubles,



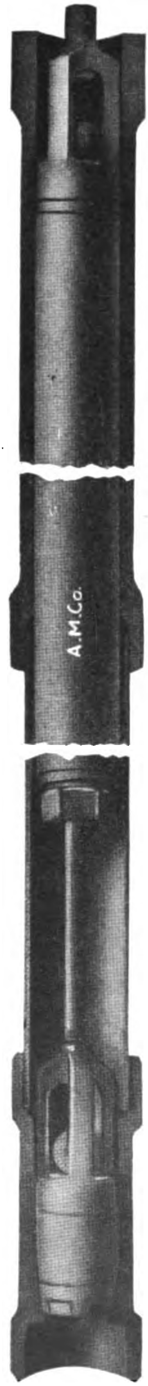

After Layne & Bowler Corp.

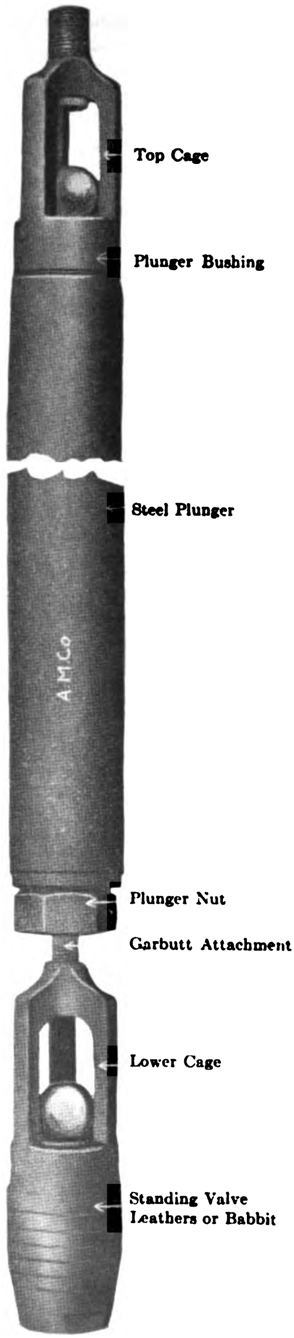
FIG. 3.—SHOWING DIFFERENT MESHES OF A WIRE-WOUND SCREEN.

leads to the selection of too fine a mesh in many instances. *The mesh should be too coarse rather than too fine.* Screen pipe came into use and was devised in connection with water wells and it was natural that the same rules were thought applicable when it was introduced in the oil fields.

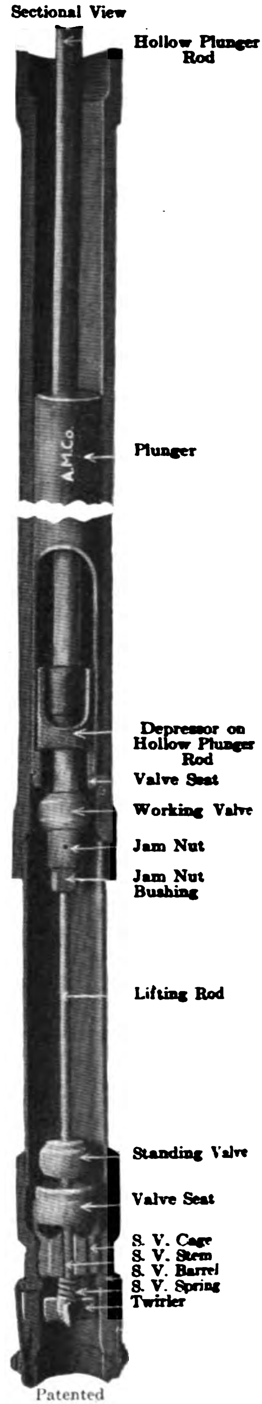
## PUMPING THE SAND AND OIL

Assuming that a well has been properly drilled, thoroughly washed, screen casing set, and that sand and oil are passing freely through the mesh, the most efficient device to bring the sand and oil to the surface is the plunger pump (Fig. 4). Although many wells flow naturally for a time or may be agitated by periodically moving an "agitator string," and others containing much water may be pumped with an air lift, the standard equipment for handling sand is the plunger pump. This device consists essentially of an outside cylinder or "working barrel" and a hollow plunger with upper and lower valves (Fig. 5). Although simple in principle, these pumps are very carefully made of the best material and ground to an exact fit, and their proper installation and operation require experience and knowledge in order to obtain the best results in








After Axelson Machine Co.

FIG. 4. FIG. 5. FIG. 6. FIG. 7.

For description, see foot of next page.

the removal of sand. Fig. 4 shows a pump completely assembled and ready to be attached to the tubing. The working barrels are made in lengths from 42 to 60 in. and from 2 to 4 in. diameter. A common size is 3 by 60 in. for the "renewable" or liner pump with extension and top collars at both ends (Fig. 6). This is one of the best types of pumps for handling sand, as the upper portion of the plunger does not leave the barrel and therefore is not in contact with sand on its downward stroke. The standing valve barrel is carried on a 3 by 24-in. nipple which allows the plunger to pass below the lower end of the barrel on each stroke, thus retarding the entry of sand between plunger and barrel.

The Parker pump shown in detail in Fig. 7 is said to be particularly efficient in handling sand on account of the large area of the valve openings and the positive action of the valves.

### *Pump Operation*

A properly installed and operated plunger pump will handle a surprising quantity of sand; in fact, it may be said that it is capable of pumping sand containing oil rather than oil containing sand. It is not at all unusual for one of these pumps to handle a mixture containing 50 per cent. of sand by volume. A careful record should be kept of the performance of the pump on each well: Depth at which it is pumping, date when new valve seats or balls were substituted for the old, when the barrel was renewed and on what date the tubing was lowered. The lasting qualities of a pump are directly dependent upon the quantity of sand produced and as this varies with each well the importance of keeping a record of pump performance is apparent. A barrel may require renewal every 6 weeks on one well while on another it shows but slight wear after 3 months' use.

The common practice of allowing a well to pump until the barrel becomes so worn that the production dwindles to a small stream or until some accident or breakage necessitates pulling should be condemned. Each pumping well should be studied and every effort made to maintain the production not only of oil but of sand.

The well should be tubed to such a depth that the maximum quantity of sand which the pump is capable of handling reaches the barrel at all times. This point can be determined only by experiment and gradual lowering of the tubing as sand production diminishes. Continued pumping of sand may save expense and labor of cleaning out "dead" sand with tools and bailer.

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Figs. 4 to 7.—Plunger Pump and Parts for Raising Sand and Oil.
Fig. 4.—Plunger and Valves in Working Barrel.
Fig. 5.—Plunger with Upper and Lower Valves.
Fig. 6.—Working Barrel with Liners and Extension Nipple to Standing Valve Barrel.
Fig. 7.—Parker Pump.

In order to determine the exact condition of the barrel and plunger it is essential that both be removed from the well, washed with distillate and then tested for fit and the absence of lateral play between the plunger and the barrel.

The common practice of removing only the rods and plunger for examination is to be condemned.

The pumping of sand is troublesome and means much work around the wells. For this reason there is a tendency to avoid it.

A long stroke is advisable for pumping wells producing sand and the wristpin should be carried in the "second hole."

If a valve becomes clogged with small gravel or sand it may often be freed by "shaking up the well" or running the pump at high speed for a few strokes. Twenty-five strokes to the minute is the usual pumping rate when the rods drop freely and prevents the sand from settling.

### *Hot Oiling*

#### Methods of Overcoming a Decline in Production

After a well has produced sand for a time varying from months to years it gradually diminishes in output even though the well is tubed to bottom. In such cases several methods are used to "liven up" the well and increase production. Steam has been used in such wells and "swabbing" is commonly practiced, but the best method is probably the introduction of hot crude oil. Where this method fails it is undoubtedly caused by faulty casing, either screen or perforated, or the hot oil is improperly applied.

The best practice is to heat about 100 bbl. of crude almost to the boiling point and allow it to flow by gravity or pump it down the well casing. It is usually treated in a nearby tank with steam coils and although some operators introduce it through the tubing this practice has the disadvantage that the necessary stoppage of the pump may cause it to sand up:

The effect of the hot oil is to wash the screen or perforations and to lower the viscosity of the underground supply, causing both the oil and sand to flow more readily toward the well.

The hot oil need not remain in the well for more than ½ to ¾ hr. and is then pumped out together with large quantities of sand and "live" oil, oftentimes accompanied by considerable gas. It frequently happens that so much sand enters the well following this treatment that the pump is unable to handle it, in which case two or three joints of tubing are removed, the barrel afterward being gradually lowered as the sand becomes exhausted. After a few weeks the pump again reaches bottom, when another hot oiling is given.

There is no fixed rule as to the frequency of this treatment. This

can be determined only by experiment and is largely dependent upon the age of the wells. In some cases it may be applied once a week with beneficial results while in others once a month is sufficient.

## IV. QUANTITY OF SAND PRODUCED

This is a most interesting subject but, unfortunately, exact figures on the actual quantity of sand produced by flowing or pumping wells are unavailable.

In speaking of the sand produced by wells in Russia, Mr. Thompson[5] says: "The oil from fountains is commonly accompanied by an equal bulk of sand, large numbers of stones, and the liberation of millions of cubic feet of gas which becomes disengaged from the oil on its exit from the tube. A recent Bibi-Eibat spouter on plat No. 29 gave as much as 10,000 tons of oil and 10,000 tons of sand in a day, and in a few weeks yielded, in addition to several million poods of oil, no less than 1,700,000 cu. ft. (85,000 tons) of sand; an amount which will be better appreciated when it is realized that this quantity of material would raise the natural level of the ground on 1 dessiatine nearly 14 ft., or cover an acre of land to a depth of 40 ft."

The famous gushers of California and other loose-sand fields have expelled tremendous quantities of sand which in some cases have completely buried the engine house, belt house, and the lower panels of the derrick.

"In the old Sunset field wells that have a strong gas pressure produce more sand than any others in the State, it being estimated that sometimes as much as two-thirds of the gross yield of the wells is sand. One well alone produced over 110,000 cu. ft. of sand in about 4 years, and another has yielded almost as much in 2 years. The yield of sand gradually decreases with the age of the well, but at no time entirely ceases. These peculiar conditions—soft sand in great quantities accompanying the flow, heavy oil, and strong gas pressure—make the problem of well operation in this field difficult."[6]

Some pumping wells in the north Midway field produce sand at the rate of over 200,000 cu. ft. a year and careful measurements would probably show that this amount is actually produced in a year or even exceeded.

## V. OIL SEPARATION AND SAND DISPOSAL

The separation of oil and sand, although more fully accomplished above ground, is one of the difficult problems connected with production from unconsolidated sands. With low-gravity, viscous oils and great

[5] A. B. Thompson: *The Oil Fields of Russia*, pp. 52–53. London, Crosby Lockwood and Son (1908).

[6] R. Arnold and V. R. Garfias: Methods of Oil Recovery in California, *U. S. Bureau of Mines, Technical Paper* 70, p. 22 (1914).

volumes of sand, nothing has been found to take the place of the open-air "sump," which is simply an excavation made near the well with teams and scrapers. The sand collects in great piles in these sumps while the oil separates by gravity and is pumped from the lowest point in the depression. With heavy oil the loss through evaporation and percolation is slight, as it has been determined that it does not penetrate the ground for more than a few inches. Light oils, however, should not be allowed to flow into sumps, as the loss is undoubtedly great; but when accompanied by much sand other disposition is difficult. As the piles of sand are gradually freed of their oil content they become fairly dry and a crust is formed on their surface which will bear the weight of a man.

Many tanks, settling boxes and other devices have been designed for the separation of oil and sand but with little success. Most of these devices are entirely lacking in scientific principle and are carelessly constructed affairs designed by the workers in the fields and their failure is in large means due to these facts. An efficient means of separating light oils and sand can undoubtedly be devised and would prove of great benefit to the producer. Heavy oils being much more difficult to separate, of less market value, and of high viscosity, may be most economically handled in properly constructed sumps.

The great disadvantage of such excavations is the space they occupy, their unsightly appearance, and the huge deposits of sand which gradually surround the well. The removal of this sand finally becomes imperative when all available space for new sumps has been utilized, but its disposal is a problem.

The only use to which it could be put in the north Midway was road surfacing and although this is possibly a very good use it is a more or less expensive one. The method used is simple.

When a sump had been drained of its oil and the sand had become sufficiently dry, it was loaded upon wagons and distributed on the many dirt roads traversing the property. The roads were sometimes dragged to prepare them for the sand but this was frequently dispensed with. A layer of sand 2 in. thick was sufficient to surface the ordinary road, which was immediately opened for traffic in order that the sand might become packed. The roads in that section of the country were in very poor condition, being formed from use and never constructed or built. They were deep with dust and ruts and the beneficial results attained through the use of oil sand were truly remarkable. This sand as it comes from the sumps contains sufficient oil to stain the hands and serves as a binding material for the road surface. It soon packs into a durable layer of asphaltum-like hardness. It withstands the traffic of heavy automobile trucks and is easily maintained by occasional dressings of fresh sand in the summer when the heat softens the mixture.

AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

# PROPOSAL FOR MEMBERSHIP

*Mr.* ______________________________
(Name in Full)

*Occupation* ______________________________

*Address* ______________________________

*is hereby proposed by the undersigned, as a* ______________

______________________________

*of the American Institute of Mining Engineers.*

______________________________
______________________________
______________________________
Signatures of three Members or Associates.

*Place of birth* ______________ *Year of birth* ______

*Education, general and technical, when, where and how acquired, with degrees, if any.*

| *Dates* | |
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*Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)*

| *Dates* | |
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*Present position:* ______________________

______________________

**(If the applicant has previously been proposed as a Member, Associate or Junior Member of the Institute, this fact should be stated.)**

**I agree, if elected, to accept election within four months, and to abide by the Constitution and By-Laws**

*Signature of Applicant:* ______________________

*Dated*______________________*191*

## EXTRACTS FROM THE CONSTITUTION.

### ARTICLE II.—MEMBERS.

**SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * ***

**SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.**

**As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * ***

**Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.**

# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 115 JULY 1916

Published Monthly by the American Institute of Mining Engineers at 212–218 York St., York, Pa., H. A. Wisotzkey, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., Bradley Stoughton, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), $10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, $5 per annum. Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

## SPECIAL TRAIN TO THE ARIZONA MEETING

As already announced, the Arizona Meeting of the Institute is to be held from the 18th to the 25th of September. There will be a special train from New York City to Arizona and return, to take members and guests of the Institute to the meeting. The Committee in charge consists of Walter Douglas, Chairman, Cleveland E. Dodge, John C. Greenway, Arthur S. Dwight and Julius Kruttschnitt, Jr., and while the plans of this Committee are not complete, it may be announced that the train will be run with the requisite number of Pullman, dining and baggage cars to accommodate those who desire to go, which will accompany the party during the entire week of the Arizona Meeting as per the following program, including, of course, the visit to the Grand Canyon.

As soon as details have been worked out, a circular will be issued giving full information concerning the trip.

# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 116 AUGUST 1916

Published Monthly by the American Institute of Mining Engineers at 212–218 York St., York, Pa., H. A. Wisotzkey, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., Bradley Stoughton, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), $10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, $5 per annum. Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

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## TRANSACTIONS FOR 1916

Vol. 51 has been sent to all members who have paid their 1916 dues. Vols. 52 and 53, the last two volumes for 1916, will be shipped about the middle of August to all except those owing current dues. Members who do not receive them within a reasonable time are asked to notify the Secretary.

Back volumes to complete sets may be purchased from the Institute at reduced rates.

---

**Arizona Meeting, Sept. 18 to 25, 1916.**

## ARIZONA MEETING, SEPTEMBER 18 to 25, 1916

For the first time in its history the Institute will hold a meeting in the State of Arizona.

A few years ago Arizona stood third in the copper-producing districts of the United States. Since that time, with the development of porphyry mines, the output has gone up with great rapidity until it not only is the leading district, but its output at the present time is at the rate of nearly double the Montana output, which now stands second in the list.

In connection with this development, most interesting departures have been made in methods of mining, concentration and smelting. Principles of mining and metallurgy of the greatest interest will be covered by papers which will appear in the *Bulletin* and be presented at Technical Sessions, while ample opportunity will be afforded for those who go to the meeting to see all the points of interest to them. In addition, members will be well rewarded by the marvelous natural features to be seen on the trip, including the petrified forests, the cliff dwellings on the Apache Trail and the Grand Canyon of Arizona.

It is planned to start the special train from New York, but members and guests may join the special at other points en route as indicated by the following schedule:

### ITINERARY

**Thursday, September 14th**

*Lv* New York, Grand Central Terminal................. 5.30 p. m.
Dinner in dining car a la carte

*New England Members via B. & A. R. R.*

*Lv* Boston.............. (Regular train).............. 2.00 p. m.
*Lv* Worcester............ (Regular train).............. 3.12 p. m.
*Lv* Springfield........... (Regular train).............. 4.37 p. m.
Dinner in Dining Car a la carte
*Ar* Albany............New York Central R. R......... 7.45 p. m.
*Lv* Albany.............................................. 9.00 p. m.
*Lv* Schenectady......................................... 9.32 p. m.
*Lv* Utica............................................... 11.08 p. m.

**Friday, September 15th**

*Lv* Syracuse............................................ 12.30 a. m.
*Lv* Cleveland........................................... 7.25 a. m.
*Lv* Toledo.............................................. 10.05 a. m.
*Ar* Chicago............................................. 4.00 p. m.
*Lv* Chicago, C. R. I. & P............................... 8.05 p. m.

**Saturday, September 16th**

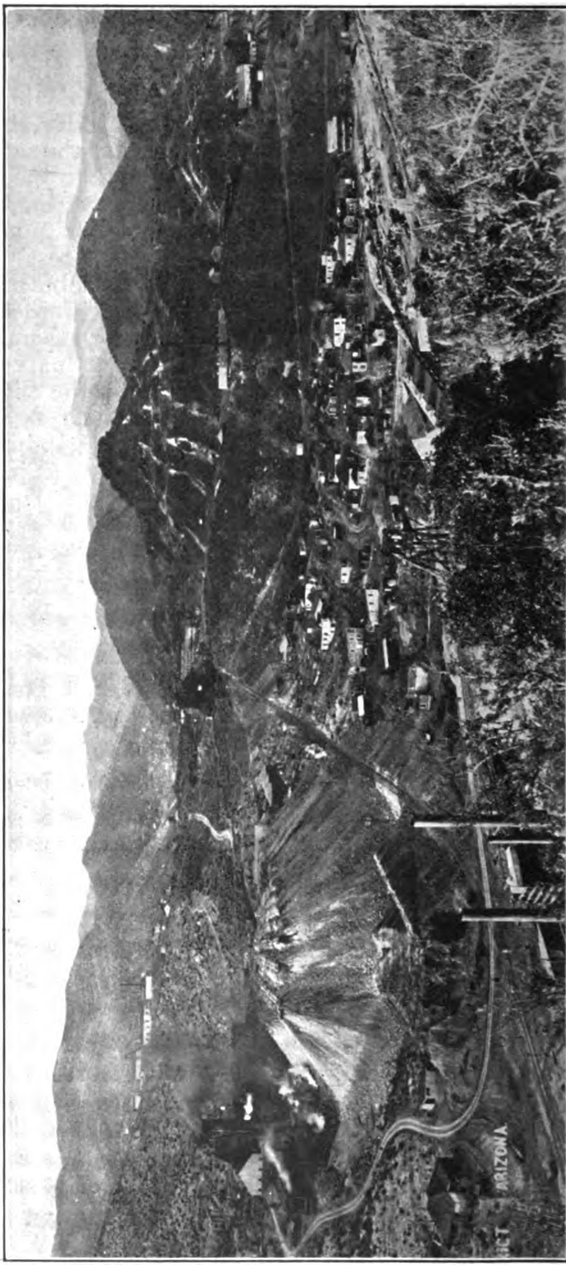
*Lv* Kansas City, C. R. I. & P........................... 11.00 a. m.
*Lv* Topeka, C. R. I. & P................................ 1.05 p. m.
*Lv* Hutchinson.......................................... 6.05 p. m.

**Sunday, September 17th**

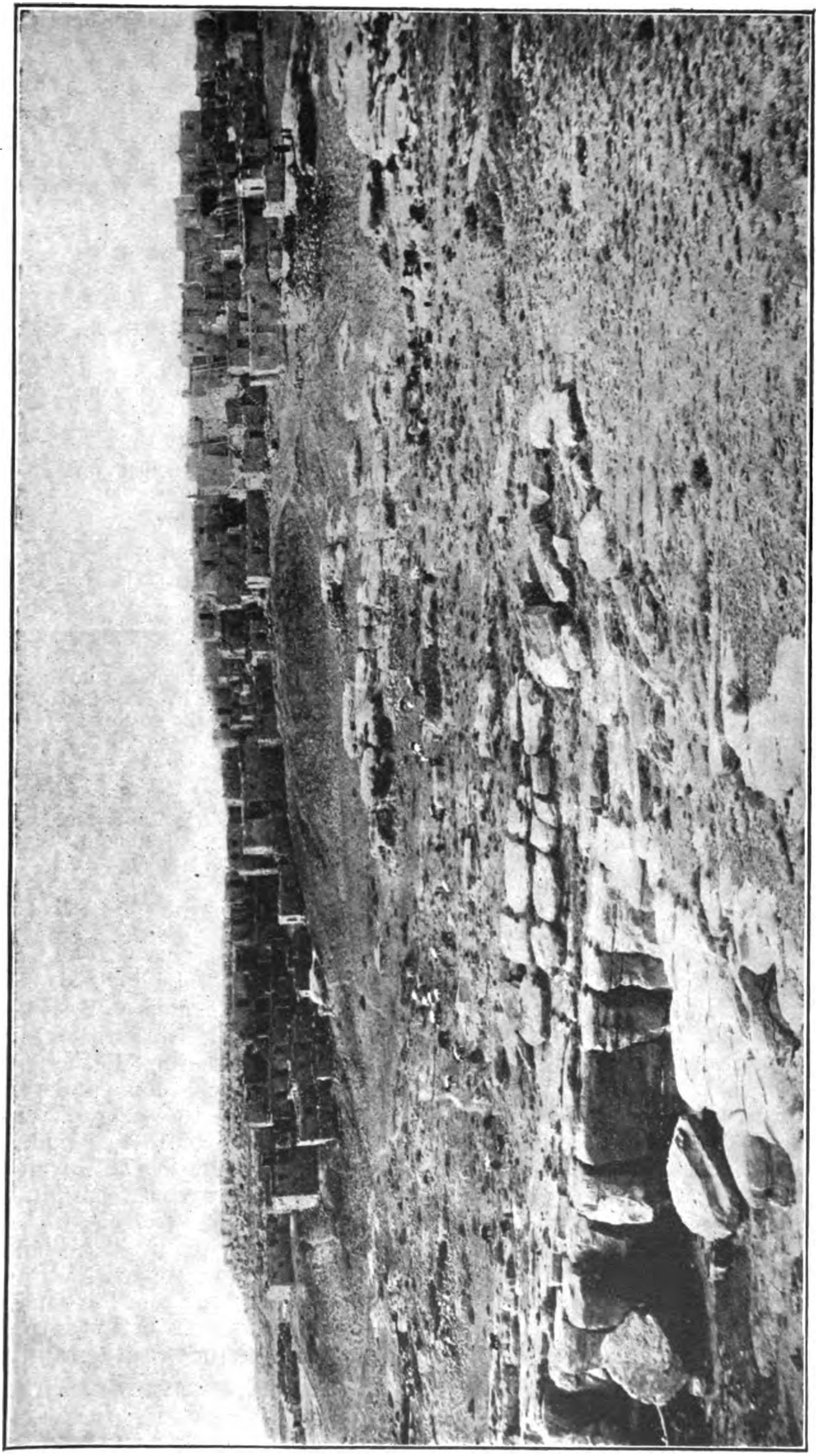
*Lv* Santa Rosa, E. P. & S. W. R. R...................... 6.30 a. m.
*Ar* El Paso............................................. 2.40 p. m.
*Lv* El Paso, Southern Pacific............................ 10.20 p. m.

**Monday, September 18th**

*Av* Santa Rita.......................................... 4.00 a. m.
Visit mines of Chino Copper Co. and mill of Empire Zinc Co.
*Lv* Santa Rita, Sante Fe................................ 2.00 p. m.




BISBEE DISTRICT, ARIZONA, SHOWING SHAFTS OF THE CALUMET & ARIZONA AND COPPER QUEEN MINING COMPANIES.



A HOPI INDIAN VILLAGE IN ARIZONA. (Bulletin 613, U. S. Geological Survey.)

*Ar* Hurley.......................................... 2.45 p. m.
Visit mill of China Copper Co.
*Lv* Hurley, Sante Fe................................ 10.15 p. m.

**Tuesday, September 19th**

*Ar* Douglas......................................... 7.00 a. m.
Visit reduction works of Copper Queen Consolidated Mining Co., and of Calumet & Arizona Mining Co.
Technical Session on "Smelting."
Technical Session on "Leaching."
*Lv* Douglas, E. P. & S. W........................... 11.00 p. m.

**Wednesday, September 20th**

*Ar* Bisbee, Arizona................................. 1.00 a. m.
Visit to mines of district.
Technical Session, subject "Mining and Geology"
*Lv* Bisbee, E. P. & S. W............................ 10.00 p. m.

**Thursday, September 21st**

*Ar* Globe........................................... 9.00 a. m.
Visit to mines and reduction works of Old Dominion Copper Co.
Technical Session, subject "Concentration and Flotation."
Technical Session, subject "Fine Grinding."

**Friday, September 22nd (Globe)**

Visit reduction works of International Smelting & Refining Co., and mills of The Inspiration Consolidated Copper Co., and the Miami Copper Co.

Technical Session, subject "Mining and Smelting."

(The Ray Consolidated Copper Co. will welcome members and guests who desire to visit the mine at Ray. Special arrangements must be made by omitting other features on the program and with the understanding that those who go to Ray must reach Phoenix in time to take the train from there.)

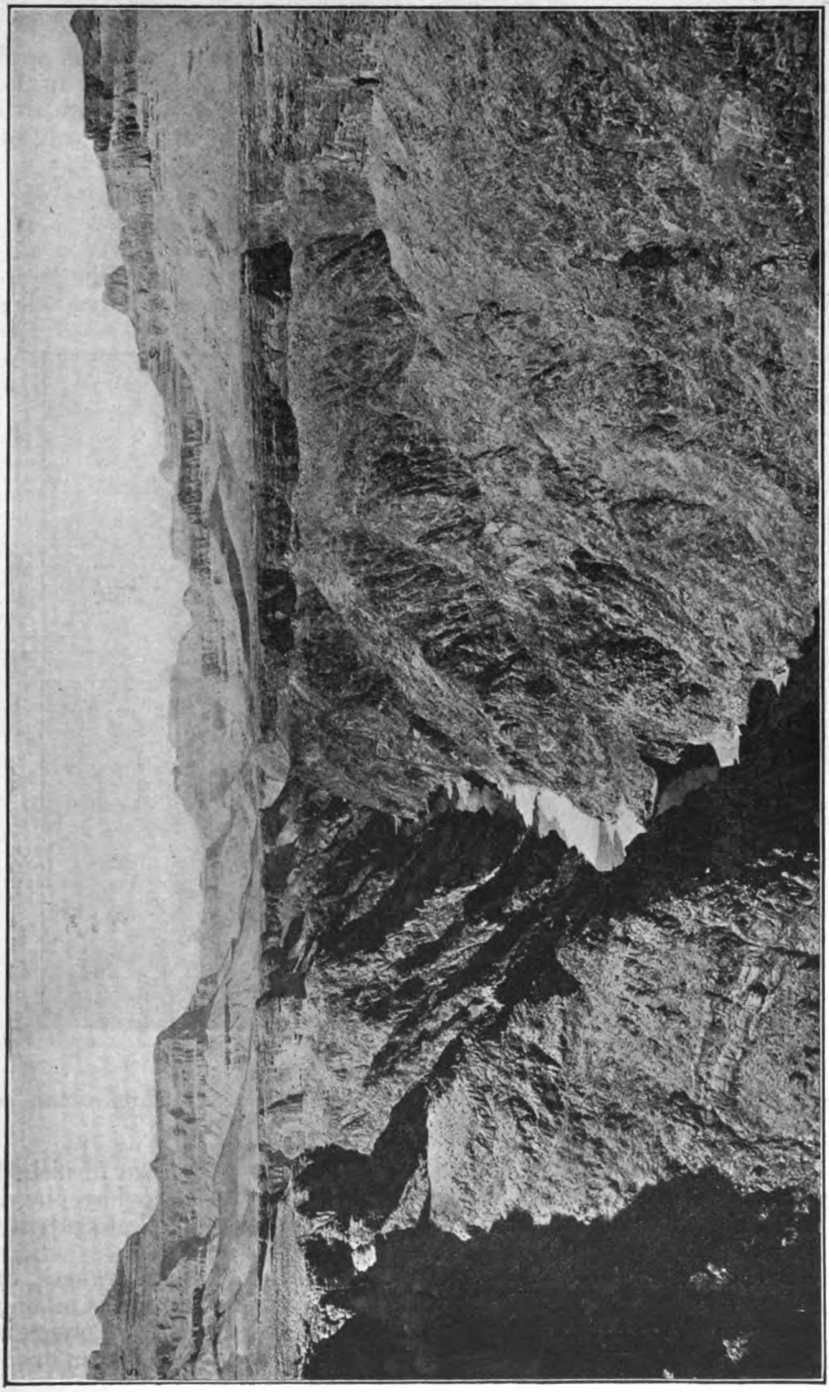
**Saturday, September 23rd (Globe)**

*Lv* Globe by automobile over Apache Trail, including visit to Roosevelt Dam.
*Ar* Phoenix about................................... 5.30 p. m.
*Lv* Phoenix, Sante Fe............................... 8.00 p. m.

## THE APACHE TRAIL

History records that the ancient trail leading to the ruined cliff dwellings in the Salt River Valley was discovered in 1540 by Francisco Vasquez de Coronado, the Spanish explorer, in his search for the treasures of the "seven cities" of Cibola, the abode of the Zuni Indians—the "Fathers of the Pueblos." Later the trail was used by the Apache Indians and this wild and rugged region was one of their last retreats—the scene of desperate encounters with our troopers in the campaign of 1872. Today the modern motor car follows the same historic and romantic trail through scenery of unusual variety and beauty over the highway between Globe and Phoenix, a distance of 120 miles.

The building of the great Roosevelt Dam, which required the construction of the roadway through the wildest and most rugged sections of the route, has made possible this unique mountain tour, and the consequent transformation of the Salt River and Tonto Creek basins into a great lake region has added a charming aspect to views already grandly impressive.



THE GRANITE GORGE IN THE GRAND CANYON, NORTHWEST OF GRANDVIEW POINT. (Bulletin 613, U. S. Geological Survey.)

### CLIFF DWELLINGS

At a point four and one-half miles east of Roosevelt, a wagon trail leads up a small canyon to a point where a foot trail may be taken up the steep mountain side to a group of ancient cliff dwellings which are well worth visiting. Another and more extensive group of ruins can be reached two miles farther by trail.

### ROOSEVELT DAM

This great dam has a maximum height of 280 feet. When the reservoir is full the water backs up fifteen miles in the Tonto and fifteen miles

A WATER BOTTLE IN THE DESERT.
Taking a drink pressed from the pulp that forms the interior of a barrel cactus, or visnaga. (Bulletin 613, U. S. Geological Survey.)

in Salt River. A hydro-electric plant located below the dam furnished current for use during the construction period and now provides power for irrigation pumping on the Salt River Project and for sale to private consumers.

The Salt River Valley is a land of perennial summer, of oranges, of cotton and of dates. Alfalfa is a staple, and ostrich farming is among the successful enterprises. Irrigation has hastened the valley's progress and confirmed its business stability. Phoenix is its metropolis, an energetic and attractive city, and the capital of the State.

**Sunday, September 24th**

| | |
|---|---|
| *Ar* Grand Canyon | 8.00 a. m. |
| *Lv* Grand Canyon, Sante Fe | 7.40 p. m. |

## GRAND CANYON OF ARIZONA

The series of tremendous chasms which form the channel of the Colorado River in its course through northern Arizona, reach their culmination in a chaotic gorge, 217 miles long, from 9 to 13 miles wide, and more than 6,600 ft. below the level of the plateau.

Standing upon the brink of the plateau at the point of the canyon's greatest width and depth, the beholder is confronted by a scene whose majesty and beauty are indescribable. It is one of the few widely advertised spots which one need not fear approaching with anticipations too exalted. It has never been adequately described, and never will be.

The picture on the cover of this *Bulletin* is of the north side of the Grand Canyon as viewed by telescope from El Tovar Hotel. This illustration and others of Arizona scenery herewith reproduced are from *Bulletin* 613, *U. S. Geological Survey*, Guidebook of the Western United States, Part C, The Santa Fe Route. These cuts were obtained through the courtesy of Dr. George Otis Smith, Director. The letters indicate rock formations as follows:

*G*, granite and gneiss; *U*, sandstone, red shale, and limestone (Unkar); *T*, sandstone of Tonto Platform; *Sh*, shale of Tonto group lying directly on quartzite of Unkar, *R*, limestone (Redwall); *S*, red sandstone and shale (Supai); *C*, gray sandstone (Coconino); *K*, limestone (Kaibab). The Redwall butte in center is Cheops Pyramid. Beyond it are Buddha and Manu temples. The background is the Kaibab Plateau.

**Monday, September 25th**

| | |
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| *Lv* Albuquerque, Santa Fe | 7.00 p. m. |

**Tuesday, September 26th**

| | |
|---|---|
| *Lv* La Junta, Sante Fe | 7.10 a. m. |
| *Lv* Kansas City, Sante Fe | 11.00 p. m. |

**Wednesday, September 27th**

| | |
|---|---|
| *Ar* Chicago | 11.15 a. m. |
| *Lv* Chicago, New York Central | 12.40 p. m. |
| *Lv* Cleveland, New York Central | 7.35 p. m. |

**Thursday, September 28th**

| | |
|---|---|
| *Ar* Albany | 6.27 a. m. |
| *Ar* Boston | 11.55 p. m. |
| *Ar* Grand Central Terminal | 9.40 a. m. |

Fares quoted below include railroad and sleeping car fares for the entire trip. Fares from New York and Albany also include excess fares applying to Chicago and return owing to extra fast schedule of train.

For the exclusive use of a compartment by one person, one and one-half railroad tickets are required and for a drawing room, two tickets are required.

The arrangements as announced, also railroad and sleeping car fares

shown below, are conditional upon one hundred fares New York to El Paso, one hundred twenty-five fares El Paso to Grand Canyon and one hundred fares Grand Canyon to New York.

| Starting point | Lower 1 Person | Upper 1 Person | Compartment 2 Persons (Each) | Drawing Room 2 Persons (Each) |
|---|---|---|---|---|
| New York | $228.50 | $217.50 | $233.50 | $261.00 |
| Albany | 216.60 | 205.60 | 221.60 | 249.10 |
| Schenectady | 208.96 | 197.96 | 213.96 | 241.46 |
| Utica | 205.84 | 194.84 | 210.84 | 238.34 |
| Syracuse | 203.84 | 192.84 | 208.84 | 236.34 |
| Rochester | 200.60 | 189.60 | 205.60 | 233.10 |
| Buffalo | 198.40 | 187.40 | 203.40 | 230.90 |
| Cleveland | 188.80 | 177.80 | 193.80 | 221.30 |
| Toledo | 186.30 | 175.30 | 191.30 | 218.80 |
| Chicago | 176.80 | 165.80 | 181.80 | 209.30 |
| Kansas City | 160.30 | 149.30 | 165.30 | 192.80 |

## APPLICATIONS

Those desirous of joining the Tour should promptly communicate with Mr. W. V. Lifsey, Assistant General Passenger Agent, New York Central Lines, 1216 Broadway, New York City. Reservations will be made in the order in which applications are received.

Members of the Arizona Section desiring accommodations on the Special Train should communicate with Mr. Arthur Notman, Box 400, Bisbee, Ariz.

## PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JUNE 23, 1916

Report was made by the Committee of National Engineering Societies on the Reserve Corps of Engineers and it was voted that a committee of three be formed—consisting of the President of the Institute, Dr. Henry S. Drinker and the Secretary of the Institute—to select members in different parts of the country to advise with the local U. S. Engineer Officers as to qualifications of applicants for the Reserve Corps of Engineers.

## NOMINATIONS FOR OFFICERS

The suggestions of the members of the Institute are very much desired by the Committee on Nominations prior to deciding upon its nominations to fill the places of those officers who retire early in 1917. The Committee must make its final report to the Board of Directors early in October. The members are, therefore, asked to send in promptly their suggestions for Directors as well as for Vice-Presidents and President. The Committee hopes to hear from a large number of the members.

The officers to be elected at the annual meeting in February, 1917, are:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N.B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, L. D. Ricketts (not eligible for re-election).

Past President, Benjamin B. Thayer.

Vice-Presidents, H. C. Hoover, District 6; Joseph W. Richards, District 2.

Directors, Reginald W. Brock, District 11; Albert R. Ledoux, District 0; D. C. Jackling, District 7; Charles W. Merrill, District 6; Henry L. Smyth, District 1.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, 1208 Hollingsworth Bldg., Los Angeles, Cal.

SEELEY W. MUDD,
*Chairman, Committee on Nominations.*

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## ENGINEERS' RESERVE CORPS

The Army Reorganization Act, which became effective July 1, 1916, contains a provision authorizing the organization of an Officers' Reserve, in which will be included a Corps of Reserve Engineers. There will also be an Enlisted Reserve of skilled workmen in technical lines. The special provisions of the bill relating to the Engineer Officers' Reserve were formulated with the help of the Joint Committee of the several national engineering societies, which has been at work on the matter for over a year with the War Department and Committees of Congress.

Recognizing that engineering science is the basis of success in modern

warfare, it is apparently the purpose, in thus mobilizing the engineering talent of the country, to give every engineer or technical specialist an opportunity, if the country can use his services, to do that which he can do best, and not necessarily to try and train all to the same standard of military engineering work for service with troops. At the same time, exceptional opportunities are offered to those who wish to qualify for this kind of active service and for those engineers who are inclined to seek commissions as engineer officers in the permanent establishment. It will be noted that the usual age limits are waived in the case of the Engineers' Reserve Corps.

But for the disarrangement of the ordinary routine of the War Department, resulting from the present emergency on the Mexican border, it is probable that the machinery for organizing the Engineers' Reserve Corps would already be in motion, but under the circumstances, it will be necessary to await the convenience of the authorities.

The report of the Joint Committee follows:

To the Boards of Direction of the

AMERICAN SOCIETY OF CIVIL ENGINEERS,

AMERICAN INSTITUTE OF MINING ENGINEERS,

AMERICAN SOCIETY OF MECHANICAL ENGINEERS,

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS,

AMERICAN INSTITUTE OF CONSULTING ENGINEERS.

DEAR SIRS:—On December 22, 1915, the Joint Committee (consisting of the Chairmen of your several committees), formed under the authority of the five societies in order to facilitate the carrying out of the organization of an Engineer Reserve as part of the military forces of the United States, reported the progress that had been made up to that time in the matter of the necessary legislation.

The Joint Committee has now to report to you that the legislation referred to is embodied in a bill which recently passed Congress, has been signed by the President, and will become effective July 1, 1916. This bill, known as the Army Reorganization Act of 1916, contains provisions for the organization of an Officers Reserve, including engineers. An abstract of the principal clauses relating to the Officers Reserve and to the Enlisted Reserve Corps is appended hereto.

The War Department is now formulating the plan of organization of the Reserve Corps of Engineers and of the Enlisted Reserve Corps and the requirements that will be prescribed for enrollment therein. As soon as practicable after July 1, the War Department will issue an invitation to the engineers of the country to apply for commissions in the Reserve Corps of Engineers, and there will be available for those interested full and complete information as to the requirements for commissions in the several grades from Second Lieutenant to Major, and for Master Engineers and Sergeants, and the method of procedure to secure commissions or enlistment.

The Joint Committee has notified the War Department that its services in these matters are at the call of the Department.

Very truly yours,

WM. BARCLAY PARSONS, Chairman,
*American Society of Civil Engineers,*
H. S. DRINKER,
*American Institute of Mining Engineers,*
WM. H. WILEY,
*American Society of Mechanical Engineers,*
B. J. ARNOLD,
*American Institute of Electrical Engineers,*
RALPH D. MERSHON,
*American Institute of Consulting Engineers.*

23D JUNE, 1916.

## ABSTRACT

## THE ENGINEER SECTION OF THE OFFICERS RESERVE CORPS AND THE ENLISTED RESERVE CORPS

From the Army Reorganization Act of 1916, Sections, 37, 38, 39, 55

SEC. 37.—The Officers Reserve Corps.—For the purpose of securing a reserve of officers available for service as temporary officers in the Regular Army, as provided for in this Act and in Section eight of the Act approved April twenty-fifth, nineteen hundred and fourteen, * * * there shall be organized, under such rules and regulations as the President may prescribe not inconsistent with the provisions of this Act, an Officers Reserve Corps of the Regular Army. Said corps shall consist of sections corresponding to the various arms, staff corps, and departments of the Regular Army. Except as otherwise herein provided, a member of the Officers Reserve Corps shall not be subject to call for service in time of peace, and whenever called upon for service shall not, without his consent, be so called in a lower grade than that held by him in said reserve corps.

The President alone shall be authorized to appoint and commission as reserve officers in the various sections of the Officers Reserve Corps, in all grades up to and including that of major, such citizens as, upon examination prescribed by the President, shall be found physically, mentally and morally qualified to hold such commissions: *Provided*, That the proportion of officers in any section of the Officers Reserve Corps shall not exceed the proportion for the same grade in the corresponding arm, corps, or department of the Regular Army, except that the number commissioned in the lowest authorized grade in any section of the Officers Reserve Corps shall not be limited.

* * * * * * * *

The commissions of all officers of the Officers Reserve Corps shall be in force for a period of five years unless sooner terminated in the discretion of the President. Such officers may be recommissioned, either in the same or higher grades, for successive periods of five years, subject to such examinations and qualifications as the President may prescribe and to the age limits prescribed herein: *Provided*, That officers of the Officers Reserve Corps shall have rank therein in the various sections of said Reserve Corps according to grades and to length of service in their grades.

SEC. 38.—The Officers Reserve Corps in War.—In time of actual or threatened hostilities the President may order officers of the Officers Reserve Corps, subject to such subsequent physical examinations as he may prescribe, to temporary duty with the Regular Army in grades thereof which can not, for the time being, be filled by promotion, or as officers in volunteer or other organizations that may be authorized by law, or as officers at recruit rendezvous and depots, or on such other duty as the President may prescribe. While such reserve officers are on such service they shall, by virtue of their commissions as reserve officers, exercise command appropriate to their grade and rank in the organizations to which they may be assigned, and shall be entitled to the pay and allowances of the corresponding grades in the Regular Army, with increase of pay for length of active service, as allowed by law for officers of the Regular Army, from the date upon which they shall be required by the terms of their orders to obey the same: *Provided*, That officers so ordered to active service shall take temporary rank among themselves and in their grades in the organizations to which assigned, according to the dates of orders placing them on active service; and they may be promoted, in accordance with such rank, to vacancies in volunteer organizations or to temporary vacancies in the Regular Army thereafter occurring in the organizations in which they shall be serving. * * *

Any officer who, while holding a commission in the Officers Reserve Corps, shall be ordered to active service by the Secretary of War shall, from the time he shall be required by the terms of his order to obey the same, be subject to the laws and regulations for the government of the Army of the United States, in so far as they are applicable to officers whose permanent retention in the military service is not contemplated.

SEC. 39.—Instruction of Officers of the Officers Reserve Corps.—To the extent provided for from time to time by appropriations for this specific purpose, the Secretary of War is authorized to order reserve officers to duty with troops or at field exercises, or for instruction, for periods not to exceed fifteen days in any one calendar year, and while so serving such officers shall receive the pay and allowances of their respective grades in the Regular Army: *Provided*, That, with the consent of the reserve officers concerned, and within the limit of funds available for the purpose, such periods of duty may be extended for reserve officers as the Secretary of War may direct. * * *

* * * * * * * *

Sec. 55.—The Enlisted Reserve Corps.—For the purpose of securing an additional reserve of enlisted men for military service with the Engineer * * * Corps * * * of the Regular Army, an Enlisted Reserve Corps, to consist of such number of enlisted men of such grade or grades as may be designated by the President from time to time, is hereby authorized, such authorization to be effective on and after the first day of July, nineteen hundred and sixteen.

There may be enlisted in the grade or grades hereinbefore specified, for a period of four years, under such rules as may be prescribed by the President, citizens of the United States, or persons who have declared their intentions to become citizens of the United States, subject to such physical, educational and practical examination as may be prescribed in said rules. For men enlisting in said grade or grades certificates of enlistment in the Enlisted Reserve Corps shall be issued by the Adjutant General of the Army, but no such man shall be enlisted in said corps unless he shall be found physically, mentally and morally qualified to hold such certificate and unless he shall be between the ages of eighteen and forty-five years. The certificates so given shall confer upon the holders when called into active service or for purposes of instruction and training, and during the period of such active service, instruction, or training, all the authority, rights and privileges of like grades of the Regular Army. * * * And the Secretary of War is hereby authorized to issue to members of the Enlisted Reserve Corps and to persons who have participated in at least one encampment for the military instruction of citizens, conducted under the auspices of the War Department, distinctive rosettes or knots designed for wear with civilian clothing.

The President is authorized to assign members of the Enlisted Reserve Corps as reserves to particular organizations of the Regular Army, or to organize the Enlisted Reserve Corps, or any part thereof, into units of detachments of any arm, corps, or department in such manner as he may prescribe, and to assign to such units and detachments officers of the Regular Army or of the Officers Reserve Corps, herein provided for.

To the extent provided from time to time by appropriations, the Secretary of War may order enlisted men of the Enlisted Reserve Corps to active service for purposes of instruction or training for periods not to exceed fifteen days in any one calendar year: *Provided*, That, with the consent of such enlisted men and within the limits of the funds available for such purposes, such periods of active service may be extended for such number of enlisted men as may be deemed necessary.

Enlisted men of the Enlisted Reserve Corps shall receive the pay and allowances of their respective grades, but only when ordered into active service, including the time required for actual travel from their homes to the places to which ordered and return to their homes. * * *

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## PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period June 10, 1916 to July 10, 1916:

**Arthur Adams**, Miami, Ariz.
**J. H. Batcheller**, Mattapoisitt, Mass.
**R. S. Bonsib**, Low Moor, Va.
**E. L. Booth**, New York, N. Y.
**C. L. Bryden**, Scranton, Pa.
**F. Crabtree**, Pittsburgh, Pa.
**L. B. Eames**, Pueblo, Colo.
**S. J. Gormly**, Guayacan, Chile.
**Ernest A. Hersam**, Berkeley, Cal.
**A. W. Hudson**, Douglas, Ariz.
**Chas. W. Johnston**, West Norfolk, Va.
**John M. Lovejoy**, New Rochelle, N. Y.
**J. F. Manning**, Seoul, Korea.
**E. P. Mathewson**, Anaconda, Mont.
**Edwin W. Mills**, Seoul, Korea.
**Arthur Notman**, Bisbee, Ariz.
**L. B. Reifsneider**, Daiquiri, Cuba.
**W. S. Schuyler**, Berkeley, Cal.
**O. N. Scott**, Toronto, Ont., Canada.
**Arthur F. Taggart**, New Haven, Conn.
**Warren D. Thompson**, Boston, Mass.
**Thor. Warner**, Kingman, Ariz.

**Rodolphe L. Agassiz** has been chosen president of the Calumet & Hecla Mining Co., an office which has been vacant for over a year.

**Percy E. Barbour**, Managing Editor of the *Engineering and Mining Journal*, resigned his commission as 2d Lieutenant in the Main Coast

Artillery to accept a commission as 1st lieutenant in the 22d Corps of Engineers, N. G. N. Y. He is now at Fort McAllen, Tex.

**H. Kenyon Burch,** Chief Engineer of the Inspiration Consolidated Copper Co., completed his work pertaining to the design and construction of the plant, and left on July 1st for an extended vacation trip throughout the East. His forwarding address for the next few months will be care of The Sierra Madre Club, L. A. Investment Bldg., Los Angeles, Cal.

**Corey C. Brayton** and **E. R. Richards** have established offices in the Hobart Bldg., San Francisco, Cal., for the purpose of conducting the business of Mining and Metallurgical Engineering.

**Juan Felix Brandés** has removed his offices to 139 Whittell Bldg., 166 Geary St., San Francisco, Cal.

**S. A. Crandall** has accepted a position as draftsman with the Bunker Hill and Sullivan Smelter, Kellogg, Ida.

**John C. Devine,** since 1906 assistant superintendent of the Ray Consolidated Copper Co., has resigned and will devote his attention to the development of the Renfro property of the Pinal Development Co., of which he is president.

**Edward B. Durham** is now with the Mammoth Copper Mining Co., Kennett, Cal.

**Edward L. Estabrook** has become geologist for the Wisconsin Zinc Co., at Platteville., Wis.

**F. Julius Fohs,** who has devoted himself to a consulting practice in oil geology for the past four years, announces that he has opened new offices in the Gallais Building at Tulsa, Okla., with a staff of six geologists and one topographer, and that he is covering an increasing scope of country with his structural surveys.

**George A. Guess,** professor of metallurgy at the University of Toronto, has been engaged by the Vermont Copper Co. to start up the copper smeltery at South Strafford, Orange Co., Vt., which has never been operated successfully.

**Henry M. Howe** has just been appointed Honorary Vice-President of the Iron and Steel Institute of Great Britain. There are only two others, Professor D. Tschernoff, of Russia; and Giorgio E. Flack, of Italy.

**Capt. Robert W. Hunt** has been honored by the Rensselaer Polytechnic Institute, with which he has been long identified, with the degree of Doctor of Engineering.

**E. T. Lednum** has been made manager of E. I. DuPont de Nemours & Co., with offices in Suite 808, Frisco Bldg., Joplin, Mo.

**E. Fleming L'Engle** has been appointed manager of the Royal Zinc Co., Joplin, Mo.

**Waldemar Lindgren,** Professor of Economic Geology at Massachusetts Institute of Technology, has been awarded the honorary degree of Doctor of Science by Princeton University.

**N. O. Lawton** has completed the exploration work in which he was engaged in Hancock Co., Tenn., and will be at South Strafford, Vermont, during July, examining copper properties.

**C. H. Macnutt** has resigned his position as manager of Burma Mines Co., Namtu, Burma.

**W. G. Matteson** has accepted the position of geologist for the Empire Gas and Fuel Co., Bartlesville, Okla.

**William B. Phillips** has resigned the presidency of the Colorado School of Mines, and has returned to Austin, Texas, to resume practice as a mining engineer and metallurgist.

**Fred S. Porter** has accepted a position in the engineering office of the Canadian Klondike Mining Co., Dawson, Yukon Territory, Canada.

**F. Danvers Power** has opened new offices in the Perpetual Trustee Chambers, Hunter St., Sydney, N. S. W., Australia.

**H. W. Ross** has accepted the position of Assistant Manager of the Backus & Johnston Co., Casapalca, Peru.

**J. D. Sperr** has resigned as mining engineer of the Tom Reed Gold Mines Co., Oatman, Ariz.

**F. W. Traphagen** has resigned the position of professor of metallurgy of the Colorado School of Mines to accept the presidency of the Colorado Metal Mining & Reduction Co.

**Carl J. Trauerman,** mining engineer, has resigned his position as mill superintendent of the August Mining Co., Landusky, Mont., and is inspecting the properties of the Beaver Creek Mines Co., Zortman, Mont. After examining properties in the Kendall, Elliston and York districts of Montana he will return to his headquarters at Butte.

**Jesse Merrick Smith** and **Miss Mabel Amy MacKinney** were married on June 24, at Tarrytown-on-the-Hudson, N. Y.

**Edward Vestal Graybeal** and **Miss Alice Seabury Mathewson** were married on June 14, at Anaconda, Mont.

## POSITIONS VACANT

Wanted, for a new smelting company now being organized in China: one man familiar with the design and operation of a zinc smelter; one experienced in the design and operation of a lead smelter; and a chemist competent to make the usual analyses in connection with lead and zinc smelting and to make clay analyses. Men with their wives preferred. Company will pay all expenses to and from China, medical attendance, salary, bonus per ton of zinc or lead produced, allowance for food, and furnish quarters. No. 120.

## ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, oil geologist, age 33 years, technical education, nine years' experience with oil and mining companies, wishes employment with repu-

table company or syndicate exploiting proven oil lands or searching for new districts. First class references. No. 300.

Member, age 30, unmarried, Columbia graduate, speaks fair French, and writes and speaks Spanish fluently. Mining and geological experience in United States, Mexico and South America. Desires position with examining engineers or exploration company. No. 301.

Member, technical graduate in mining engineering, three years' experience in surveying, sampling and underground engineering in metal mines of the West, desires position as mine surveyor, field engineer or assistant to consulting engineer. No. 302.

Member, age 30, technical graduate, 4 years' experience in tin concentration; at present Superintendent of Mills in largest Bolivian tin mines; executive ability; seeks change, preferably with American company interested in locating or investigating in South America. Fluent Spanish. No. 303.

Open for engagement. Master Mechanic. A man experienced in smelter construction and operation. Lead and copper especially. 18 years as M. M. in British Columbia and Mexico. Reliable, efficient, with good record. Best of reference. No. 304.

Engineer, 42 years old, married, experienced in cyanidation, concentration and construction of plants, desires position as operator or construction engineer or in plant designing. Speaks Spanish and some German. No. 305.

Mining engineer with 6 years' operating experience in Colorado and Arizona desires position with substantial company. Employed at present. Age, 30. Married. References. No. 306.

Member, technical graduate, who has specialized in the treatment of gold and silver ores, desires position as mill superintendent or foreman; 7 years' practical experience. Highest references as to character and ability to get results. No. 307.

Member with broad general business experience. Past 5 years president and general manager of active developing and producing oil company in California. Open for engagement in California preferred. No. 308.

Member, technical metallurgical graduate. 5 years' experience in various mining operations. Mill and laboratory work including concentrating, cyaniding and flotation. Desires position in flotation work. Age, 30. Married. No. 309.

Member, aged 30, technical graduate in mining and metallurgy. At present employed as assistant superintendent of concentrator and flotation plant. Also had practical and engineering mining experience. Desires change. No. 310.

Member, aged 26, graduate in mining and metallurgy, 2 years' experience with engineering work of lead and copper smelters, good draftsman; desires permanent connection with engineering or operating department of smelter. Excellent references. At present employed, but available at short notice. No. 311.

## LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

### Apparatus for Making Photographic Reproductions

The United Engineering Society has installed a photographic duplicating machine for the reproduction of pages from books and periodicals, maps and charts in the Library of the Engineering Societies, and it is now in operation. Orders may be sent to the Library of the Engineering Societies, 29 West 39th Street, New York.

The process produces a photograph (white on black) direct on bromide paper, and from this negative print a positive may be made by re-photographing. The charge for a print 11 by 14 in., which may cover two pages of an octavo volme that face each other, is 25 cents for a negative print and 50 cents for a positive print. Enlargements and reductions to 11 by 14 in. are easily made. The process has the advantages of accuracy and cheapness.

## LIBRARY ACCESSIONS

## PARTIAL LIST CLASSIFIED BY SUBJECTS

### Mining and Metallurgy

BY-PRODUCTS RECOVERED IN THE MANUFACTURE OF COKE. By W. H. CHILDS. n.p., 1916. (Gift of D. D. Berolzheimer.)

COAL DISTILLATION, GASIFICATION AND BY-PRODUCTS. By J. E. CHRISTOPHER WIGAN, 1915.

THE ANALYSIS OF PERMISSIBLE EXPLOSIVES. Bull. 96, U. S. Bureau of Mines. Washington, 1916.

SAND TEST FOR DETERMINING THE STRENGTH OF DETONATORS. Tech. Paper No. 125, U. S. Bureau of Mines. Washington, 1916.

ABSTRACTS OF CURRENT DECISIONS ON MINES AND MINING, REPORTED FROM OCTOBER TO DECEMBER, 1915. Bull. No. 118, U. S. Bureau of Mines. Washington, 1916.

EFFECTS OF ATMOSPHERES DEFICIENT IN OXYGEN ON SMALL ANIMALS AND ON MEN. Tech. Paper No. 122, U. S. Bureau of Mines. Washington, 1915.

MINERS' NYSTAGMUS. Bull. No. 93, U. S. Bureau of Mines, Washington, 1916.

LIST OF BOOKS RELATING TO THE IRON AND STEEL INDUSTRY. Contained in the principal libraries in the West of Scotland, Glasgow, 1916. (Gift of West of Scotland Iron and Steel Institute.)

### Geology and Mineral Resources

REVIEW OF PRINCIPLES OF OIL AND GAS PRODUCTION. Compiled by ROSWELL H. JOHNSON and L. G. HUNTLEY. John Wiley & Sons, Inc., New York, 1916. Price $3.75. Gift of publishers.

[A book on oil by the two distinguished authors of this volume is sure to arouse great interest among members of the Institute who are familiar with the writings of these same authors in our publications. To such it is not necessary for us to say anything as to the accuracy, care, and thoroughness of the work, and we commend the volume to all. It is well and appropriately illustrated by geological sketches, drawings, curves, photographs, and so forth, and it covers the following subjects:

Varieties of Oil and Gas, The Origin of Oil and Gas, Distribution of the Oil and Gas, Reservoirs of Oil and Gas, Accumulation of Oil and Gas, Pressure in Oil and Gas Reservoirs, Origin of the Shape of the Reservoir, Classification of the Attitude of Geologic Surfaces, Effect of the Different Attitudes upon Accumulation, Locating Oil and Gas Wells, Oil and Gas Lands, Drilling for Oil and Gas, Bringing in a Well, The Management of Oil Wells, Completing the Extraction of the Oil, The Management of Gas Wells, Condensation of Gasoline from Gas, The Natural Gas Industry, Size and Scope of Oil and Gas Companies, Reports upon Oil and Gas Prospects or Properties, The Valuation of Oil Properties, Oil and Gas Fields of North America, Oil Market and the Future Supply.]

DEPOSITS OF THE USEFUL MINERALS AND ROCKS, THEIR ORIGIN, FORM AND CONTENT. By F. Beyschlag and others. Vol. II, London, 1916.

OIL AND GAS MAP OF SOUTHWESTERN PENNSYLVANIA, 1915. Harrisburg, 1916.

THE CADDO OIL AND GAS FIELD, LOUISIANA AND TEXAS. Bull. No. 619, U. S. Geological Survey. Washington, 1916.

PETROLEUM AND NATURAL GAS PROSPECTS AT ROMA. Publication 247, Queensland Geological Survey. Brisbane, 1915.

TUNGSTEN AND MOLYBDENUM. Part I–II. (Tasmania. Geological Survey Mineral Resources, No. 1.) Tasmania, 1916.

THE WOLFRAM, MOLYBDENITE AND BISMUTH MINES OF BAMFORD, NORTH QUEENSLAND. Publication 248, Queensland Geological Survey. Brisbane, 1915.

FELDSPARS OF THE NEW ENGLAND AND NORTH APPALACHIAN STATES. Bull. No. 92, U. S. Bureau of Mines. Washington, 1916.

ZINC-LEAD SULPHIDE DEPOSITS OF THE READ-ROSEBERY DISTRICT PT. II.—Rosebery group with maps. Bull. No. 23, Tasmania Geological Survey. Tasmania, 1915.

ATLANTIC GOLD DISTRICT AND THE NORTH LARAMIE MOUNTAINS, FREMONT, CONVERSE, AND ALBANY COUNTIES, WYOMING. Bull. No. 626, U. S. Geological Survey. Washington, 1916.

GEOLOGY AND COAL RESOURCES OF CASTLE VALLEY IN CARBON, EMERY, AND SEVIER COUNTIES, UTAH. Bull. 628, U. S. Geological Survey. Washington, 1916.

GEOLOGY AND MINERAL RESOURCES OF THE SOUTHERN COAL FIELD. Part I—SOUTH COSTAL PORTION. With Maps and Sections. New South Wales. Memoirs of the Geological Survey. Geology No. 7. Sydney, 1915.
GEOLOGY AND ORE DEPOSITS OF ROSSLAND, BRITISH COLUMBIA. Memoir No. 77. Canada Dept. of Mines. Ottawa, 1915.
GEOLOGY OF FIELD MAP-AREA, B. C. AND ALBERTA. Memoir No. 55, Canada Dept. of Mines. Ottawa, 1914.

## General

CIVIL ENGINEERS' COST BOOK. Ed. 2. By T. E. Coleman. London, 1916.
DIRECTORY OF THE IRON AND STEEL WORKS OF THE UNITED STATES AND CANADA, 1916. 18th Ed. New York, 1916.
LAND AND MARINE DIESEL ENGINES. By Giorgio Supino. Translated by A. G. Bremner and James Richardson. London, 1915.
THE DIESEL ENGINE IN PRACTICE. By J. E. Megson and H. S. Jones. San Francisco, 1916.

## Gift of Hill Publishing Company

A good collection of 29 text books.

## Trade Catalogs

E. I. DUPONT DE NEMOURS & Co. Wilmington, Del. DuPont Products (A-430). April, 1916.
SULLIVAN MACHINERY Co., Chicago, Ill. Bull. 70-C. The Sullivan "Dr-6" mounted water hammer drill. May, 1916.
SPARTA IRON WORKS Co., Sparta, Wis. Sludge Bucket. June, 1916.
STEPHENS-ADAMSON MFG. Co., Aurora, Ill. Labor Saver. June, 1916.
TEXAS Co., Houston & New York. About Texaco crater compound, 1916.

## Exposition of Chemical Industries

That the chemical industries are alive to the situation that they now occupy in the sum of affairs is evident in the preparations being made by many firms for the Second National Exposition of Chemical Industries to be held at the Grand Central Palace, New York City, during the week of September 25–30, 1916. The list of exhibitors indicates that the Exposition is already twice the size of its successful predecessor.

The American Chemical Society will hold its annual meeting during the whole week, the American Electrochemical Society has arranged to hold its meetings on September 28, 29 and 30, and the Technical Association of the American Pulp and Paper Industry is also planning to meet during the week.

The Bureau of Commercial Economics at Washington is again coöperating with the Exposition by arranging an elaborate program of motion pictures covering subjects dealing with the industries depending on chemistry. Many of these films will have their first showing at the Exposition. They will take the visitor through industrial plants, shops, manufactories, mines, etc., where visitors are never admitted; they will show operations that in ordinary light would never be seen; they will demonstrate more than any other kind of motion picture the great strides and improvement in the art of photography.

# MEMBERSHIP

## NEW MEMBERS

The following list comprises the names of those persons who became members during the period June 10, 1916, to July 10, 1916:

ADAMS, WILLIAM S., Filtration Engr., Anaconda Copper Min. Co., 701 Cherry, Anaconda, Mont.
ALLEY, JAMES E., Shift-boss............Belmont Milling Co., Tonopah, Nev.
BADGLEY, CHARLES WELLING, Research Chemist, El Paso Smelting Works, El Paso, Tex.
BALLENBERG, ADOLF G., Engr., Porcupine Exploration Co., Ontonagon Co., Mich.
BARKER, RALPH FEINER, Met..................Garfield Smelt. Co., Garfield, Utah.
BLOSSOM, EDWARD LUDLAM, Staff of Research Corpn., 63 Wall St., New York, N. Y.
BROUGHTON, EUGENE HARDING, Experimental Engr., Copper Queen Cons. Min. Co., Bisbee, Ariz.
BURFORD, SAMUEL W., Genl. Mgr., Cleveland Mining Co., 223 West Third St., Joplin, Mo.
BUTTERFIELD, GEORGE BRUCE, Inspector, The Associated Companies, First National Bank Bldg., Pittsburgh, Pa.
CAHILL, EDWARD GUSTAV, Contracting Engr., 460 Montgomery St., San Francisco, Cal.
DAULTON, THEODORE M.....................208 New York Bldg., Seattle, Wash.
DEANE, ELBRIDGE GERRY, Min. Engr..........Miami Copper Co., Miami, Ariz.
DUNCAN, L. GRANT, Met. Engr.................Mineral Point Zinc Co., Depue, Ill.
EISELE, GEORGE J., Asst. Genl. Supt., Oliver Iron Mining Co., Iron Mountain, Mich.
FLYNN, JOHN G......................Mining, Miami Copper Co., Miami, Ariz.
FRENCH, R. W., Met...............Goldfield Cons. Mines Co., Goldfield, Nev.
FREUD, BENJAMIN BALL, Teacher, Associate Prof. of Chemistry, Armour Institute of Technology, Chicago, Ill.
GARLOCK, GUY M., Fuel Dept., Prime Western Spelter Co., Box 83, Iola, Kans.
GEOHEGAN, LESLIE E.......Asst. to Prest., Gulf States Steel Co., Birmingham, Ala.
GOODALE, F. A., Supt. and Agent, You Bet Mining Co., P. O. Box. 290, Dutch Flat, Cal.
HANDY, ROYAL SHEPPART, Mill Supt., Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
HARDING, PHILIP ORR, Engr., Designing Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
HENSLEY, JAMES H. JR., Min. Engr., Miami Copper Co., Box 100, Miami, Ariz.
HERSHEY, OSCAR H., Min. Geol., Burch, Caetani & Hershey, Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
HOFFMANN, ROBERT BERNARD, Chief of Service Dept., Mineral Point Zinc Co., Depue, Ill.
HULL, MONROE RICHMOND.....Chief Engr., Arizona Copper Co., Clifton, Ariz.
JONES, J. CLAUDE, Prof. of Geol............University of Nevada, Reno, Nev.
KELSEY, W. M., Genl. Supt.................Mineral Point Zinc Co., Depue, Ill.
LIDDELL, DONALD M., Cons. Met. Engr., Merrill, Lynch & Co., 7 Wall St., New York, N. Y.
LUNT, CHARLES FREDERICK, Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
MANN, HORACE THARP, Associate Prof. of Met. and Ore Dressing, Missouri School of Mines and Metallurgy, P. O. Box 248, Rolla, Mo.
MENDELSOHN, CHARLES, Mech. Engr., Old Dominion Copper Min. & Smelt. Co,. Box 657, Globe, Ariz.
MOORE, GEORGE PIERSON, Engr., The Henry Souther Engrg. Co., 11 Laurel St., Hartford, Conn.
READ, NORMAN HATFIELD, Min. Geol......................Manchester, Mass.
ROBBINS, O. D., Met. Engr., Chief of Ore and Testing Dept., Mineral Point Zinc Co., Depue, Ill.
SCOTT, DAVID BURNET, Min. Engr., Efficiency Engr., Miami Copper Co., Box 100, Miami, Ariz.
SCOTT, J. RALPH, Mine Engr., Dome Mines Co., Ltd., Lock Box 505, So. Porcupine, Ont., Canada.
SMITH, W. R., Supt..................Mineral Point Zinc Co., Mineral Point, Wis.

STAY, THERON D., Asst. Research Met., Aluminum Castings Co.,
2077 East 88th St., Cleveland, O.
STEIDLE, EDWARD, Technical Asst. to Chief Min. Engr., U. S. Bureau of Mines,
Washington, D C.
STOCKETT, NORMAN AYLEWORTH, Min. Engr., Mgr., Paragon Cons. Min. Co.,
Paragon, Idaho.
STRACHAN, CHARLES B., Mill Supt., American Zinc Co. of Tennessee, Mascot, Tenn.
STRAUB, CHARLES EDWARD, Cons. Oil Geol. & Engr.,
211–12 Lynch Bldg., Tulsa, Okla.
STROUP, THOMAS A., Mech. and Met. Engr., Tennessee Copper Co., Copperhill, Tenn.
TODD, W. PARSONS, Vice-Prest., Quincy Min. Co., 32 Broadway, New York, N. Y.
WATSON, HUGH C., Min. Engr., Genl. Supt., Wisconsin Zinc Co., Platteville, Wis.
WARD, ARTHUR THOMAS, Experimental Engr., Braden Copper Co., Sewell,
via Rancagua, Chile, S. Amer.
WEINTZ, LOUIS JOHN, Mill Supt., The New Jersey Zinc Co.,
Ogdensburg, Sussex Co., N. J.
WHITE, HENRY ARTHUR, Cons. Met., Geduld Prop. Mines, Ltd., Box 41,
Springs, Transvaal, S. Africa.
WOODWARD, TRUMAN STEPHEN, Chief Chemist, Ohio Works, Carnegie Steel Co.,
R. 2. Youngstown, O.
YERXA, R. B., Asst. Supt. of Concentrator, Miami Copper Co.,
P. O. Box 100, Miami, Ariz.
YOUNG, EARL BURDETT, Asst. in Geol. Dept., Anaconda Copper Min. Co.,
526 Hennessy Bldg., Butte, Mont.

*Associate Members*

CAJANDER, A. W., Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
RIVES, HENRY MACON, Secy. and Treas., Nevada Mine Operators Assn.,
210-211 Reno National Bank Bldg., Reno, Nev.
ROBINSON, CHARLES H., Iron Mining, Pettit & Robinson Lands,
826 First National Soo Line Bldg., Minneapolis, Minn.

*Junior Members*

BERTHIER, ULYSSES HENRY, Asst. in Testing Dept., International Smelt. Co.,
Box 115, Tooele, Utah.
BIRKETT, HOWARD, Student..........Columbia University, New York, N. Y.
BLAUROCK, CARL ALBERT, Gold and Silver Refiner, 2530 W. 37th Ave., Denver, Colo.
CHATIN, AUGUST H., Laborer, Engr. of Mines, Anaconda Copper Co.,
316½ W. Copper St., Butte, Mont.
CORBY, HARRY GILBERT, Student................1138 S. Maple St., Carthage, Mo.
CRAIG, JOHN J., Min. Engr., Northwestern Improvement Co., Brainerd, Minn.
EHLERS, LOUIS WILMER, Student..........Missouri School of Mines, Rolla, Mo.
ERDOFY, MAXWELL EMORY, Student.....Columbia University, New York, N. Y.
FERGUSON, KENNETH SEARS, Student, Colorado School of Mines, Golden, Colo.
FLOSS, FRITZ CARL, Student..................................Milwaukie, Ore.
HOFIUS, MAX T., Student..........Colorado School of Mines, Golden, Colo.
HU, SHIH-HUNG, Student.....................P. O. Box 732, Golden, Colo.
KLEIN, KARL F., Min. Engr., Morococha Mining Co., Morococha, Peru, S. Amer.
PIERCE, COLWELL ARBA, Student......3417 Virginia Ave., Kansas City, Mo.
SCHMIDT, ROBERT DROEGE, Student..........1509 Alameda Ave., Lakewood, O.
STOTZ, NORMAN I., Student...............Lehigh University, S. Bethlehem, Pa.
SUTTON, HARRY ALLEN, Senior Student, Oregon Agricultural College, Corvallis, Ore.
WIEBELT, FRANK J., Min. Engr., Laborer, Anaconda Copper Co.,
217 Colorado St., Butte, Mont.

**Total Membership, July 10, 1916........................... 5,612**

## CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that

persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

## MEMBERS

The following persons have been proposed during the period June 10, 1916 to July 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

### *Members*

**Robert Sexton Allen**, Globe, Ariz.
Proposed by Rudolf Gahl, Guy H. Ruggles, C. E. Arnold, W. B. Cramer.
Born 1889, New Haven, Conn. 1896–1906, Grade and High School, Kansas City, Mo. 1906–09, Missouri School of Mines. 1909–12, Lab. Work and Control Chem., Nev. Cons. Copper Co. 1912–13, Slag Chemist, Old Dominion Copper Co. 1913–14, Chem. and Asst. Engr., Superior & Boston Copper Co. 1914–16, Sr. Control Chem., Nev. Cons. Copper Co.
Present position: Chief Chem. of Concentrator, Inspiration Cons. Copper Co.

**Frederick Eurith Beasley**, Tacoma, Wash.
Proposed by Willard V. Morse, Roger E. Chase, Brent N. Rickard.
Born 1882, Peoria, Ill. 1891–1901, Elementary and High School, Peoria, Ill. 1901–04, Mech. Engrg., Univ. of Illinois, Urbana, Ill. 1902, Experimental Dept., Acme Harvest Machine Co., Peoria, Ill. 1903, Civil Engrg. Dept., Toledo, Peoria & Western R. R. 1904–05, Asst. Engr., Canadian Min. & Smelt. Co., Chem. Dept., Trail Smelter, Trail, B. C. 1905–16, Representative Engr., Tacoma Smelter for Trail Smelter shipments. 1910–12, Asst. Representative, Treadwell Min. Co., Tacoma, Wash. 1914, Asst. Representative, Kennicott Min. Co., Tacoma, Wash. 1912–13, Asst. to A. McCullough, Met., Tacoma, Wash.
Present position: Resident Engr., Consolidated Min. & Smelt. Co., Tacoma, Wash.

**Howard A. Beck**, Aurora, Nev.
Proposed by W. M. Dake, Jr., R. A. Hardy, F. A. Downes.
Born 1892, New York, N. Y. Grammar and High School, New York City and Denver. 1912–13, Mine Foreman, Leadville Mines Co., Lead, Mo. 1913–14, Night Foreman, St. Kevin Min. Co., Lead, Mo. 1915, Aurora Cons. Mine and Mill. 1915, Shift Foreman, Golden Boulder Co., Fairview, Nev. 1915 to date, Filterman, Tank man, Tube mill man, Aurora Cons. Mines Co.
Present position: Shift Foreman, Aurora Cons. Mines Co. Mill.

**P. McDonald Biddison**, Columbus, O.
Proposed by Samuel S. Wyer, Harry C. Reeser, Daniel M. Armstead.
Born 1884, Lane, Kan. 1904, Kansas State Agricultural College, B. Sc. in Elec. Engrg. 1904, Draftsman and Electrician, Cement Plant Const., Hunt Engrg. Co., Iola, Kansas. 1905, Sta. Engr., Iola Portland Cement Co., Iola, Kans. 1905, Electrician, Zeigler Coal Co., Zeigler, Ill. 1905, Draftsman. 1905–06, Chief Engr., Logan Nat. Gas & Fuel Co., Columbus, O. 1906, Sales Engr., Hope Engrg. & Supply Co., Joplin, Mo. 1907, Asst. Chief Engr., Kansas Nat. Gas Co., Independence, Kans. 1908, Const. Engr., Corcoran Cons. Co., Huntington, W. Va. 1909, Const. Engr., Columbia Gas & Electric Co., Kenova, W. Va. 1910, Mech. Engr., Ohio Fuel Supply Co., Homer, O. 1911, Mech. Engr., The Texas Co., Port Arthur, Tex. 1912–16, Const. Engr., The Ohio Fuel Supply Co., Columbus, O.
Present position: Const. Engr., The Ohio Fuel Supply Co.

**Paul Truman Boise,** Salt Lake City, Utah.
Proposed by Charles P. Brooks, Charles W. Stimpson, J. C. Jones.
Born 1888, Lincoln, Kans. 1904–06, Univ. of Utah. 1906–07, Michigan College of Mines. 1908, Shift Boss, Gemini Min. Co., Eureka, Utah. 1909–10, Supt., Yosemite Mines Co., Bingham, Utah. 1911–14, Shift Boss, Silver King Coalition Mines Co., Park City, Utah.
Present position: 1914 to date; Stimpson Equipment Co.

**Ralph H. Bourne,** Chicago, Ill.
Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.
Born 1881, Cleveland, O. 1905, Grad., Cornell Univ., Mech. Engrg. Member of Amer. Soc. of Mech. Engineers. 1906, Draftsman, 8 months, Wellman Seaver-Morgan Co. 1907, 8 months, Hoover & Mason, Chicago, Ill.
Present position: Vice-Pres. & Asst. Sales Mgr., Whiting Foundry Equipment Co., Harvey, Ill.

**Henri Brings,** Paris, France.
Proposed by E. Gybbon Spilsbury, J. E. Kennedy, P. De P. Ricketts.
Born 1879, Duren. 1889–97, Grad., Realgymnasium, Aix-la-Chapelle. 1898–1902, Technical High School, Aix-la-Chapelle. 1902, Grad. Min. & Met. Engr., Technical High School, Aix-la-Chapelle. 1902, Min. Engr. Ivory Coast Exploring Syndicate, Ltd., Salisbury House, London. Exploration work in French West Africa. 1903, Surveying and exploration work, French Guinea and Senegal for same company. 1904–07, Chief Engr. and Genl. Mgr., Société des Dragages Auriferes du Tinkisso, Paris, France. 1908–10, Chief Engr., Cie. Minière de l'Ouest Africain, Paris, France.
Present position: 1910 to date; Chief Engr. & Mgr., Cie. Formiere des Mines d'Or d'El Dorado, Paris, France.

**Percy Bennett Butler,** Joplin, Mo.
Proposed by A. F. Truex, G. B. Corless, C. A. Wright.
Born 1876, San Francisco, Cal. 1893, Grad., Commercial High School, San Francisco, Cal. 1913, Student, International Correspondence School, mining course. 1899, Parral Mill Co., Parral, Chih., Mex. 1900, Cia. Restauradora, Guanacevi, Mex. 1903, Montezuma Lead Co., Santa Barbara, Chih., Mex. 1905, U. S. Min. Co., Minas Nuevas, Chih., Mex. 1907, Veta Colorada Min. & Smelt. Co., Minas Nuevas, Chih., Mex. 1907–12, Operating for own account in Parral and Santa Barbara, Mex.
Present position: 1914 to date; Mgr. of mines in Joplin district.

**Robert Stuart Butler,** Joplin, Mo.
Proposed by A. F. Truex, G. B. Corless, C. A. Wright.
Born 1891, San Francisco, Cal. 1907, Berkeley High School, Berkeley, Cal. Student, International Correspondence Schools. 1910–13, Operating in Santa Barbara and Parral, Mex. 1914, Supt. of Paragon and other properties in Joplin district.
Present position: Supt.

**Allan B. Calhoun,** Globe, Ariz.
Proposed by W. B. Cramer, J. C. Low, E. M. Marshall.
Born 1879, Greenock, Scotland. Grade and high school education in public schools, Minneapolis, Minn. 1905, Grad., Minnesota School of Mines, E. M. 1906–07, Assayer, Shift boss, Foreman, Cornucopia Mines of Oregon, Cornucopia, Ore. 1908–09, and 1911, Engr., Cons. Min. & Smelt. Co. of Canada, Rossland, B. C. 1912–13, Supt., Motherlode Sheep Creek Min. Co., Sheep Creek, B. C. 1914, Supt., Arabian Cons. Mines, Kingman, Ariz. 1905, Supt., El Rayo Mine, Santa Barbara, Mex.
Present position: Efficiency Engr., Old Dominion M. & S. Co.

**Byron H. Carpenter,** Benton, Ill.
Proposed by B. Schettler, Frank A. Ray, Oscar Cartlidge.
Born 1891, Mentor, O. 1905–09, Willoughby High School, Willoughby, O. 1910–14, Ohio State Univ., Columbus, O., E. M. 1909–10, Rodman, Clark & Pike, Topographers and Land Surveyors, Willoughby, O.
Present position: Min. Engr., The Benton Coal Co.

**Allison Robert Chambers,** New Glasgow, N. S.
Proposed by R. E. Chambers, Thomas Cantley, J. B. Porter.
Born 1879, Halifax, N. S. High Schools, New Glasgow & Halifax. Dalhousie Univ., Chemistry. 1900–04, McGill Univ., B. S. 1896–99, Chaining and Transit-Dan, Nova Scotia Steel Co., Nova Scotia and Newfoundland. 1899–1900, Civ. Engr., mominion Iron & Steel Co., Wabana, Newfoundland. 1904 to date, Min. Engr.,

Nova Scotia Steel & Coal Co., Ltd., Wabana. Resident Mgr., Wabana. Asst. Mgr., Ore Mines and Quarries, New Glasgow, N. S.
Present position: Asst. Mgr. on Ore Mines and Quarries.

**Marcu F. Chase,** St. Louis, Mo.
Proposed by George C. Stone, J. H. Janeway, G. S. Brooks.
Born 1876, Alton, Ill. 1897, Trinity College, Hartford, Conn., B. S. 1898-1900, Chem., Mineral Point Zinc Co. 1900-04, Acid Expert, New Jersey Zinc Co. 1904-12, Supt. of Depue Plant, Mineral Point Zinc Co. 1912-15, Genl. Mgr., Mineral Point Zinc.
Present position: Cons. Engr.

**Gilbert Ernest Cheda,** La Libertad, Nicaragua.
Proposed by E. W. Armstrong, Charles E. Brinker, S. M. Parker, F. B. Forbes.
Born 1884, San Luis Obispo, Cal. 1911, Dept. of Geol. and Min., Stanford Univ., Cal., A. B. 1905-06, Underground, Cananea Cons. Copper Co., Cananea, Mexico. 1908, Underground, Keystone Mine, Cal. 1909, Underground, Pittsburgh Silver Peak Mine, Nev. 1911-16, Engr. and Mine Foreman, Babilonia Gold Mines, Ltd., La Libertad, Nicaragua.
Present position: Mine Foreman, Babilonia Gold Mines.

**Charles W. Clark,** San Mateo, Cal.
Proposed by Horace V. Winchell, Robert E. Tally, W. A. Clark.
Born 1871, Deer Lodge, Mont.
Present position: Genl. Mgr., United Verde Copper Co., Jerome, Ariz.

**Isaac N. Dally,** Seattle, Wash.
Proposed by F. C. Greene, Henry L. Manley, Amos Slater.
Born 1872, Webster City, Iowa. 1896-1901, Private development work on metalliferous mines in British Columbia. 1901-05, Geological and Coal Surveys, British Columbia and Alberta. 1905-07, Halls Min. & Smelt. Co., Nelson, B. C.
Present position: 1909 to date; Coal Dept., Chicago, Milwaukee & St. Paul R. R.

**W. Val De Camp,** Mayer, Ariz.
Proposed by G. M. Colvocoresses, Robert E. Tally, J. L. White.
Born 1886, Cirsco, Iowa. 1908, Colorado School of Mines, E. M. 1908, Foreman, Atlas Min. Co., Ouray, Colo. 1909-13, Supt. and Mgr., Pacific Copper Co., Crown King, Ariz. 1913, Supt., Haynes Copper Co., Jerome, Ariz. 1914, Foreman and Boss, United Verde Copper Co., Jerome, Ariz. 1915-16, Supt., Blue Bell Mine, Mayer, Ariz.
Present position: Supt., Blue Bell Mine.

**H. DeWitt Decker,** Anaconda, Mont.
Proposed by Charles D. Demond, C. R. Wraith, Enoch A. Barnard.
Born 1892, La Porte, Ind. 1914, Grad. Wisconsin School of Mines. 1915, 3 mos., Machineman, Frontier Mining Co., Benton, Wis.; 3 mos., Shaft and Machineman, Admiralty Zinc Co., Guapan, Okla. 1916, Washoe Reduction Works, Anaconda Copper Mining Co., Anaconda, Mont.
Present position: Testing Dept., Anaconda Copper Mining Co., Anaconda, Mont.

**Axel Eric,** Salt Lake City, Utah.
Proposed by R. C. Gemmell, H. C. Goodrich, D. C. Jackling.
Born 1879, Denmark. 1897, Grad., High School. 1897-1900, Machine Shop practice. 1907, Grad., Copenhagen Univ. of Polytechnic, Mech. Engr. 1906, Worked for Prof. Rung, projecting electrification of State Railroads near Copenhagen. 1907-09, Messrs. Lossen & Hjorth, London, England, steam and water softening plants. 1909, Franklin Railway Supply Co., N. Y. 1910, Midvale Steel Co. and American Bridge Co., Philadelphia. 1911, Robins Conveying Belt Co., N. Y. 1912-15, Designing and erecting Leaching Plant, Chile Exploration Co., N. Y., and Chile, S. A.
Present position: Designing Leaching Plant, Utah Copper Co.

**C. L. French,** Ely, Nev.
Proposed by J. A. Rule, A. M. Day, A. B. Young.
Born 1890, Charlotte, Mich. 1909, College Prep., Fort Collins, Colo. 1913, Colorado School of Mines, E. M. 1913, Classifier Man, Ray Cons. Copper Co., Hayden, Ariz.; Cyanide Millman and Assayer, Churchill Milling Co., Wonder, Nev. 1914-16, Cottrell Foreman, Testing Dept., Asst. Smoke Investigation Dept., Custom Ore Sampler, International Smelting Co., Tooele, Utah.
Present position: Millman, Independent Tungsten Co.

**John Creighton Garvin,** Spokane, Wash.
Proposed by L. K. Armstrong, J. McD. Porter, A. W. Paterson.
Born 1840, County Londonderry, Ireland. 1848–69, Common and Grammar Schools, 1873, Queens prize in Geology in Marghafalt National School, Dublin. 1873, Examinations Eastern States Mineral Belt, Colombia, S. Amer. 1881, 6 years gold and silver reduction works, Philadelphia. 1881–97, 10 years in Leadville; balance at Puzzler Mine, Boulder Co., in mining and metallurgical practice. 1897 to date, at Spokane in professional work.
Present position: Consulting Engineer.

**Augustus Brunner Hardie,** Philadelphia, Pa.
Proposed by A. W. Hudson, H. H. Colley, Utley Wedge.
Born 1886, Philadelphia, Pa. 1895–1903, Girard College, Philadelphia. 1903–04, Franklin Institute, Philadelphia. 1908, Univ. of Penn. 1910–12, Univ. of Wisconsin. 1903–05, Shop practice, etc., Queen & Co., Inc., Philadelphia. 1905–07, Draughtsman, Standard Steel Works, Pa. 1907–08, Draughtsman, General Electric Co., Philadelphia. 1909, Asst. Plant Engr., Stanley G. Glagg, Pa.; Die Designer, Pressed Steel Car Co., Pittsburgh. 1910–11, Const. Supt., Tennessee Copper Co., Tenn. 1912–14, Const. Supt., Braden Copper Co., Chile. 1914, Design, supervision of erection, metallurgical plants with Utley Wedge.
Present position: Supt. of erection, Sulphuric Acid Plant, for Utley Wedge at Calumet & Arizona Copper Co..

**Fredrik Jörgen Ording Hurum,** Pittsburgh, Pa.
Proposed by Stephen L. Goodale, Robert M. Black, Horatio C. Ray.
Born 1893, Christiania, Norway. 1911, Examination Artium, Christiania. 1911–14, Royal German Inst. of Tech., Aachen. 1914, Cand. rer. Met., Aachen, Germany. 1915–16, Univ. of Pittsburgh, Met. Engrg.
Present position: Met. Engr., Univ. of Pittsburgh.

**William Wallace Inglis,** Scranton, Pa.
Proposed by H. M. Warren, J. M. Humphrey, Paul Sterling.
Born 1871, Scranton, Pa. High School, Scranton, Pa. Hillside Coal & Iron Co. Pennsylvania Coal, N. Y., Susquehanna & Western Coal Co.
Present position: Genl. Mgr., Coal Min. Dept., Delaware, Lackawanna & Western R. R. Co.

**Junzo Inouye,** Kenjiho, Koshin, Korea.
Proposed by Kenroku Ide, Shinji Harada, Takeshi Kawamura.
Born 1888, Osaka, Japan. 1913, Grad., Min. & Met. Institute, Imperial Univ., Kyoto, Japan. 1913, Mitsubishi Co., Tokyo, Japan.
Present position: Met. Engr., Iron & Steel Dept., Mitsubishi Co.

**Phillips F. Jarvis,** St. Louis, Mo.
Proposed by Frederick K. Copeland, P. N. Moore, E. J. Rossbach, O. M. Bilharz, H. T. Walsh.
Born 1878, Claremont, N. H. 1897, Stephens High School, Claremont, N. H. 1897, Sullivan Machinery Co., Chicago office. Shop practice, sale of mining machinery, metal mines and quarries, and specializing considerably in coal mining. Have been connected with metal mines and was president, Columbia Zinc Co., Joplin, Mo.
Present position: Mgr., Sullivan Machinery Co.

**Edward Charles King,** Humboldt, Ariz.
Proposed by G. M. Colvocoresses, J. N. D. Gray, A. F. Bassett.
Born 1875, Rockland, Mich. 1880–90, Public School. 1893, Bryant & Stratton Business College, Buffalo, N. Y. 1905, Special course in metallurgy, Michigan College of Mines. 1890–1900, Calumet & Hecla Smelteries, Lake Linden, Mich. & Buffalo, N. Y. 1900–04, In charge of the Casting Dept. and later Night Supt. Washoe Smeltery, Mont. 1904–05, Asst. Supt., Arizona Smelt. Co. 1905–07, Supt. of Smelter, Orogrande, N. M. 1907–10, Asst. Supt. of Smelter, Cananea Cons. Copper Co., Cananea, Mex. 1910–13, Met. on staff of southern division, Amer. Smelt. & Ref. Co.
Present position: 1914 to date; Supt. of Smelters, Cons. Arizona Smelt. Co.

**Victor E. Lieb,** Elephant Butte, N. Mex.
Proposed by L. J. Charles, S. H. Worrell, William B. Phillips.
Born 1883, Brenham, Tex. 1900–02, Academic course, Polytechnic Inst. of Brooklyn. 1903–07, Min. Engrg., Univ. of Tex., worked for W. Waldo and F. A. Jones of Houston, Tex., general engineering practice. 1907–08, Supt., Las Machos

Min. Co., Tude Durango, Mex. 1908–09, Mine examination, prospecting and development of own property, Chih., Mex.; chem. and engr., Southern Min. Co. Ocampo, Chile. 1909–10, Engr., Candelaria Min. Co., San Pedro, Chih., Mex. 1910 to present, Engr. and foreman, U. S. Reclamation Service.
Present position: Foreman, U. S. Reclamation Service.

**Alexander Watts McCoy,** Ponca City, Okla.
Proposed by E. De Golyer, R. A. Conkling, Dorsey Hager.
Born 1889, Independnce, Mo. 1912, Univ. of Missouri, C. E., 1914, M. A. 1912–13, Engrg. Experiment Station, Univ. of Missouri. 1914–16, Instructor, Geol., Univ. of Oklahoma.
Present position: Geol., Morland Oil Company.

**Richard Southgate McIntyre,** Namtu, North Shan States, Burma.
Proposed by Andrew C. Lawson, Walter S. Weeks, T. A. Rickard.
Born 1895, Pittsburgh, Pa. 1910–12, Preparatory work, College of the City of New York, New York, N. Y. 1912–16, Univ. of California, B. S. 1915–16, Asst. Statistician, U. S. Geological Survey, San Francisco.
Present position: Asst. Min. Engr., Bowdwin Mine, Burma. Corpn., Ltd., Namtu, North Shan States, Burma.

**Charles Mayott,** Aurora, Nev.
Proposed by W. M. Dake, Jr., R. A. Hardy, F. A. Downes.
Born 1891, Salt Lake City, Utah. Grade Schools, etc., Salt Lake City. 1908–09, Mucker, Red Top Mine, Goldfield, Nev. 1909–11, Machine man, Timber man, Red Top Mine. 1911–13, Engrg. Dept., Goldfield Cons. Mines Co., Goldfield, Nev. 1913–14, Surface Boss, Aurora Cons. Mines Co., Aurora, Nev. 1914–15, Shift Boss. Aurora Cons. Mines Co., Aurora, Nev.
Present position: 1915 to date; Mine Supt., Aurora Cons. Mines Co.

**Whitney Player Mee,** Santa Rita, N. Mex.
Proposed by J. M. Sully, Horace Moses, Frank R. Wicks.
Born 1891, Salt Lake City, Utah. 1900–09, Public Schools, El Paso, Tex. 1909–13 Univ. of Cal., Berkeley, Cal., B. S. 1911, summer, Surveying, Mesilla Valley, U. S. R. S. 1912, summer, Mining, Copper Queen Mines, Bisbee, Ariz. 1913–14, Efficiency Engr., Oakland Lamp Works, of General Electric Co. 1914, Chem. and Asst. Engr., Shannon Copper Co., Gleeson, Ariz. 1914, Solution man, Simonds and Latham Cyanide Plant, Melones, Cal. 1914–15, Assayer, A. S. & R. Co., smelter, Asarco, Dgo., Mex. 1915, Car checker and jigger boss, Ray Copper Co., Ray, Ariz. 1915, Assayer, Asarco, Dgo., Mex.
Present position: 1915 to date; Chem., Chino Copper Co.

**William Bruce Montgomery, Jr.,** Cleveland, O.
Proposed by J. A. Ede, Thomas T. Read, William H. Shearman.
Born 1880, Lynchburg, Va. 1900, Virginia Military Institute, Lexington, Va., B. S. 1909, School of Mines, Columbia Univ., E. M. 1901–06, Engr., Arlington C. & C. Co., Arlington, W. Va., and Gilliam C. & C. Co., Gilliam, W. Va. Shawnee C. & C. Co. and Glen Alum Coal Co., Glen Alum, W. Va. 1909–15, Engr., examination and report work, The Grasselli Chemical Co., of Tenn., New Market, Tenn. Mgr. last year of operation. Since 1915, The Grasselli Chemical Co., Cleveland, O. Principally examination and report work, on zinc propositions.
Present position: Min. Engr., The Grasselli Chemical Co.

**Edgar L. Newhouse, Jr.,** Salt Lake City, Utah.
Proposed by P. A. Mosman, C. W. Whitley, C. W. Stimpson.
Born 1890, Moncton, N. B., Canada. 1906–11, Columbia Univ., Min. & Met. 1912–14, Chem., and Assayer, East Helena Plant, American Smelt. & Ref. Co., Montana. 1914–16, Asst. Met., Garfield Smelt. Co., Utah.
Present position: Mgr., Garfield Chemical & Mfg. Corpn.

**Chase S. Osborn,** Sault Ste. Marie, Mich.
Proposed by O. C. Davidson, C. H. Baxter, George J. Eisele.
Born 1860, Huntington Co., Ind. LL.D. from several universities. Former Governor of Michigan, etc. 1883–1916, Chairman, Engineering Committee Board of Regents, Univ. of Mich. Field work in Lake Superior districts of Michigan, Wisconsin, Minnesota, Ontario, in Russia, Germany, Sweden, Lapland, Africa, Asia, Madagascar and South America, Cuba, China, etc., etc.
Present position: Explorer and Prospector.

**Dale L. Pitt,** Tacoma, Wash.
Proposed by Willard V. Morse, Brent N. Rickard, Roger E. Chase.
Born 1884, Salt Lake City, Utah. 1899–1903, Salt Lake High School. 1903–07, Utah School of Mines, B. S. 1907–09, Asst., J. W. Wade, M. E., Salt Lake City. 1909–12, Field Supt., B. F. Tibby Co., Salt Lake City. 1912–15, Chief Engr., Federal Lead Co., Flat River, Mo. 1915–16, Min. Eng., Salt Lake City.
Present position: Min. Eng., Tacoma Smelting Co.

**Rastus S. Ransom, Jr.,** Golden, Colo.
Proposed by Charles F. Rand, Harry J. Wolf, W. G. Haldane.
Born 1890, New York, N. Y. 1907–09, Columbia Univ., School of Applied Science. 1910–13, Colorado School of Mines, E. M. 1913–14, Assayer, Bingham-New Haven Copper and Gold Min. Co., Bingham, Utah. 1914, Leasing and working underground, Central City and Idaho Springs Min. Districts, Colo.
Present position: Min. Engr., Golden, Colo.

**Charles Henderson Rich,** Conshohocken, Pa.
Proposed by John W. Logan, R. M. Bird, R. J. Wysor.
Born 1868, Yardley, Pa. Common school followed by technical training and special study. 1893–98, Analyst, Chemical Laboratory, Duquesne Works, Carnegie Steel Co. 1898–1901, Chem., Aliquippa Steel Co. 1901–02, Asst. Chem., Duquesne Works, Carnegie Steel Co. 1902–12, Chief Chem., Clairton Works, Carnegie Steel Co.
Present position: 1912 to date; Met., Alan Wood Iron & Steel Co.

**Herman F. Schlundt,** St. Francois, Mo.
Proposed by Waldo H. Comins, Philip H. Pipkin, James A. Caselton.
Born 1878, Rahway, N. J. 1899–13, Stevens Institute of Technology. 1903–04; 1905–06, Union Pacific R. R. 1906–07, Rock Island Lines. 1907–09, Taylor Gold Min. Co. 1909–11, Northwestern States Portland Cement Co. 1911–12, Cia. Metallurgica de Torreon. 1912–13, Amer. Smelt. & Ref. Co.
Present position: 1913 to date; St. Louis Smelt. & Ref. Co.

**Ellwood Spencer Smith,** Humboldt, Ariz.
Proposed by G. M. Colvocoresses, J. N. D. Gray, A. F. Bassett.
Born 1890, Brooklyn, N. Y. 1895–1902, Grade Schools, New York, N. Y. 1902–06, High School, New York, N. Y. 1906–10, U. S. Naval Academy, Annapolis. 1911, Post Graduate School, Univ. of Illinois. 1912, Clerk, International State Bank, Trinidad, Colo. 1914, Millman, Cons. Arizona Smelt. Co. 1915, Engrg. Staff, Cons. Arizona Smelt. Co.
Present position: Asst. to Genl. Mgr., Cons. Arizona Smelt. Co.

**Frank J. Staral, Jr.,** Cleveland, O.
Proposed by S. H. Pitkin, Franklin Moeller, Joseph Struthers.
Born 1878, Akron, O. Akron and Cleveland Grammar Schools. 1899, Grad., Cleveland Central High School. 1902–05, Case School of Applied Science. 1904, Grad., Columbia School of Mines, E. M. 1904–06, Surveying Engr., The Boston and Montana Cons. Copper and Silver Min. Co., Butte, Mont., also experimental work at Great Falls Smelter. 1906–07, Draftsman and Surveying Engr., Cia Real del Monte y Pachuca, Pachuca, Hdg., Mex. 1907–13, Cashier, Min. Engr. and Supt., American Smelt. & Ref. Co., and allied interests in Mexico. 1913–14, Examination work and Mgr., The Assets Gold Min. Co., Jefferson City, Mont. 1914–16, Gaylord W. Feogo Co., Cleveland, O.
Present position: Supt., The Investors Co.

**William Carl Strohbach,** Hayden, Ariz.
Proposed by David D. Moffat, William T. MacDonald, David M. Kay.
Born 1891, San Francisco, Cal. 1896–1904, Public School, Oakland, Cal. 1904–08, Polytechnic High School, Oakland, Cal. 1909–14, Univ. of Cal., Berkeley, Cal., B. S. 1910–14, Summers, underground and cyanide plant, North Star Mine and Empire Mine, Grass Valley, Cal. 1915, Attendant at Bureau of Mines Exhibit, P. P. I. E., San Francisco, Cal.
Present position: 1915 to date; Chemist, Ray Cons. Copper Co.

**Clarke Sullivan,** Oakland, Cal.
Proposed by H. M. Alley, Robert G. Davies, E. E. Carpenter.
Born 1882, Atlanta, Ga. 1902, Oakland High School. 1907, Univ. of Cal., B. S. 1907–11 Assayer and Mill Supt., Progreso Min. Co. 1911–14, Mill Supt. and

Genl. Supt., Zopitoto Min. Co. 1914–15, Const. Supt., Mill work, Arrondo Min. Co. 1915. Mill work, Churchill Mill. Co.,

Present position: Mill Supt., Eden Min. Co., Nicaragua, C. A.

**E. L. Talbot,** Salt Lake City, Utah.

Proposed by Alfred Frank, George H. Dern, Ernest Gayford.

Born 1862, Central City, Colo. Public grade school also High School. Daly West Min. Co., Park City, Utah. Ontario Silver Min. Co., Park City, Utah.

Present position: Genl. Supt., Daly West Min. Co., also Ontario Silver Min. Co.

**Floyd Alpine Warren,** Sr., Collinsville, Ill.

Proposed by Jesse O. Betterton, A. T. March, W. E. Newnam.

Born 1886, Hiawatha, Kans. 1904, Academy, Hiawatha, Kans. 1910, Highland Park College, Des Moines, Ia. 1910–15, Shift Boss, Foreman, Supt., Pattinson Plant Dept., Experimental work and Assaying, American Smelt. & Ref. Co., Omaha, Nebr.

Present position: Supt., Blast Furnace, Roasting Depts. and Newnam Hearths, St. Louis Smelt. & Ref. Co.

**Sidney E. Werthan,** Bisbee, Ariz.

Proposed by Gerald Sherman, Arthur Notman, H. C. Henrie.

Born 1891, Denver, Colo. 1910, Public Schools and High School, Denver, Colo. 1914, Univ. of Denver, B. A.; 1915, M. A. 1913–14, Instr., Dept. of Chem., Univ. of Denver. 1915, Research work on Flour, Great Western Mill. & Granary Co. 1915, Mine Laboratory, Copper Queen Cons. Min. Co.

Present position: Chemist, Copper Queen Cons. Min. Co.

**George Hazzard West,** Globe, Ariz.

Proposed by William B. Cramer, L. O. Howard, Walter Harris.

Born 1882, Philadelphia, Pa. 1896–1900, Central High School, Philadelphia, Pa., B. A. 1900–04, Univ. of Penn., Philadelphia, Pa., B. S. 1904–06, Assayer, Rincon Mines Co., Congress Junction, Ariz. 1906–07, Chem., Cons. Arizona Smelt. Co., Humboldt, Ariz. 1907–11, Chem. and Engr., Orogrande Smelt. Co., Orogrande, N. M. 1911 to date; Chem. and Smelter Foreman, Old Dominion Copper Min. & Smelt. Co., Globe Ariz.

Present position: Smelter Foreman, Old Dominion Copper Min. & Smelt. Co.

**Henry Palmer Westcott,** Pittsburgh, Pa.

Proposed by R. B. Woodworth, Thomas T. Read, William H. Shearman.

Born 1878, Seneca Falls, N. Y. High School at Seneca Falls and Rochester, N. Y. Five years, Alden Bateria National Gas Co., Alden, N. Y. 1908 to date; Metric Metals Works, Erie, Pa. Author, Hand Book of Natural Gas; Measurement of Gases Where Density Changes. Now in press, Hand Book of Casinghead Gas.

Present position: Engr., National Gas Construction.

**Garratt S. Wilkin,** Moscow, Utah.

Proposed by Edward R. Zalinski, Earl R. Pembrooke, C. W. Stimpson, W. A. Wilson, Ernest Gayford.

Born 1882, New Philadelphia, Pa. 1895–99, New Philadelphia High School. 1900–02, Western Reserve Univ. 1904, Miner, Katherine Mine, Stockton, Utah. 1905, Foot nipper, Bingham and New Haven Min. Co., Bingham, Utah. 1905–07, Sampler and Foreman, Boston Cons., Bingham, Utah. 1907–09, Supt., Utah Copper Co., Bingham, Utah. 1909–11, Supt., Wild Bill Min. Co., Milford, Utah.

Present position: 1911 to date; Supt. and Asst. Mgr., Moscow Min. & Mill. Co.

**Charles F. Willis,** Tucson, Ariz.

Proposed by Arthur Notman, Roger T. Pelton, Gerald F. G. Sherman.

Born 1885, Boston, Mass. 1902–06, Mass. Inst. of Tech., B. S. 1906–09, Assayer to Supt., Enterprise Min. Co., Cooney, N. M. 1909–10, Mgr., Pelican Min. & Mill. Co., Lake City, Colo. 1910–11, Mgr., Granite State Mica Co., Alstead, N. H. 1911, Prof. of Min. Engrg. and Director, Arizona State Bureau of Mines, Univ. of Arizona, Tucson, Ariz.

Present position: Director, Arizona State Bureau of Mines.

**Harold Edwin Willson,** Urbana, Ill.

Proposed by H. H. Stoek, C. M. Young, C. E. Krebs.

Born 1881, Lykens, Pa. 1900, Grad., Baltimore City College. 1916, Univ. of Ill., B. S. 1900–01, U. S. Fidelity and Guaranty Co., Baltimore, Md. 1901–13, Chainman, rodman, levellor, transitman, office man and chief engineer, McKell Coal

& Coke Co., Glen Jean, W. Va. 1904–13, Engr., McKell Heirs. 1905–13, Engr., Nichol Colliery Co. 1904–13, Engr., Laura Min. Co.
Present position: Student.

**Frank A. Woodward,** Cooper Hill, Ariz.
Proposed by B. Britton Gottsberger, P. G. Beckett, C. E. Mills.
Born 1852, Worcester, Mass. General education only. 1906 to date, Iron Cap Copper Co.
Present position: Supt., Iron Cap Copper Co.

**Herbert Watson Woodward,** Copper Hill, Ariz.
Proposed by W. B. Cramer, C. Mendelsohn, Walter Harris.
Born 1883, So. Framingham, Mass. 1900, Superior, Wis., Grammar and High Schools. 1900–02, High school, Somerville, Mass. 1903–06, Tufts College. 1906–08, Surveying mining claims. 1908, Assayer, Mine Surveyor, Copper Basin Min. Co. (defunct), Nev. 1908, Cyanide Mill, North Star Mine, Grass Valley, Cal. 1909–16, Engr., Iron Cap Copper Co., Copper Hill, Ariz.
Present position: Asst. Supt., Iron Cap Copper Co.

**Yeh-Tze Yen,** Hanyang, China.
Proposed by Chin Tao Huang, Cho Yang, C. F. Wang.
Born 1886, Kiangsu, China. 1901–03, St. John's College, Shanghai. 1904–07, Kyoto High School, Japan. 1907–10, Min. course, Kyoto Univ., Japan. 1911–12, Engr., Pinchuho Iron & Coal Co.
Present position: 1912 to date; Engr., Blast Furnace Dept., Hanyang Iron & Steel Works.

**Stuart M. Young,** Bingham, Utah.
Proposed by A. M. Day, J. A. Rule, A. B. Young.
Born 1890, Salt Lake City, Utah. 1896–03, Salt Lake Public schools. 1903–07, Salt Lake High School. 1907–11, Univ. of Utah, B. S. 1911–12, Chem., Grasselli Chemical Co., Park City, Utah.
Present position: 1912 to date; Assayer, Min. Dept., United States Smelt. Co.

## *Associate Members*

**William Wright Armstrong,** Salt Lake City, Utah.
Proposed by C. W. Stimpson, Ernest Gayford, Duncan MacVichie.
Born 1865, Darlington, Wis. 1885, Baker Univ., Baldwin, Kans., B. S. 1887, Univ. of Wisconsin, Madison, Wis., LL. B.
Present position: Pres., National Copper Bank.

**Henry Ayman Bomberger,** Philadelphia. Pa.
Proposed by Arthur J. Hoskin, F. W. Smith, J. W. Finch.
Born 1866, Philadelphia, Pa. 1884, Ursinus College, Pa., A. B.; 1886, A. M. 1900, Vice-Prest., Temple Univ., Philadelphia. 1905, Prest. & Genl. Mgr., Southwestern Dev. Co. 1912, Genl. Mgr., Minnesota-Connor Min. & Mill, Co. **Have had no special technical training in the schools as a mining engineer. Have been a student all my life and for nearly fifteen years have made a special study of mining in all its branches, about one-third of which time has been spent in practical field work associated with some of the leading mining engineers and mine managers of the country, none of whom will know that I have ever made any claims at all as an engineer for I have never made any such claims. I personally organized the three companies above mentioned, and my services for mine reports, etc., are constantly sought without solicitation on my part.**
Present position: Prest. & Genl. Mgr., Mohave Minerals Co.

**William Paul Gage,** Fort Worth, Tex.
Proposed by Samuel S. Wyer, Harry C. Reeser, David T. Day.
Born 1875, Denver, Colo. 1894, Grad., Riverside (Cal.) High School, scientific course. 1895–96, Finished Sophomore year at Marietta (Ohio) College; Latin, scientific course. 1895, summer, and from spring, 1896 to spring, 1903, Eureka Pipe Line Co. and Buckeye Pipe Line Co. in W. Va. and Ohio. Worked in all the pipe line departments on construction of lines and stations, office and field work of all kinds. 1903–09, Security Oil Co., Beaumont, Tex.; Inventory Clerk; Chief Dispatcher; Asst. to Genl. Supt., Pipe Line Dept. 1909–16, Vice-Prest. and Genl. Supt., Lone Star Gas Co., Fort Worth, Tex., active executive officer. Practical management of construction and operation almost 500 miles pipe line, supplying about

30 towns in North Texas in addition to development of the gas field, installation of compressor station, etc.
Present position: First Vice-Prest. and Genl. Supt., Lone Star Gas Co.

**Charles A. Gillette**, Salt Lake City, Utah.
Proposed by C. W. Stimpson, Ernest Gayford, Walker J. Boudwin.
Born 1869, Sioux City, Iowa. 1891, B. S., Washington & Jefferson. 1893, Law, University of Iowa.
Present position: Attorney at Law.

**William Joseph Quigly**, El Paso, Texas.
Proposed by William D. Gordon, William S. Noyes, W. N. Small.
Born 1872, Carbondale, Pa. 1896–1900, Ibex Mining Co., Leadville, Colo. 1900–02, Penn M. & L. Co., Leadville, Colo. 1902–06, Chihuahua Min. Co. 1906–12, Genl. Agent, Chihuahua Min. Co. and El Potosi Min. Co., Chihuahua, Mex.
Present position: 1912 to date; Genl. Mgr., El Potosi Min. Co., and Chihuahua Min. Co. and Director of both companies.

**Henry Denison Randall**, Salt Lake City, Utah.
Proposed by C. W. Whitley, Ernest Gayford, C. W. Stimpson.
Born 1881, Ledyard, Conn. 1902, Amherst College, B. S. 1902–03, Lawyer, Mann Electric Co. 1903–08, Westinghouse Electric Mfg. Co. 1908–12, Allis-Chalmers Mfg. Co.
Present position: 1912 to date, Mgr., General Electric Co.

**James B. Walker, Jr.**, Salt Lake City, Utah.
Proposed by C. W. Stimpson, O. J. Salisbury, Ernest Gayford.
Born 1889, Helena, Mont. 1911, Cornell Univ., B. L., Mech. Engrg.
Present position: Mgr., Kelly Filter Press Co.

## CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period June 10, 1916 to July 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 115, February to July, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of July 10, 1916.

ALDERSON, VICTOR C., Cons. Engr., Winnemucca Mountain Min. Co., Winnemucca, Nev.
ALLEN, ARTHUR P. .......................... 233 College Ave., Houghton, Mich.
ANDERSON, A. J. ......... Care Moose Mountain, Ltd., Sellwood, Ont., Canada.
ANDERSON, RAY B. .......................... 11 Cliff St., New York, N. Y.
BABB, P. A., Care Blaisdell Coscotitlan Syndicate, Apartado No. 92, Pachuca, Hidalgo, Mexico.
BARD, D. C. .......................... 660 Stuart Bldg., Seattle, Wash.
BASSETT, THOMAS E. .......................... 59 Auburn Ave., Pontiac, Mich.
BATCHELLER, J. H., Care Virginia Lead & Zinc Corpn., Jone's Store P. O., Spottsylvania Co., Va.
BENNETT, A. F. .......................... 2 Rector St., New York, N. Y.
BINFORD, C. M. .......................... J. B. B. Coal Co., Twin Branch, W. Va.
BRADT, HARLAN H. .......................... 709 Alworth Bldg., Duluth, Minn.
BRANDES, JUAN FELIX ...... 139 Whittell Bldg., 166 Geary St., San Francisco, Cal.
BRAYTON, COREY C., Min. Engr .............. Hobart Bldg., San Francisco, Cal.
BRENNEMAN, F. G. .......................... 105 N. George St., Pottsville, Pa.
BRENNON, J. C. .................... Room 868, 11 Broadway, New York, N. Y.
BROOKE, LIONEL, .... Care Morococha Mining Co., Morococha, Peru, S. Amer.
BRUNTON, FREDERIC K. .................... Garfield Smelter, Garfield, Utah.
BRYAN, J. K. ....... Care Electrolytic Zinc Co., Colgate P. O., Baltimore, Md.
BURCH, H. KENYON, Care The Sierra Madre Club, L. A. Investment Bldg., Los Angeles, Cal.
CALLAWAY, L. A. .......................... 512 Pine St., Anaconda, Mont.
CAMPBELL, W. C. .................... 705 South Main St., Salt Lake City, Utah.
CARSON, A. C. .......................... 800 Bush St., San Francisco, Cal.
CLAGHORN, D. M. .......................... Care Morning Hotel, Mullan, Idaho.
COOK, PAUL R. .......................... Box 655, Guayaquil, Ecuador, S. Amer.

COONER, JOHN D. .......................... 417 Wyoming Ave., Scranton, Pa.
CORNELISSEN, JOHANNES .......................... Santa Marta, Colombia, S. Amer.
COXE, EDWARD H., Care United Coal Corpn., First National Bank Bldg., Pittsburgh, Pa.
CULLUM, J. BARLOW .......................... Elaridge, Leetsdale, Allegheny Co., Pa.
DEWEY, F. P. .......................... 832 Varnum St., N. W., Washington, D. C.
DOWD, JAMES J. .......................... Hotel Dee, Houghton, Mich.
DRUMMOND, T. R. .......................... Hongkong and Shanghai Bank, Manila, P. I.
EARLE, THEODORE .......................... 165 Broadway, New York, N. Y.
ELLIS, ERNEST W. ....... Care Northern Ore Co., Edwards, St. Lawrence Co., N. Y.
ENZIAN, CHARLES, Philadelphia and Reading Coal & Iron Co., Pottsville, Pa.
Estabrook, E. L. Geol. .......................... Wisconsin Zinc Co., Platteville, Wis.
DE FARIA, CICERO COELHO, Engr. Fiscal das Estradas, Inspectoria Federal das Estradas, Rua Conde do Bomfim, No. 46, Rio de Janeiro, Brazil, S. Amer.
FEARING, F. C., Cons. Engr., Kelly, Cooke & Co., Drexel Bldg., Philadelphia, Pa.
FEARN, PERCY LER .......................... 52 William St., New York, N. Y.
FELDENHEIMER, ROY .......................... Moberly, Mo.
FISCHER, SIEGFRIED, JR. .......................... Care Mrs. C. E. Parfet, Golden, Colo.
FISHER, THOMAS E. .......................... 112 East 17th St., New York, N. Y.
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GOLD, CHARLES B., Chemist and Assayer, Bartlesville Zinc Co., Collinsville, Okla.
GORDON, ROBERT, Mine Supt., Abangarez Gold Fields, Abangarez, Costa Rica, C. A.
GORMLY, S. J., Min. Engr. .......................... Guayacan, Chile, S. Amer.
GRIEVE, RICHARD P. .......... Narcozari, Son., Mexico, via Douglas, Ariz.
GRUNOW, WILLIAM R., S'g't. Co. "M," Arizona 1st Infantry, Camp Harry J. Jones, Douglas, Ariz.
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LADOO, RAYMOND B. .......................... The Low Moor Iron Co., Low Moor, Va.
LAIRD, GEORGE A. .......................... North Yakima, Wash.
LAUDIG, O. O., Mgr. .......................... Standard Iron Co., Deseronto, Ont., Canada.
LAWTON, N. O. .......................... Lawton, Mich.
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LOERPABEL, W. HARRISON .......................... Care Carlota Mine, Cumanyaqua, Cuba.
LUNN, ROBERT, JR. .......................... Jerome, Ariz.

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SMITH, FRANK G. D .......... U. S. Metals Ref. Co., East Chicago, Ind.
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STAVER, W. H .......... 505 Krise Bldg., Lynchburg, Va.
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TSAI, HSIANG .......... Care Oliver & Co., Hankow, China.
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WILLIAMS, RALPH B .......... Room 2202, 120 Broadway, New York, N. Y.
YATSEVITCH, M. G., Care Prof. N. T. Stepanoff, Chemical Lab., Min. Institute, Vasilieff Ostrov. 21 Line, Petrograd, Russia.

## MEMBERS' ADDRESSES WANTED

| Name. | Last Address of Record from which Mail has been Returned. |
|---|---|
| BARNES, BLAKESLEE | Arrow Engineering Co., Palmyra, Mo. |
| BELL, D. A. S | 136 McLaren St., Ottawa, Canada. |

BLOW, J. J. .................................172 Rodney St., Brooklyn, N. Y.
BOOTH, E. L. ..........................349 W. 145th St., New York, N. Y.
BOYS, H. R. ............................Aurora Cons. Min. Co., Aurora, Nev.
CAIRNS, JOHN M. .......Henry S. King & Co., 65 Cornhill, London, E. C., England.
COLE, ROBERT J. .........McKay Apartments, 7th & Pike Sts., Seattle, Wash.
CRARY, CHARLES N. ..............................................Kimberly, Nev.
DRAPER, CARL H. .......................Apartado 77, Guadalajara, Jal., Mexico.
FOSTER, GEORGE C. ...................18 Cadogan Blk., Calgary, Alta., Canada.
GOEDICKE, CARL ...............................Box 535, San Antonio, Tex.
GORDON-FIREBRACE, W. E. ....812 Salisbury House, London, E. C., England.
GRANT, ULYSSES S., IV ...................Westmorly Court, Cambridge, Mass.
GUNTHER, C. GODFREY ..........................................Stratford, Conn.
HANLON, JOHN EDWARD ...........................Timmins, Ont., Canada.
HOBART, EDMUND NORRIS ......................................Clifton, Ariz.
JONES, THOMAS J., Mine Mgr., Kyshtim Min. Wks., Perm Govt.,
via Petrograd, Russia.
KELLOGG, R. M. .................................Wellington, Lyon Co., Nev.
KISHMAN, MAURICE W. .................101 Masonic Ave., Cripple Creek, Colo.
LENOIR, FRANK H. ...............................................Douglas, Alaska.
MCGEE, JOHN ....................United Greenwater Co., Dale via Amboy, Cal.
MAINWARING, H. M. C., Mine Mgr., Chillagoe, Ltd., Chillagoe, Queensland, Australia.
MILLER, FRANK BARTON .........................Blair, Esmeralda Co., Nev.
MOHRMAN, E. M. ..........................1293 W. 111th St., Cleveland, O.
MOORE, REDICK R. ...............................................Zortman, Mont.
O'BRIEN, P. C. K. ..............Hotel Essex, 684 Larkin St., San Francisco, Cal.
PARRISH, S. F. ....................................Battle Mountain, Nev.
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RALPH, E. W. ...........................Boston Ely Min. Co., Kimberly, Nev.
REDFEARN, A. M. ..........................................Hold publications.
REVELL, G. E. ..........................Box 132, Nelson, B. C., Canada.
REYNOLDS, L. ..........................Apartado 25, Guauajuato, Mex.
RHODES, W. B. ...............................................Golden, Colo.
RODRIQUEZ, J. C. ......................Apartado, 87 Saltillo, Coah., Mexico.
ROGERS, B. C., Care Standard Mine, Detroit Copper Min. Co., Box 427,
Metcalf, Ariz.
SALES, A. J. ........................Giroux Cons. Mines Co., Kimberly, Nev.
SANDIFER, H. C. ......................Box 15, Bis., Mexico City, Mexico.
SMITH, A. H. .................................Gateway, B. C., Canada.
SPARKS, J. T. ...............................326 Erie St., El Paso, Tex.
STODDART, A. W. ..................638 Salisbury House, London, E. C., England.
SULLIVAN, W. P. ......................The Henry Walke Co., Norfolk, Va.
TAYLOR, A. W. ............Korean Exploration Co., Chiksan Mines, Chiksan, Korea.
THOMAS, EDMUND ..........................Weaver Mine, Gibson, N. Mex.
TOD, GRANT H. ..................Care Coast Mfg. & Supply Co., Livermore, Cal.
TONG, S. K. ..............................413 W. 115th St., New York, N. Y.
VAN NESS, W. W. ..................622 Salisbury House, London, E. C., England.
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WARD, HARRY J. ...........................948 Market St., San Francisco, Cal.
WENTWORTH, I. H. ......245 Beldon Ave., Harlandale Addition, San Antonio, Tex.
WRAIGHT, E. C. ....63 Wavertree Road, Streatham Hill, London, S. W., England.
YEANDLE, W. H., JR., Care Guilermo Brockmann,
2a Capuchines, No. 55, Mexico City, Mex.

## NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period June 10, 1916 to July 10, 1916.

| Date of Election. | Name. | Date of Decease. |
|---|---|---|
| 1888 | *Gatzmer, William C. | May 2, 1916. |

* Member.

## EXECUTIVE COMMITTEES OF LOCAL SECTIONS

### *New York*

Meets first Wednesday after first Tuesday of each month.
DAVID H. BROWNE, *Chairman.* PERCY E. BARBOUR, *Vice-Chairman.*
A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer.*

### *Boston*

Meets first Monday of each winter month.
W. E. C. EUSTIS, *Chairman.* R. L. AGASSIZ, *Vice-Chairman.*
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
ALBERT SAUVEUR, H. L. SMYTH.

### *Columbia*

Holds four sessions during year. Annual meeting in September or October.
STANLY A. EASTON, *Chairman.* FREDERIC KEFFER, *Vice-Chairman.*
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
D. C. LIVINGSTON, FRANK A. ROSS.

### *Puget Sound*

Meets second Saturday of each month.
GLENVILLE A. COLLINS, *Chairman.* H. L. MANLEY, *Vice-Chairman.*
AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
I. F. LAUCKS, JOHN N. POTT.

### *Southern California*

C. COLCOCK JONES, *Chairman.* ALVIN B. CARPENTER, *Vice-Chairman.*
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Bldg., Los Angeles, Cal.
A. B. W. HODGES, R. A. PEREZ,
E. A. MONTGOMERY, WILLIAM F. STAUNTON.

### *Colorado*

L. P. HAMMOND, *Chairman.* F. H. BOSTWICK, *Vice-Chairman.*
P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
G. A. KENNEDY. M. S. MacCARTHY.

### *Montana*

J. L. BRUCE, *Chairman.* W. C. SIDERFIN, *Vice-Chairman.*
M. H. GIDEL, *Secretary-Treasurer*, 1102 W. Galena St., Butte, Mont.
W. T. BURNS, N. B. BRALY.

### *San Francisco*

Meets second Tuesday of each month.
T. A. RICKARD, *Chairman.* W. H. SHOCKLEY, *Vice-Chairman.*
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
E. A. HERSAM, H. W. YOUNG.

### *Pennsylvania Anthracite*

R. V. NORRIS, *Chairman.*
CHARLES F. HUBER, *Vice-Chairman.* EDWIN LUDLOW, *Vice-Chairman.*
W. J. RICHARDS, *Vice-Chairman.* ARTHUR H. STORRS, *Vice-Chairman.*
PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP,
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

### *St. Louis*

C. J. ADAMI. *Chairman.* HERMAN GARLICH, *Vice-Chairman.*
F. W. DeWOLF, *Vice-Chairman.* M. M. VALERIUS, *Vice-Chairman.*
WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
C. R. FORBES, F. D. RASH,

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HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
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GEORGE P. HULST, FREDERICK T. SNYDER.

### *Utah*

C. W. WHITLEY, *Chairman.* WALTER FITCH, *Vice-Chairman.*
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
E. R. ZALINSKI, WILLIAM WRAITH.

### *Arizona*

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NORMAN CARMICHAEL, *1st Vice-Chair.* B. BRITTON GOTTSBERGER, *2nd Vice-Chair.*
ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
W. L. CLARK, J. C. GREENWAY.
W. G. McBRIDE. FOREST RUTHERFORD.

### *Nevada*

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HENRY M. RIVES, *Secretary-Treasurer*, Reno, Nevada.
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EMMET D. BOYLE, JOHN G. KIRCHEN,
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CHARLES F. RAND.

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FRANCIS A. THOMSON,
F. W. TRAPHAGEN,
CLINTON M. YOUNG.

[1] Until Feb., 1917. [2] Until Feb., 1918. [3] Until Feb., 1919. [4] Until Feb., 1920.

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PHILIP W. HENRY, *First Vice-Chairman.*

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WILLIAM H. SHEARMAN, *Secretary*, 55 Wall St., New York, N. Y.

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29 West 39th Street, New York, N. Y.

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I agree, if elected, to accept election within four months, and to abide by the Constitution and By-Laws

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Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 117 SEPTEMBER 1916

Published Monthly by the American Institute of Mining Engineers at 212–218 York St., York, Pa., H. A. Wilotsky, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., Bradley Stoughton, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), $10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, $5 per annum. Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

## MANUSCRIPT CLOSING DATE FOR THE NEW YORK MEETING, 1917

The 114th (New York) Meeting of the Institute will be held in the third week of February, 1917. The Committee on Papers and Publications has set Dec. 1, 1916, as the closing date for the receipt of manuscripts intended for this meeting.

Manuscripts must be received by the Secretary on or before this date. If they are sent through members of the technical committee they must be sent to those committees enough in advance of the time to be forwarded and in the hands of the Secretary by the time specified.

While Dec. 1, 1916, is the final closing date, it is a great advantage to get papers in as much before that time as possible as it enables them to be submitted in proof form to their authors. Papers received on or before Nov. 1, 1916, will, as far as possible, be published in the January Bulletin or earlier, thus giving good distribution among the members in advance of the meeting.

The Institute's records show that the discussion of papers is very much better when they are distributed somewhat in advance of the meeting instead of in the Bulletin which goes out immediately preceding the meeting.

// BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 116 AUGUST 1916

Published Monthly by the American Institute of Mining Engineers at 212–218 York St., York, Pa., H. A. Wisotzkey, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., Bradley Stoughton, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), $10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, $5 per annum. Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

## TRANSACTIONS FOR 1916

Vol. 51 has been sent to all members who have paid their 1916 dues. Vols. 52 and 53, the last two volumes for 1916, will be shipped about the middle of August to all except those owing current dues. Members who do not receive them within a reasonable time are asked to notify the Secretary.

Back volumes to complete sets may be purchased from the Institute at reduced rates.

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**Arizona Meeting, Sept. 18 to 25, 1916.**

## ARIZONA MEETING, SEPTEMBER 18 to 25, 1916

For the first time in its history the Institute will hold a meeting in the ate of Arizona.

A few years ago Arizona stood third in the copper-producing districts the United States. Since that time, with the development of porphyry nes, the output has gone up with great rapidity until it not only is the ding district, but its output at the present time is at the rate of nearly uble the Montana output, which now stands second in the list.

In connection with this development, most interesting departures ve been made in methods of mining, concentration and smelting. inciples of mining and metallurgy of the greatest interest will be cov- d by papers which will appear in the *Bulletin* and be presented at chnical Sessions, while ample opportunity will be afforded for those o go to the meeting to see all the points of interest to them. In addi- n, members will be well rewarded by the marvelous natural features be seen on the trip, including the petrified forests, the cliff dwellings the Apache Trail and the Grand Canyon of Arizona.

It is planned to start the special train from New York, but members d guests may join the special at other points en route as indicated by e following schedule:

### ITINERARY

**Thursday, September 14th**

| | | |
|---|---|---|
| *Lv* | New York, Grand Central Terminal | 5.30 p. m. |
| | Dinner in dining car a la carte | |

*New England Members via B. & A. R. R.*

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|---|---|---|
| *Lv* | Boston .............. (Regular train) | 2.00 p. m. |
| *Lv* | Worcester ........... (Regular train) | 3.12 p. m. |
| *Lv* | Springfield ......... (Regular train) | 4.37 p. m. |
| | Dinner in Dining Car a la carte | |
| *Ar* | Albany ............ New York Central R. R. | 7.45 p. m. |
| *Lv* | Albany | 9.00 p. m. |
| *Lv* | Schenectady | 9.32 p. m. |
| *Lv* | Utica | 11.08 p. m. |

**Friday, September 15th**

| | | |
|---|---|---|
| *Lv* | Syracuse | 12.30 a. m. |
| *Lv* | Cleveland | 7.25 a. m. |
| *Lv* | Toledo | 10.05 a. m. |
| *Ar* | Chicago | 4.00 p. m. |
| *Lv* | Chicago, C. R. I. & P. | 8.05 p. m. |

**Saturday, September 16th**

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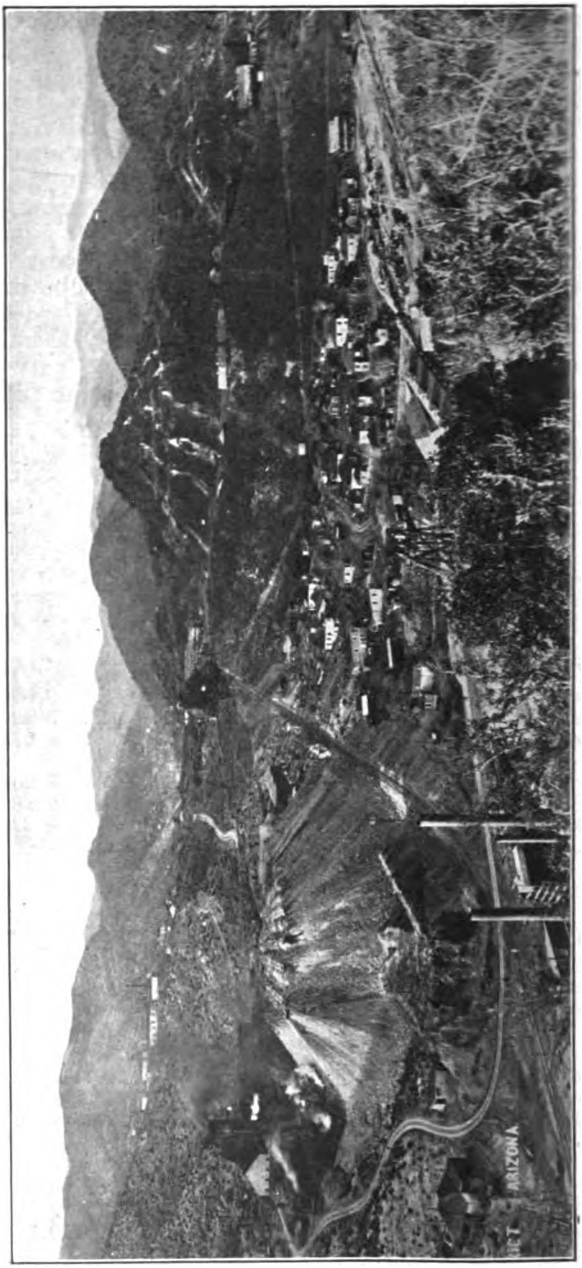
|---|---|---|
| *Lv* | Kansas City, C. R. I. & P. | 11.00 a. m. |
| *Lv* | Topeka, C. R. I. & P. | 1.05 p. m. |
| *Lv* | Hutchinson | 6.05 p. m. |

**Sunday, September 17th**

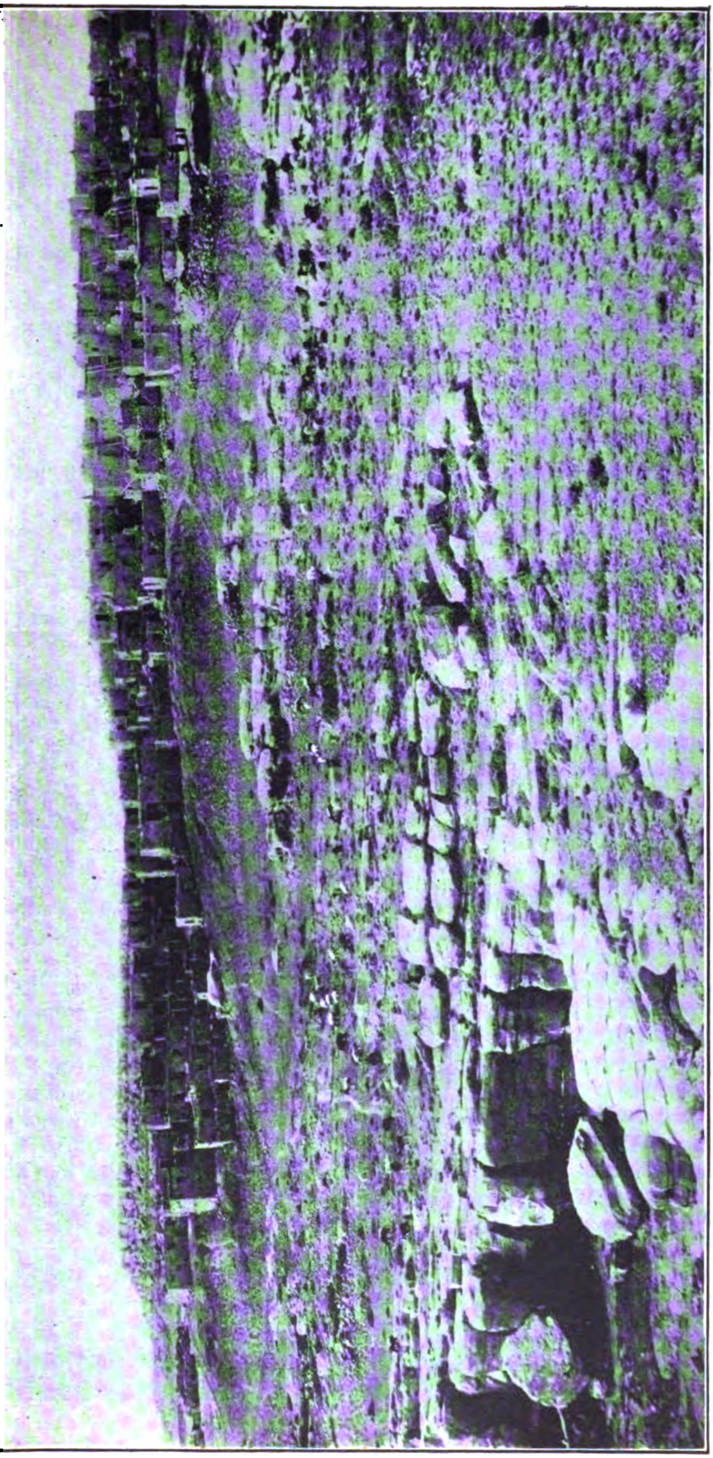
| | | |
|---|---|---|
| *Lv* | Santa Rosa, E. P. & S. W. R. R. | 6.30 a. m. |
| *Ar* | El Paso | 2.40 p. m. |
| *Lv* | El Paso, Southern Pacific | 10.20 p. m. |

**Monday, September 18th**

| | | |
|---|---|---|
| *Av* | Santa Rita | 4.00 a. m. |
| | Visit mines of Chino Copper Co. and mill of Empire Zinc Co. | |
| *Lv* | Santa Rita, Sante Fe | 2.00 p. m. |




BISBEE DISTRICT, ARIZONA, SHOWING SHAFTS OF THE CALUMET & ARIZONA AND COPPER QUEEN MINING COMPANIES.



A HOPI INDIAN VILLAGE IN ARIZONA. (Bulletin 613, U. S. Geological Survey.)

*Ar* Hurley .......... 2.45 p. m.
Visit mill of China Copper Co.
*Lv* Hurley, Sante Fe .......... 10.15 p. m.

**Tuesday, September 19th**

*Ar* Douglas .......... 7.00 a. m.
Visit reduction works of Copper Queen Consolidated Mining Co., and of Calumet & Arizona Mining Co.
Technical Session on "Smelting."
Technical Session on "Leaching."
*Lv* Douglas, E. P. & S. W. .......... 11.00 p. m.

**Wednesday, September 20th**

*Ar* Bisbee, Arizona .......... 1.00 a. m.
Visit to mines of district.
Technical Session, subject "Mining and Geology"
*Lv* Bisbee, E. P. & S. W. .......... 10.00 p. m.

**Thursday, September 21st**

*Ar* Globe .......... 9.00 a. m.
Visit to mines and reduction works of Old Dominion Copper Co.
Technical Session, subject "Concentration and Flotation."
Technical Session, subject "Fine Grinding."

**Friday, September 22nd (Globe)**

Visit reduction works of International Smelting & Refining Co., and mills of The Inspiration Consolidated Copper Co., and the Miami Copper Co.

Technical Session, subject "Mining and Smelting."

(The Ray Consolidated Copper Co. will welcome members and guests who desire to visit the mine at Ray. Special arrangements must be made by omitting other features on the program and with the understanding that those who go to Ray must reach Phoenix in time to take the train from there.)

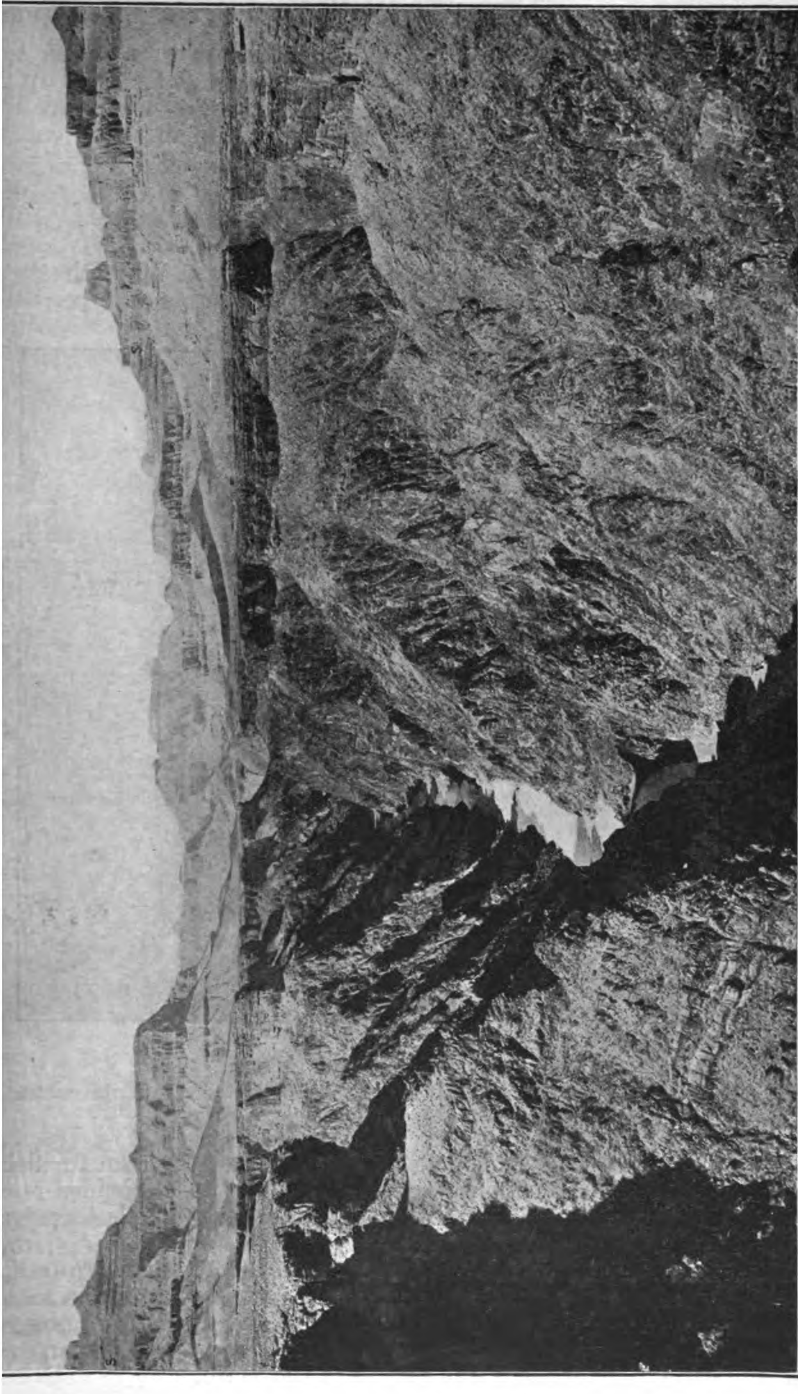
**Saturday, September 23rd (Globe)**

*Lv* Globe by automobile over Apache Trail, including visit to Roosevelt Dam.
*Ar* Phoenix about .......... 5.30 p. m.
*Lv* Phoenix, Sante Fe .......... 8.00 p. m.

## THE APACHE TRAIL

History records that the ancient trail leading to the ruined cliff dwellings in the Salt River Valley was discovered in 1540 by Francisco Vasquez de Coronado, the Spanish explorer, in his search for the treasures of the "seven cities" of Cibola, the abode of the Zuni Indians—the "Fathers of the Pueblos." Later the trail was used by the Apache Indians and this wild and rugged region was one of their last retreats—the scene of desperate encounters with our troopers in the campaign of 1872. Today the modern motor car follows the same historic and romantic trail through scenery of unusual variety and beauty over the highway between Globe and Phoenix, a distance of 120 miles.

The building of the great Roosevelt Dam, which required the construction of the roadway through the wildest and most rugged sections of the route, has made possible this unique mountain tour, and the consequent transformation of the Salt River and Tonto Creek basins into a great lake region has added a charming aspect to views already grandly impressive.



THE GRANITE GORGE IN THE GRAND CANYON, NORTHWEST OF GRANDVIEW POINT. (Bulletin 613, U. S. Geological Survey.)

## CLIFF DWELLINGS

At a point four and one-half miles east of Roosevelt, a wagon trail leads up a small canyon to a point where a foot trail may be taken up the steep mountain side to a group of ancient cliff dwellings which are well worth visiting. Another and more extensive group of ruins can be reached two miles farther by trail.

## ROOSEVELT DAM

This great dam has a maximum height of 280 feet. When the reservoir is full the water backs up fifteen miles in the Tonto and fifteen miles

A WATER BOTTLE IN THE DESERT.
Taking a drink pressed from the pulp that forms the interior of a barrel cactus, or visnaga. (Bulletin 613, U. S. Geological Survey.)

in Salt River. A hydro-electric plant located below the dam furnished current for use during the construction period and now provides power for irrigation pumping on the Salt River Project and for sale to private consumers.

The Salt River Valley is a land of perennial summer, of oranges, of cotton and of dates. Alfalfa is a staple, and ostrich farming is among the successful enterprises. Irrigation has hastened the valley's progress and confirmed its business stability. Phoenix is its metropolis, an energetic and attractive city, and the capital of the State.

**Sunday, September 24th**

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|---|---|
| *Ar* Grand Canyon | 8.00 a. m. |
| *Lv* Grand Canyon, Sante Fe | 7.40 p. m. |

## GRAND CANYON OF ARIZONA

The series of tremendous chasms which form the channel of the Colo-
do River in its course through northern Arizona, reach their culmina-
on in a chaotic gorge, 217 miles long, from 9 to 13 miles wide, and more
ian 6,600 ft. below the level of the plateau.

Standing upon the brink of the plateau at the point of the canyon's
eatest width and depth, the beholder is confronted by a scene whose
ajesty and beauty are indescribable. It is one of the few widely adver-
sed spots which one need not fear approaching with anticipations too
alted. It has never been adequately described, and never will be.

The picture on the cover of this *Bulletin* is of the north side of the
rand Canyon as viewed by telescope from El Tovar Hotel. This illus-
ation and others of Arizona scenery herewith reproduced are from
*ulletin* 613, *U. S. Geological Survey,* Guidebook of the Western United
ates, Part C, The Santa Fe Route. These cuts were obtained through
ie courtesy of Dr. George Otis Smith, Director. The letters indicate
ck formations as follows:

*G*, granite and gneiss; *U*, sandstone, red shale, and limestone (Unkar);
, sandstone of Tonto Platform; *Sh*, shale of Tonto group lying directly
ı quartzite of Unkar, *R*, limestone (Redwall); *S*, red sandstone and shale
upai); *C*, gray sandstone (Coconino); *K*, limestone (Kaibab). The
edwall butte in center is Cheops Pyramid. Beyond it are Buddha and
anu temples. The background is the Kaibab Plateau.

**Monday, September 25th**

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| *Lv* Albuquerque, Santa Fe | 7.00 p. m. |

**Tuesday, September 26th**

| | |
|---|---|
| *Lv* La Junta, Sante Fe | 7.10 a. m. |
| *Lv* Kansas City, Sante Fe | 11.00 p. m. |

**Wednesday, September 27th**

| | |
|---|---|
| *Ar* Chicago | 11.15 a. m. |
| *Lv* Chicago, New York Central | 12.40 p. m. |
| *Lv* Cleveland, New York Central | 7.35 p. m. |

**Thursday, September 28th**

| | |
|---|---|
| *Ar* Albany | 6.27 a. m. |
| *Ar* Boston | 11.55 p. m. |
| *Ar* Grand Central Terminal | 9.40 a. m. |

Fares quoted below include railroad and sleeping car fares for the
itire trip. Fares from New York and Albany also include excess fares
plying to Chicago and return owing to extra fast schedule of train.

For the exclusive use of a compartment by one person, one and one-
alf railroad tickets are required and for a drawing room, two tickets are
quired.

The arrangements as announced, also railroad and sleeping car fares

shown below, are conditional upon one hundred fares New York to El Paso, one hundred twenty-five fares El Paso to Grand Canyon and one hundred fares Grand Canyon to New York.

| Starting point | Lower 1 Person | Upper 1 Person | Compartment 2 Persons (Each) | Drawing Room 2 Persons (Each) |
|---|---|---|---|---|
| New York | $228.50 | $217.50 | $233.50 | $261.00 |
| Albany | 216.60 | 205.60 | 221.60 | 249.10 |
| Schenectady | 208.96 | 197.96 | 213.96 | 241.46 |
| Utica | 205.84 | 194.84 | 210.84 | 238.34 |
| Syracuse | 203.84 | 192.84 | 208.84 | 236.34 |
| Rochester | 200.60 | 189.60 | 205.60 | 233.10 |
| Buffalo | 198.40 | 187.40 | 203.40 | 230.90 |
| Cleveland | 188.80 | 177.80 | 193.80 | 221.30 |
| Toledo | 186.30 | 175.30 | 191.30 | 218.80 |
| Chicago | 176.80 | 165.80 | 181.80 | 209 30 |
| Kansas City | 160.30 | 149.30 | 165.30 | 192.80 |

## APPLICATIONS

Those desirous of joining the Tour should promptly communicate with Mr. W. V. Lifsey, Assistant General Passenger Agent, New York Central Lines, 1216 Broadway, New York City. Reservations will be made in the order in which applications are received.

Members of the Arizona Section desiring accommodations on the Special Train should communicate with Mr. Arthur Notman, Box 400, Bisbee, Ariz.

## PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JUNE 23, 1916

Report was made by the Committee of National Engineering Societies on the Reserve Corps of Engineers and it was voted that a committee of three be formed—consisting of the President of the Institute, Dr. Henry S. Drinker and the Secretary of the Institute—to select members in different parts of the country to advise with the local U. S. Engineer Officers as to qualifications of applicants for the Reserve Corps of Engineers.

## NOMINATIONS FOR OFFICERS

The suggestions of the members of the Institute are very much desired by the Committee on Nominations prior to deciding upon its nominations to fill the places of those officers who retire early in 1917. The Committee must make its final report to the Board of Directors early in October. The members are, therefore, asked to send in promptly their suggestions for Directors as well as for Vice-Presidents and President. The Committee hopes to hear from a large number of the members.

The officers to be elected at the annual meeting in February, 1917, are:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N.B.—New York City and district is designated District 0.)
District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, L. D. Ricketts (not eligible for re-election).

Past President, Benjamin B. Thayer.

Vice-Presidents, H. C. Hoover, District 6; Joseph W. Richards, District 2.

Directors, Reginald W. Brock, District 11; Albert R. Ledoux, District 0; D. C. Jackling, District 7; Charles W. Merrill, District 6; Henry L. Smyth, District 1.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, 1208 Hollingsworth Bldg., Los Angeles, Cal.

SEELEY W. MUDD,
*Chairman, Committee on Nominations.*

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## ENGINEERS' RESERVE CORPS

The Army Reorganization Act, which became effective July 1, 1916, contains a provision authorizing the organization of an Officers' Reserve, in which will be included a Corps of Reserve Engineers. There will also be an Enlisted Reserve of skilled workmen in technical lines. The special provisions of the bill relating to the Engineer Officers' Reserve were formulated with the help of the Joint Committee of the several national engineering societies, which has been at work on the matter for over a year with the War Department and Committees of Congress.

Recognizing that engineering science is the basis of success in modern

warfare, it is apparently the purpose, in thus mobilizing the engineering talent of the country, to give every engineer or technical specialist an opportunity, if the country can use his services, to do that which he can do best, and not necessarily to try and train all to the same standard of military engineering work for service with troops. At the same time, exceptional opportunities are offered to those who wish to qualify for this kind of active service and for those engineers who are inclined to seek commissions as engineer officers in the permanent establishment. It will be noted that the usual age limits are waived in the case of the Engineers' Reserve Corps.

But for the disarrangement of the ordinary routine of the War Department, resulting from the present emergency on the Mexican border, it is probable that the machinery for organizing the Engineers' Reserve Corps would already be in motion, but under the circumstances, it will be necessary to await the convenience of the authorities.

The report of the Joint Committee follows:

To the Boards of Direction of the

AMERICAN SOCIETY OF CIVIL ENGINEERS,

AMERICAN INSTITUTE OF MINING ENGINEERS,

AMERICAN SOCIETY OF MECHANICAL ENGINEERS,

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS,

AMERICAN INSTITUTE OF CONSULTING ENGINEERS.

DEAR SIRS:—On December 22, 1915, the Joint Committee (consisting of the Chairmen of your several committees), formed under the authority of the five societies in order to facilitate the carrying out of the organization of an Engineer Reserve as part of the military forces of the United States, reported the progress that had been made up to that time in the matter of the necessary legislation.

The Joint Committee has now to report to you that the legislation referred to is embodied in a bill which recently passed Congress, has been signed by the President, and will become effective July 1, 1916. This bill, known as the Army Reorganization Act of 1916, contains provisions for the organization of an Officers Reserve, including engineers. An abstract of the principal clauses relating to the Officers Reserve and to the Enlisted Reserve Corps is appended hereto.

The War Department is now formulating the plan of organization of the Reserve Corps of Engineers and of the Enlisted Reserve Corps and the requirements that will be prescribed for enrollment therein. As soon as practicable after July 1, the War Department will issue an invitation to the engineers of the country to apply for commissions in the Reserve Corps of Engineers, and there will be available for those interested full and complete information as to the requirements for commissions in the several grades from Second Lieutenant to Major, and for Master Engineers and Sergeants, and the method of procedure to secure commissions or enlistment.

The Joint Committee has notified the War Department that its services in these matters are at the call of the Department.

Very truly yours,

WM. BARCLAY PARSONS, Chairman,
*American Society of Civil Engineers,*
H. S. DRINKER,
*American Institute of Mining Engineers,*
WM. H. WILEY,
*American Society of Mechanical Engineers,*
B. J. ARNOLD,
*American Institute of Electrical Engineers,*
RALPH D. MERSHON,
*American Institute of Consulting Engineers.*

23D JUNE, 1916.

## ABSTRACT

## THE ENGINEER SECTION OF THE OFFICERS RESERVE CORPS AND THE ENLISTED RESERVE CORPS

### From the Army Reorganization Act of 1916, Sections, 37, 38, 39, 55

Sec. 37.—The Officers Reserve Corps.—For the purpose of securing a reserve of officers available for service as temporary officers in the Regular Army, as provided for in this Act and in Section eight of the Act approved April twenty-fifth, nineteen hundred and fourteen, * * * there shall be organized, under such rules and regulations as the President may prescribe not inconsistent with the provisions of this Act, an Officers Reserve Corps of the Regular Army. Said corps shall consist of sections corresponding to the various arms, staff corps, and departments of the Regular Army. Except as otherwise herein provided, a member of the Officers Reserve Corps shall not be subject to call for service in time of peace, and whenever called upon for service shall not, without his consent, be so called in a lower grade than that held by him in said reserve corps.

The President alone shall be authorized to appoint and commission as reserve officers in the various sections of the Officers Reserve Corps, in all grades up to and including that of major, such citizens as, upon examination prescribed by the President, shall be found physically, mentally and morally qualified to hold such commissions: *Provided*, That the proportion of officers in any section of the Officers Reserve Corps shall not exceed the proportion for the same grade in the corresponding arm, corps, or department of the Regular Army, except that the number commissioned in the lowest authorized grade in any section of the Officers Reserve Corps shall not be limited.

* * * * * * * *

The commissions of all officers of the Officers Reserve Corps shall be in force for a period of five years unless sooner terminated in the discretion of the President. Such officers may be recommissioned, either in the same or higher grades, for successive periods of five years, subject to such examinations and qualifications as the President may prescribe and to the age limits prescribed herein: *Provided*, That officers of the Officers Reserve Corps shall have rank therein in the various sections of said Reserve Corps according to grades and to length of service in their grades.

Sec. 38.—The Officers Reserve Corps in War.—In time of actual or threatened hostilities the President may order officers of the Officers Reserve Corps, subject to such subsequent physical examinations as he may prescribe, to temporary duty with the Regular Army in grades thereof which can not, for the time being, be filled by promotion, or as officers in volunteer or other organizations that may be authorized by law, or as officers at recruit rendezvous and depots, or on such other duty as the President may prescribe. While such reserve officers are on such service they shall, by virtue of their commissions as reserve officers, exercise command appropriate to their grade and rank in the organizations to which they may be assigned, and shall be entitled to the pay and allowances of the corresponding grades in the Regular Army, with increase of pay for length of active service, as allowed by law for officers of the Regular Army, from the date upon which they shall be required by the terms of their orders to obey the same: *Provided*, That officers so ordered to active service shall take temporary rank among themselves and in their grades in the organizations to which assigned, according to the dates of orders placing them on active service; and they may be promoted, in accordance with such rank, to vacancies in volunteer organizations or to temporary vacancies in the Regular Army thereafter occurring in the organizations in which they shall be serving. * * *

Any officer who, while holding a commission in the Officers Reserve Corps, shall be ordered to active service by the Secretary of War shall, from the time he shall be required by the terms of his order to obey the same, be subject to the laws and regulations for the government of the Army of the United States, in so far as they are applicable to officers whose permanent retention in the military service is not contemplated.

Sec. 39.—Instruction of Officers of the Officers Reserve Corps.—To the extent provided for from time to time by appropriations for this specific purpose, the Secretary of War is authorized to order reserve officers to duty with troops or at field exercises, or for instruction, for periods not to exceed fifteen days in any one calendar year, and while so serving such officers shall receive the pay and allowances of their respective grades in the Regular Army: *Provided*, That, with the consent of the reserve officers concerned, and within the limit of funds available for the purpose, such periods of duty may be extended for reserve officers as the Secretary of War may direct. * * *

* * * * * * * *

Sec. 55.—The Enlisted Reserve Corps.—For the purpose of securing an additional reserve of enlisted men for military service with the Engineer * * * Corps * * * of the Regular Army, an Enlisted Reserve Corps, to consist of such number of enlisted men of such grade or grades as may be designated by the President from time to time, is hereby authorized, such authorization to be effective on and after the first day of July, nineteen hundred and sixteen.

There may be enlisted in the grade or grades hereinbefore specified, for a period of four years, under such rules as may be prescribed by the President, citizens of the United States, or persons who have declared their intentions to become citizens of the United States, subject to such physical, educational and practical examination as may be prescribed in said rules. For men enlisting in said grade or grades certificates of enlistment in the Enlisted Reserve Corps shall be issued by the Adjutant General of the Army, but no such man shall be enlisted in said corps unless he shall be found physically, mentally and morally qualified to hold such certificate and unless he shall be between the ages of eighteen and forty-five years. The certificates so given shall confer upon the holders when called into active service or for purposes of instruction and training, and during the period of such active service, instruction, or training, all the authority, rights and privileges of like grades of the Regular Army. * * * And the Secretary of War is hereby authorized to issue to members of the Enlisted Reserve Corps and to persons who have participated in at least one encampment for the military instruction of citizens, conducted under the auspices of the War Department, distinctive rosettes or knots designed for wear with civilian clothing.

The President is authorized to assign members of the Enlisted Reserve Corps as reserves to particular organizations of the Regular Army, or to organize the Enlisted Reserve Corps, or any part thereof, into units of detachments of any arm, corps, or department in such manner as he may prescribe, and to assign to such units and detachments officers of the Regular Army or of the Officers Reserve Corps, herein provided for.

To the extent provided from time to time by appropriations, the Secretary of War may order enlisted men of the Enlisted Reserve Corps to active service for purposes of instruction or training for periods not to exceed fifteen days in any one calendar year: *Provided*, That, with the consent of such enlisted men and within the limits of the funds available for such purposes, such periods of active service may be extended for such number of enlisted men as may be deemed necessary.

Enlisted men of the Enlisted Reserve Corps shall receive the pay and allowances of their respective grades, but only when ordered into active service, including the time required for actual travel from their homes to the places to which ordered and return to their homes. * * *

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## PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period June 10, 1916 to July 10, 1916:

**Arthur Adams**, Miami, Ariz.
**J. H. Batcheller**, Mattapoisitt, Mass.
**R. S. Bonsib**, Low Moor, Va.
**E. L. Booth**, New York, N. Y.
**C. L. Bryden**, Scranton, Pa.
**F. Crabtree**, Pittsburgh, Pa.
**L. B. Eames**, Pueblo, Colo.
**S. J. Gormly**, Guayacan, Chile.
**Ernest A. Hersam**, Berkeley, Cal.
**A. W. Hudson**, Douglas, Ariz.
**Chas. W. Johnston**, West Norfolk, Va.
**John M. Lovejoy**, New Rochelle, N. Y.
**J. F. Manning**, Seoul, Korea.
**E. P. Mathewson**, Anaconda, Mont.
**Edwin W. Mills**, Seoul, Korea.
**Arthur Notman**, Bisbee, Ariz.
**L. B. Reifsneider**, Daiquiri, Cuba.
**W. S. Schuyler**, Berkeley, Cal.
**O. N. Scott**, Toronto, Ont., Canada.
**Arthur F. Taggart**, New Haven, Conn.
**Warren D. Thompson**, Boston, Mass.
**Thor. Warner**, Kingman, Ariz.

**Rodolphe L. Agassiz** has been chosen president of the Calumet & Hecla Mining Co., an office which has been vacant for over a year.

**Percy E. Barbour**, Managing Editor of the *Engineering and Mining Journal*, resigned his commission as 2d Lieutenant in the Main Coast

illery to accept a commission as 1st lieutenant in the 22d Corps of ;ineers, N. G. N. Y. He is now at Fort McAllen, Tex.

**H. Kenyon Burch,** Chief Engineer of the Inspiration Consolidated ›per Co., completed his work pertaining to the design and construction he plant, and left on July 1st for an extended vacation trip throughout East. His forwarding address for the next few months will be care of : Sierra Madre Club, L. A. Investment Bldg., Los Angeles, Cal.

**Corey C. Brayton** and **E. R. Richards** have established offices in the ›art Bldg., San Francisco, Cal., for the purpose of conducting the iness of Mining and Metallurgical Engineering.

**Juan Felix Brandés** has removed his offices to 139 Whittell Bldg., 166 ıry St., San Francisco, Cal.

**S. A. Crandall** has accepted a position as draftsman with the ıker Hill and Sullivan Smelter, Kellogg, Ida.

**John C. Devine,** since 1906 assistant superintendent of the Ray Condated Copper Co., has resigned and will devote his attention to the elopment of the Renfro property of the Pinal Development Co., of ch he is président.

**Edward B. Durham** is now with the Mammoth Copper Mining Co., ınett, Cal.

**Edward L. Estabrook** has become geologist for the Wisconsin Zinc , at Platteville., Wis.

**F. Julius Fohs,** who has devoted himself to a consulting practice in geology for the past four years, announces that he has opened new ces in the Gallais Building at Tulsa, Okla., with a staff of six geolsts and one topographer, and that he is covering an increasing scope :ountry with his structural surveys.

**George A. Guess,** professor of metallurgy at the University of Toronto, been engaged by the Vermont Copper Co. to start up the copper smeltat South Strafford, Orange Co., Vt., which has never been operated cessfully.

**Henry M. Howe** has just been appointed Honorary Vice-President of Iron and Steel Institute of Great Britain. There are only two others, ›fessor D. Tschernoff, of Russia; and Giorgio E. Flack, of Italy.

**Capt. Robert W. Hunt** has been honored by the Rensselaer Polyhnic Institute, with which he has been long identified, with the degree Doctor of Engineering.

**E. T. Lednum** has been made manager of E. I. DuPont de Nemours Co., with offices in Suite 808, Frisco Bldg., Joplin, Mo.

**E. Fleming L'Engle** has been appointed manager of the Royal Zinc ., Joplin, Mo.

**Waldemar Lindgren,** Professor of Economic Geology at Massachusetts titute of Technology, has been awarded the honorary degree of Doctor Science by Princeton University.

**N. O. Lawton** has completed the exploration work in which he was engaged in Hancock Co., Tenn., and will be at South Strafford, Vermont, during July, examining copper properties.

**C. H. Macnutt** has resigned his position as manager of Burma Mines Co., Namtu, Burma.

**W. G. Matteson** has accepted the position of geologist for the Empire Gas and Fuel Co., Bartlesville, Okla.

**William B. Phillips** has resigned the presidency of the Colorado School of Mines, and has returned to Austin, Texas, to resume practice as a mining engineer and metallurgist.

**Fred S. Porter** has accepted a position in the engineering office of the Canadian Klondike Mining Co., Dawson, Yukon Territory, Canada.

**F. Danvers Power** has opened new offices in the Perpetual Trustee Chambers, Hunter St., Sydney, N. S. W., Australia.

**H. W. Ross** has accepted the position of Assistant Manager of the Backus & Johnston Co., Casapalca, Peru.

**J. D. Sperr** has resigned as mining engineer of the Tom Reed Gold Mines Co., Oatman, Ariz.

**F. W. Traphagen** has resigned the position of professor of metallurgy of the Colorado School of Mines to accept the presidency of the Colorado Metal Mining & Reduction Co.

**Carl J. Trauerman,** mining engineer, has resigned his position as mill superintendent of the August Mining Co., Landusky, Mont., and is inspecting the properties of the Beaver Creek Mines Co., Zortman, Mont. After examining properties in the Kendall, Elliston and York districts of Montana he will return to his headquarters at Butte.

**Jesse Merrick Smith** and **Miss Mabel Amy MacKinney** were married on June 24, at Tarrytown-on-the-Hudson, N. Y.

**Edward Vestal Graybeal** and **Miss Alice Seabury Mathewson** were married on June 14, at Anaconda, Mont.

## POSITIONS VACANT

Wanted, for a new smelting company now being organized in China: one man familiar with the design and operation of a zinc smelter; one experienced in the design and operation of a lead smelter; and a chemist competent to make the usual analyses in connection with lead and zinc smelting and to make clay analyses. Men with their wives preferred. Company will pay all expenses to and from China, medical attendance, salary, bonus per ton of zinc or lead produced, allowance for food, and furnish quarters. No. 120.

## ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, oil geologist, age 33 years, technical education, nine years' experience with oil and mining companies, wishes employment with repu-

le company or syndicate exploiting proven oil lands or searching for r districts. First class references. No. 300.

Member, age 30, unmarried, Columbia graduate, speaks fair French, writes and speaks Spanish fluently. Mining and geological experie in United States, Mexico and South America. Desires position h examining engineers or exploration company. No. 301.

Member, technical graduate in mining engineering, three years' exience in surveying, sampling and underground engineering in metal es of the West, desires position as mine surveyor, field engineer or stant to consulting engineer. No. 302.

Member, age 30, technical graduate, 4 years' experience in tin contration; at present Superintendent of Mills in largest Bolivian tin es; executive ability; seeks change, preferably with American comy interested in locating or investigating in South America. Fluent nish. No. 303.

Open for engagement. Master Mechanic. A man experienced in lter construction and operation. Lead and copper especially. 18 rs as M. M. in British Columbia and Mexico. Reliable, efficient, h good record. Best of reference. No. 304.

Engineer, 42 years old, married, experienced in cyanidation, concenion and construction of plants, desires position as operator or construc- engineer or in plant designing. Speaks Spanish and some German. . 305.

Mining engineer with 6 years' operating experience in Colorado and zona desires position with substantial company. Employed at pres- . Age, 30. Married. References. No. 306.

Member, technical graduate, who has specialized in the treatment of d and silver ores, desires position as mill superintendent or foreman; ears' practical experience. Highest references as to character and lity to get results. No. 307.

Member with broad general business experience. Past 5 years presit and general manager of active developing and producing oil comy in California. Open for engagement in California preferred. . 308.

Member, technical metallurgical graduate. 5 years' experience in ious mining operations. Mill and laboratory work including concenting, cyaniding and flotation. Desires position in flotation work. e, 30. Married. No. 309.

Member, aged 30, technical graduate in mining and metallurgy. At sent employed as assistant superintendent of concentrator and flota- plant. Also had practical and engineering mining experience. sires change. No. 310.

Member, aged 26, graduate in mining and metallurgy, 2 years' experie with engineering work of lead and copper smelters, good draftsman; ires permanent connection with engineering or operating department smelter. Excellent references. At present employed, but available short notice. No. 311.

## LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

### Apparatus for Making Photographic Reproductions

The United Engineering Society has installed a photographic duplicating machine for the reproduction of pages from books and periodicals, maps and charts in the Library of the Engineering Societies, and it is now in operation. Orders may be sent to the Library of the Engineering Societies, 29 West 39th Street, New York.

The process produces a photograph (white on black) direct on bromide paper, and from this negative print a positive may be made by re-photographing. The charge for a print 11 by 14 in., which may cover two pages of an octavo volme that face each other, is 25 cents for a negative print and 50 cents for a positive print. Enlargements and reductions to 11 by 14 in. are easily made. The process has the advantages of accuracy and cheapness.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

## Mining and Metallurgy

'-PRODUCTS RECOVERED IN THE MANUFACTURE OF COKE. By W. H. CHILDS. n.p., 1916. (Gift of D. D. Berolzheimer.)

›AL DISTILLATION, GASIFICATION AND BY-PRODUCTS. By J. E. CHRISTOPHER WIGAN, 1915.

IE ANALYSIS OF PERMISSIBLE EXPLOSIVES. Bull. 96, U. S. Bureau of Mines. Washington, 1916.

.ND TEST FOR DETERMINING THE STRENGTH OF DETONATORS. Tech. Paper No. 125, U. S. Bureau of Mines. Washington, 1916.

›STRACTS OF CURRENT DECISIONS ON MINES AND MINING, REPORTED FROM OCTOBER TO DECEMBER, 1915. Bull. No. 118, U. S. Bureau of Mines. Washington, 1916.

'FECTS OF ATMOSPHERES DEFICIENT IN OXYGEN ON SMALL ANIMALS AND ON MEN. Tech. Paper No. 122, U. S. Bureau of Mines. Washington, 1915.

INERS' NYSTAGMUS. Bull. No. 93, U. S. Bureau of Mines, Washington, 1916.

ST OF BOOKS RELATING TO THE IRON AND STEEL INDUSTRY. Contained in the principal libraries in the West of Scotland, Glasgow, 1916. (Gift of West of Scotland Iron and Steel Institute.)

## Geology and Mineral Resources

EVIEW OF PRINCIPLES OF OIL AND GAS PRODUCTION. Compiled by ROSWELL H. JOHNSON and L. G. HUNTLEY. John Wiley & Sons, Inc., New York, 1916. Price $3.75. Gift of publishers.

[A book on oil by the two distinguished authors of this volume is sure to arouse eat interest among members of the Institute who are familiar with the writings of ese same authors in our publications. To such it is not necessary for us to say anying as to the accuracy, care, and thoroughness of the work, and we commend the ›lume to all. It is well and appropriately illustrated by geological sketches, drawgs, curves, photographs, and so forth, and it covers the following subjects:

Varieties of Oil and Gas, The Origin of Oil and Gas, Distribution of the Oil and as, Reservoirs of Oil and Gas, Accumulation of Oil and Gas, Pressure in Oil and Gas eservoirs, Origin of the Shape of the Reservoir, Classification of the Attitude of eologic Surfaces, Effect of the Different Attitudes upon Accumulation, Locating Oil ıd Gas Wells, Oil and Gas Lands, Drilling for Oil and Gas, Bringing in a Well, The [anagement of Oil Wells, Completing the Extraction of the Oil, The Management Gas Wells, Condensation of Gasoline from Gas, The Natural Gas Industry, Size ıd Scope of Oil and Gas Companies, Reports upon Oil and Gas Prospects or roperties, The Valuation of Oil Properties, Oil and Gas Fields of North America, il Market and the Future Supply.]

'EPOSITS OF THE USEFUL MINERALS AND ROCKS, THEIR ORIGIN, FORM AND CONTENT. By F. Beyschlag and others. Vol. II, London, 1916.

IL AND GAS MAP OF SOUTHWESTERN PENNSYLVANIA, 1915. Harrisburg, 1916.

HE CADDO OIL AND GAS FIELD, LOUISIANA AND TEXAS. Bull. No. 619, U. S. Geological Survey. Washington, 1916.

ETROLEUM AND NATURAL GAS PROSPECTS AT ROMA. Publication 247, Queensland Geological Survey. Brisbane, 1915.

'UNGSTEN AND MOLYBDENUM. Part I–II. (Tasmania. Geological Survey Mineral Resources, No. 1.) Tasmania, 1916.

'HE WOLFRAM, MOLYBDENITE AND BISMUTH MINES OF BAMFORD, NORTH QUEENSLAND. Publication 248, Queensland Geological Survey. Brisbane, 1915.

ELDSPARS OF THE NEW ENGLAND AND NORTH APPALACHIAN STATES. Bull. No. 92, U. S. Bureau of Mines. Washington, 1916.

INC-LEAD SULPHIDE DEPOSITS OF THE READ-ROSEBERY DISTRICT PT. II.—Rosebery group with maps. Bull. No. 23, Tasmania Geological Survey. Tasmania, 1915.

.TLANTIC GOLD DISTRICT AND THE NORTH LARAMIE MOUNTAINS, FREMONT, CONVERSE, AND ALBANY COUNTIES, WYOMING. Bull. No. 626, U. S. Geological Survey. Washington, 1916.

ŀEOLOGY AND COAL RESOURCES OF CASTLE VALLEY IN CARBON, EMERY, AND SEVIER COUNTIES, UTAH. Bull. 628, U. S. Geological Survey. Washington, 1916.

GEOLOGY AND MINERAL RESOURCES OF THE SOUTHERN COAL FIELD. Part I—SOUTH COSTAL PORTION. With Maps and Sections. New South Wales. Memoirs of the Geological Survey. Geology No. 7. Sydney, 1915.
GEOLOGY AND ORE DEPOSITS OF ROSSLAND, BRITISH COLUMBIA. Memoir No. 77. Canada Dept. of Mines. Ottawa, 1915.
GEOLOGY OF FIELD MAP-AREA, B. C. AND ALBERTA. Memoir No. 55, Canada Dept. of Mines. Ottawa, 1914.

## General

CIVIL ENGINEERS' COST BOOK. Ed. 2. By T. E. Coleman. London, 1916.
DIRECTORY OF THE IRON AND STEEL WORKS OF THE UNITED STATES AND CANADA, 1916. 18th Ed. New York, 1916.
LAND AND MARINE DIESEL ENGINES. By Giorgio Supino. Translated by A. G. Bremner and James Richardson. London, 1915.
THE DIESEL ENGINE IN PRACTICE. By J. E. Megson and H. S. Jones. San Francisco, 1916.

## Gift of Hill Publishing Company

A good collection of 29 text books.

## Trade Catalogs

E. I. DUPONT DE NEMOURS & CO. Wilmington, Del. DuPont Products (A-430). April, 1916.
SULLIVAN MACHINERY CO., Chicago, Ill. Bull. 70-C. The Sullivan "Dr-6" mounted water hammer drill. May, 1916.
SPARTA IRON WORKS CO., Sparta, Wis. Sludge Bucket. June, 1916.
STEPHENS-ADAMSON MFG. CO., Aurora, Ill. Labor Saver. June, 1916.
TEXAS CO., Houston & New York. About Texaco crater compound, 1916.

## Exposition of Chemical Industries

That the chemical industries are alive to the situation that they now occupy in the sum of affairs is evident in the preparations being made by many firms for the Second National Exposition of Chemical Industries to be held at the Grand Central Palace, New York City, during the week of September 25–30, 1916. The list of exhibitors indicates that the Exposition is already twice the size of its successful predecessor.

The American Chemical Society will hold its annual meeting during the whole week, the American Electrochemical Society has arranged to hold its meetings on September 28, 29 and 30, and the Technical Association of the American Pulp and Paper Industry is also planning to meet during the week.

The Bureau of Commercial Economics at Washington is again coöperating with the Exposition by arranging an elaborate program of motion pictures covering subjects dealing with the industries depending on chemistry. Many of these films will have their first showing at the Exposition. They will take the visitor through industrial plants, shops, manufactories, mines, etc., where visitors are never admitted; they will show operations that in ordinary light would never be seen; they will demonstrate more than any other kind of motion picture the great strides and improvement in the art of photography.

## MEMBERSHIP

### NEW MEMBERS

The following list comprises the names of those persons who became embers during the period June 10, 1916, to July 10, 1916:

›AMS, WILLIAM S., Filtration Engr., Anaconda Copper Min. Co., 701 Cherry, Anaconda, Mont.
.LEY, JAMES E., Shift-boss............Belmont Milling Co., Tonopah, Nev.
.DGLEY, CHARLES WELLING, Research Chemist, El Paso Smelting Works, El Paso, Tex.
.LLENBERG, ADOLF G., Engr., Porcupine Exploration Co., Ontonagon Co., Mich.
.RKER, RALPH FEINER, Met..................Garfield Smelt. Co., Garfield, Utah.
.OSSOM, EDWARD LUDLAM, Staff of Research Corpn., 63 Wall St., New York, N. Y.
.OUGHTON, EUGENE HARDING, Experimental Engr., Copper Queen Cons. Min. Co., Bisbee, Ariz.
.RFORD, SAMUEL W., Genl. Mgr., Cleveland Mining Co., 223 West Third St., Joplin, Mo.
.UTTERFIELD, GEORGE BRUCE, Inspector, The Associated Companies, First National Bank Bldg., Pittsburgh, Pa.
.AHILL, EDWARD GUSTAV, Contracting Engr., 460 Montgomery St., San Francisco, Cal.
.AULTON, THEODORE M.....................208 New York Bldg., Seattle, Wash.
.EANE, ELBRIDGE GERRY, Min. Engr..........Miami Copper Co., Miami, Ariz.
.UNCAN, L. GRANT, Met. Engr..................Mineral Point Zinc Co., Depue, Ill.
.ISELE, GEORGE J., Asst. Genl. Supt., Oliver Iron Mining Co., Iron Mountain, Mich.
.LYNN, JOHN G....................Mining, Miami Copper Co., Miami, Ariz.
.RENCH, R. W., Met...............Goldfield Cons. Mines Co., Goldfield, Nev.
.REUD, BENJAMIN BALL, Teacher, Associate Prof. of Chemistry, Armour Institute of Technology, Chicago, Ill.
.ARLOCK, GUY M., Fuel Dept., Prime Western Spelter Co., Box 83, Iola, Kans.
.EOHEGAN, LESLIE E.......Asst. to Prest., Gulf States Steel Co., Birmingham, Ala.
.OODALE, F. A., Supt. and Agent, You Bet Mining Co., P. O. Box. 290, Dutch Flat, Cal.
.ANDY, ROYAL SHEPPART, Mill Supt., Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
.ARDING, PHILIP ORR, Engr., Designing Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
.ENSLEY, JAMES H. JR., Min. Engr., Miami Copper Co., Box 100, Miami, Ariz.
.ERSHEY, OSCAR H., Min. Geol., Burch, Caetani & Hershey, Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
.OFFMANN, ROBERT BERNARD, Chief of Service Dept., Mineral Point Zinc Co., Depue, Ill.
.ULL, MONROE RICHMOND.....Chief Engr., Arizona Copper Co., Clifton, Ariz.
.ONES, J. CLAUDE, Prof. of Geol.............University of Nevada, Reno, Nev.
.ELSEY, W. M., Genl. Supt..................Mineral Point Zinc Co., Depue, Ill.
.IDDELL, DONALD M., Cons. Met. Engr.. Merrill, Lynch & Co., 7 Wall St., New York, N. Y.
.UNT, CHARLES FREDERICK, Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
.ANN, HORACE THARP, Associate Prof. of Met. and Ore Dressing, Missouri School of Mines and Metallurgy, P. O. Box 248, Rolla, Mo.
.ENDELSOHN, CHARLES, Mech. Engr., Old Dominion Copper Min. & Smelt. Co,. Box 657, Globe, Ariz.
.OORE, GEORGE PIERSON, Engr., The Henry Souther Engrg. Co., 11 Laurel St., Hartford, Conn.
.EAD, NORMAN HATFIELD, Min. Geol......................Manchester, Mass.
.OBBINS, O. D., Met. Engr., Chief of Ore and Testing Dept., Mineral Point Zinc Co., Depue, Ill.
.COTT, DAVID BURNET, Min. Engr., Efficiency Engr., Miami Copper Co., Box 100, Miami, Ariz.
.COTT, J. RALPH, Mine Engr., Dome Mines Co., Ltd., Lock Box 505, So. Porcupine, Ont., Canada.
.MITH, W. R., Supt..................Mineral Point Zinc Co., Mineral Point, Wis.

STAY, THERON D., Asst. Research Met., Aluminum Castings Co.,
2077 East 88th St., Cleveland, O.
STEIDLE, EDWARD, Technical Asst. to Chief Min. Engr., U. S. Bureau of Mines,
Washington, D C.
STOCKETT, NORMAN AYLEWORTH, Min. Engr., Mgr., Paragon Cons. Min. Co.,
Paragon, Idaho.
STRACHAN, CHARLES B., Mill Supt., American Zinc Co. of Tennessee, Mascot, Tenn.
STRAUB, CHARLES EDWARD, Cons. Oil Geol. & Engr.,
211-12 Lynch Bldg., Tulsa, Okla.
STROUP, THOMAS A., Mech. and Met. Engr., Tennessee Copper Co., Copperhill, Tenn.
TODD, W. PARSONS, Vice-Prest., Quincy Min. Co., 32 Broadway, New York, N. Y.
WATSON, HUGH C., Min. Engr., Genl. Supt., Wisconsin Zinc Co., Platteville, Wis.
WARD, ARTHUR THOMAS, Experimental Engr., Braden Copper Co., Sewell,
via Rancagua, Chile, S. Amer.
WEINTZ, LOUIS JOHN, Mill Supt., The New Jersey Zinc Co.,
Ogdensburg, Sussex Co., N. J.
WHITE, HENRY ARTHUR, Cons. Met., Geduld Prop. Mines, Ltd., Box 41,
Springs, Transvaal, S. Africa.
WOODWARD, TRUMAN STEPHEN, Chief Chemist, Ohio Works, Carnegie Steel Co.,
R. 2. Youngstown, O.
YERXA, R. B., Asst. Supt. of Concentrator, Miami Copper Co.,
P. O. Box 100, Miami, Ariz.
YOUNG, EARL BURDETT, Asst. in Geol. Dept., Anaconda Copper Min. Co.,
526 Hennessy Bldg., Butte, Mont.

## *Associate Members*

CAJANDER, A. W., Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
RIVES, HENRY MACON, Secy. and Treas., Nevada Mine Operators Assn.,
210-211 Reno National Bank Bldg., Reno, Nev.
ROBINSON, CHARLES H., Iron Mining, Pettit & Robinson Lands,
826 First National Soo Line Bldg., Minneapolis, Minn.

## *Junior Members*

BERTHIER, ULYSSES HENRY, Asst. in Testing Dept., International Smelt. Co.,
Box 115, Tooele, Utah.
BIRKETT, HOWARD, Student..........Columbia University, New York, N. Y.
BLAUROCK, CARL ALBERT, Gold and Silver Refiner, 2530 W. 37th Ave., Denver, Colo.
CHATIN, AUGUST H., Laborer, Engr. of Mines, Anaconda Copper Co.,
316½ W. Copper St., Butte, Mont.
CORBY, HARRY GILBERT, Student................1138 S. Maple St., Carthage, Mo.
CRAIG, JOHN J., Min. Engr., Northwestern Improvement Co., Brainerd, Minn.
EHLERS, LOUIS WILMER, Student..........Missouri School of Mines, Rolla, Mo.
ERDOFY, MAXWELL EMORY, Student.....Columbia University, New York, N. Y.
FERGUSON, KENNETH SEARS, Student, Colorado School of Mines, Golden, Colo.
FLOSS, FRITZ CARL, Student....................................Milwaukie, Ore.
HOFIUS, MAX T., Student..........Colorado School of Mines, Golden, Colo.
HU, SHIH-HUNG, Student......................P. O. Box 732, Golden, Colo.
KLEIN, KARL F., Min. Engr., Morococha Mining Co., Morococha, Peru, S. Amer.
PIERCE, COLWELL ARBA, Student......3417 Virginia Ave., Kansas City, Mo.
SCHMIDT, ROBERT DROEGE, Student..........1509 Alameda Ave., Lakewood, O.
STOTZ, NORMAN I., Student................Lehigh University, S. Bethlehem, Pa.
SUTTON, HARRY ALLEN, Senior Student, Oregon Agricultural College, Corvallis, Ore.
WIEBELT, FRANK J., Min. Engr., Laborer, Anaconda Copper Co.,
217 Colorado St., Butte, Mont.

**Total Membership, July 10, 1916............................ 5,612**

## CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that

-sons should be admitted to membership in classes for which they are not qualified. embers of the Institute can be of great service if they will make a practice of glancing rough the list of applicants and promptly notifying the Committee on Membership, the Secretary of the Institute, of any persons whom they think should not be ssified in accordance with the list given.

## MEMBERS

The following persons have been proposed during the period June 10, 16 to July 10, 1916, for election as members of the Institute. Their mes are published for the information of Members and Associates, from nom the Committee on Membership earnestly invites confidential comunications, favorable or unfavorable, concerning these candidates. A fficient period (varying in the discretion of the Committee, according the residence of the candidate) will be allowed for the reception of ch communications, before any action upon these names by the Comttee. After the lapse of this period, the Committee will recommend tion by the Board of Directors, which has the power of final election.

### *Members*

**Robert Sexton Allen,** Globe, Ariz.
Proposed by Rudolf Gahl, Guy H. Ruggles, C. E. Arnold, W. B. Cramer.
Born 1889, New Haven, Conn. 1896–1906, Grade and High School, Kansas City, ›. 1906–09, Missouri School of Mines. 1909–12, Lab. Work and Control Chem., v. Cons. Copper Co. 1912–13, Slag Chemist, Old Dominion Copper Co. 1913–14, em. and Asst. Engr., Superior & Boston Copper Co. 1914–16, Sr. Control Chem., v. Cons. Copper Co.
Present position: Chief Chem. of Concentrator, Inspiration Cons. Copper Co.

**Frederick Eurith Beasley,** Tacoma, Wash.
Proposed by Willard V. Morse, Roger E. Chase, Brent N. Rickard.
Born 1882, Peoria, Ill. 1891–1901, Elementary and High School, Peoria, Ill. )1–04, Mech. Engrg., Univ. of Illinois, Urbana, Ill. 1902, Experimental Dept., me Harvest Machine Co., Peoria, Ill. 1903, Civil Engrg. Dept., Toledo, Peoria & estern R. R. 1904–05, Asst. Engr., Canadian Min. & Smelt. Co., Chem. Dept., ail Smelter, Trail, B. C. 1905–16, Representative Engr., Tacoma Smelter for ail Smelter shipments. 1910–12, Asst. Representative, Treadwell Min. Co., coma, Wash. 1914, Asst. Representative, Kennicott Min. Co., Tacoma, ish. 1912–13, Asst. to A. McCullough, Met., Tacoma, Wash.
Present position: Resident Engr., Consolidated Min. & Smelt. Co., Tacoma, ish.

**Howard A. Beck,** Aurora, Nev.
Proposed by W. M. Dake, Jr., R. A. Hardy, F. A. Downes.
Born 1892, New York, N. Y. Grammar and High School, New York City and nver. 1912–13, Mine Foreman, Leadville Mines Co., Lead, Mo. 1913–14, Night reman, St. Kevin Min. Co., Lead, Mo. 1915, Aurora Cons. Mine and Mill. 15, Shift Foreman, Golden Boulder Co., Fairview, Nev. 1915 to date, Filterman, nk man, Tube mill man, Aurora Cons. Mines Co.
Present position: Shift Foreman, Aurora Cons. Mines Co. Mill.

**P. McDonald Biddison,** Columbus, O.
Proposed by Samuel S. Wyer, Harry C. Reeser, Daniel M. Armstead.
Born 1884, Lane, Kan. 1904, Kansas State Agricultural College, B. Sc. in Elec. grg. 1904, Draftsman and Electrician, Cement Plant Const., Hunt Engrg. Co., a, Kansas. 1905, Sta. Engr., Iola Portland Cement Co., Iola, Kans. 1905, Eleccian, Zeigler Coal Co., Zeigler, Ill. 1905, Draftsman. 1905–06, Chief Engr., Logan ıt. Gas & Fuel Co., Columbus, O. 1906, Sales Engr., Hope Engrg. & Supply Co., plin, Mo. 1907, Asst. Chief Engr., Kansas Nat. Gas Co., Independence, Kans. 08, Const. Engr., Corcoran Cons. Co., Huntington, W. Va. 1909, Const. Engr., lumbia Gas & Electric Co., Kenova, W. Va. 1910, Mech. Engr., Ohio Fuel Supply ›., Homer, O. 1911, Mech. Engr., The Texas Co., Port Arthur, Tex. 1912–16, nst. Engr., The Ohio Fuel Supply Co., Columbus, O.
Present position: Const. Engr., The Ohio Fuel Supply Co.

**Paul Truman Boise**, Salt Lake City, Utah.
Proposed by Charles P. Brooks, Charles W. Stimpson, J. C. Jones.
Born 1888, Lincoln, Kans. 1904–06, Univ. of Utah. 1906–07, Michigan College of Mines. 1908, Shift Boss, Gemini Min. Co., Eureka, Utah. 1909–10, Supt., Yosemite Mines Co., Bingham, Utah. 1911–14, Shift Boss, Silver King Coalition Mines Co., Park City, Utah.
Present position: 1914 to date; Stimpson Equipment Co.

**Ralph H. Bourne**, Chicago, Ill.
Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.
Born 1881, Cleveland, O. 1905, Grad., Cornell Univ., Mech. Engrg. Member of Amer. Soc. of Mech. Engineers. 1906, Draftsman, 8 months, Wellman Seaver-Morgan Co. 1907, 8 months, Hoover & Mason, Chicago, Ill.
Present position: Vice-Pres. & Asst. Sales Mgr., Whiting Foundry Equipment Co., Harvey, Ill.

**Henri Brings**, Paris, France.
Proposed by E. Gybbon Spilsbury, J. E. Kennedy, P. De P. Ricketts.
Born 1879, Duren. 1889–97, Grad., Realgymnasium, Aix-la-Chapelle. 1898–1902, Technical High School, Aix-la-Chapelle. 1902, Grad. Min. & Met. Engr., Technical High School, Aix-la-Chapelle. 1902, Min. Engr. Ivory Coast Exploring Syndicate, Ltd., Salisbury House, London. Exploration work in French West Africa. 1903, Surveying and exploration work, French Guinea and Senegal for same company. 1904–07, Chief Engr. and Genl. Mgr., Société des Dragages Auriferes du Tinkisso, Paris, France. 1908–10, Chief Engr., Cie. Minière de l'Ouest Africain, Paris, France.
Present position: 1910 to date; Chief Engr. & Mgr., Cie. Formiere des Mines d'Or d'El Dorado, Paris, France.

**Percy Bennett Butler**, Joplin, Mo.
Proposed by A. F. Truex, G. B. Corless, C. A. Wright.
Born 1876, San Francisco, Cal. 1893, Grad., Commercial High School, San Francisco, Cal. 1913, Student, International Correspondence School, mining course. 1899, Parral Mill Co., Parral, Chih., Mex. 1900, Cia. Restauradora, Guanacevi, Mex. 1903, Montezuma Lead Co., Santa Barbara, Chih., Mex. 1905, U. S. Min. Co., Minas Nuevas, Chih., Mex. 1907, Veta Colorada Min. & Smelt. Co., Minas Nuevas, Chih., Mex. 1907–12, Operating for own account in Parral and Santa Barbara, Mex.
Present position: 1914 to date; Mgr. of mines in Joplin district.

**Robert Stuart Butler**, Joplin, Mo.
Proposed by A. F. Truex, G. B. Corless, C. A. Wright.
Born 1891, San Francisco, Cal. 1907, Berkeley High School, Berkeley, Cal. Student, International Correspondence Schools. 1910–13, Operating in Santa Barbara and Parral, Mex. 1914, Supt. of Paragon and other properties in Joplin district.
Present position: Supt.

**Allan B. Calhoun**, Globe, Ariz.
Proposed by W. B. Cramer, J. C. Low, E. M. Marshall.
Born 1879, Greenock, Scotland. Grade and high school education in public schools, Minneapolis, Minn. 1905, Grad., Minnesota School of Mines, E. M. 1906–07, Assayer, Shift boss, Foreman, Cornucopia Mines of Oregon, Cornucopia, Ore. 1908–09, and 1911, Engr., Cons. Min. & Smelt. Co. of Canada, Rossland, B. C. 1912–13, Supt., Motherlode Sheep Creek Min. Co., Sheep Creek, B. C. 1914, Supt., Arabian Cons. Mines, Kingman, Ariz. 1905, Supt., El Rayo Mine, Santa Barbara, Mex.
Present position: Efficiency Engr., Old Dominion M. & S. Co.

**Byron H. Carpenter**, Benton, Ill.
Proposed by B. Schettler, Frank A. Ray, Oscar Cartlidge.
Born 1891, Mentor, O. 1905–09, Willoughby High School, Willoughby, O. 1910–14, Ohio State Univ., Columbus, O., E. M. 1909–10, Rodman, Clark & Pike, Topographers and Land Surveyors, Willoughby, O.
Present position: Min. Engr., The Benton Coal Co.

**Allison Robert Chambers**, New Glasgow, N. S.
Proposed by R. E. Chambers, Thomas Cantley, J. B. Porter.
Born 1879, Halifax, N. S. High Schools, New Glasgow & Halifax. Dalhousie Univ., Chemistry. 1900–04, McGill Univ., B. S. 1896–99, Chaining and Transit-Dan, Nova Scotia Steel Co., Nova Scotia and Newfoundland. 1899–1900, Civ. Engr., mominion Iron & Steel Co., Wabana, Newfoundland. 1904 to date, Min. Engr.,

›va Scotia Steel & Coal Co., Ltd., Wabana. Resident Mgr., Wabana. Asst. Mgr., e Mines and Quarries, New Glasgow, N. S.
Present position: Asst. Mgr. on Ore Mines and Quarries.

**Marcu F. Chase,** St. Louis, Mo.
Proposed by George C. Stone, J. H. Janeway, G. S. Brooks.
Born 1876, Alton, Ill. 1897, Trinity College, Hartford, Conn., B. S. 1898–1900, ıem., Mineral Point Zinc Co. 1900–04, Acid Expert, New Jersey Zinc Co. 1904–, Supt. of Depue Plant, Mineral Point Zinc Co. 1912–15, Genl. Mgr., Mineral int Zinc.
Present position: Cons. Engr.

**Gilbert Ernest Cheda,** La Libertad, Nicaragua.
Proposed by E. W. Armstrong, Charles E. Brinker, S. M. Parker, F. B. Forbes.
Born 1884, San Luis Obispo, Cal. 1911, Dept. of Geol. and Min., Stanford Univ., .l., A. B. 1905–06, Underground, Cananea Cons. Copper Co., Cananea, Mexico. 08, Underground, Keystone Mine, Cal. 1909, Underground, Pittsburgh Silver ak Mine, Nev. 1911–16, Engr. and Mine Foreman, Babilonia Gold Mines, Ltd., Libertad, Nicaragua.
Present position: Mine Foreman, Babilonia Gold Mines.

**Charles W. Clark,** San Mateo, Cal.
Proposed by Horace V. Winchell, Robert E. Tally, W. A. Clark.
Born 1871, Deer Lodge, Mont.
Present position: Genl. Mgr., United Verde Copper Co., Jerome, Ariz.

**Isaac N. Dally,** Seattle, Wash.
Proposed by F. C. Greene, Henry L. Manley, Amos Slater.
Born 1872, Webster City, Iowa. 1896–1901, Private development work on metal- rous mines in British Columbia. 1901–05, Geological and Coal Surveys, British ılumbia and Alberta. 1905–07, Halls Min. & Smelt. Co., Nelson, B. C.
Present position: 1909 to date; Coal Dept., Chicago, Milwaukee & St. Paul R. R.

**W. Val De Camp,** Mayer, Ariz.
Proposed by G. M. Colvocoresses, Robert E. Tally, J. L. White.
Born 1886, Cirsco, Iowa. 1908, Colorado School of Mines, E. M. 1908, Foreman, las Min. Co., Ouray, Colo. 1909–13, Supt. and Mgr., Pacific Copper Co., Crown ng, Ariz. 1913, Supt., Haynes Copper Co., Jerome, Ariz. 1914, Foreman and ›ss, United Verde Copper Co., Jerome, Ariz. 1915–16, Supt., Blue Bell Mine. ayer, Ariz.
Present position: Supt., Blue Bell Mine.

**H. DeWitt Decker,** Anaconda, Mont.
Proposed by Charles D. Demond, C. R. Wraith, Enoch A. Barnard.
Born 1892, La Porte, Ind. 1914, Grad. Wisconsin School of Mines. 1915, nos., Machineman, Frontier Mining Co.. Benton, Wis.; 3 mos., Shaft and Machine- ın, Admiralty Zinc Co., Guapan, Okla. 1916, Washoe Reduction Works, Anaconda ›pper Mining Co., Anaconda, Mont.
Present position: Testing Dept., Anaconda Copper Mining Co., Anaconda, Mont.

**Axel Eric,** Salt Lake City, Utah.
Proposed by R. C. Gemmell, H. C. Goodrich, D. C. Jackling.
Born 1879, Denmark. 1897, Grad., High School. 1897–1900, Machine Shop actice. 1907, Grad., Copenhagen Univ. of Polytechnic, Mech. Engr. 1906, Worked r Prof. Rung, projecting electrification of State Railroads near Copenhagen. 07–09, Messrs. Lossen & Hjorth, London, England, steam and water softening ınts. 1909, Franklin Railway Supply Co., N. Y. 1910, Midvale Steel Co. and nerican Bridge Co., Philadelphia. 1911, Robins Conveying Belt Co., N. Y. 1912–, Designing and erecting Leaching Plant, Chile Exploration Co., N. Y., and Chile, A.
Present position: Designing Leaching Plant, Utah Copper Co.

**C. L. French,** Ely, Nev.
Proposed by J. A. Rule, A. M. Day, A. B. Young.
Born 1890, Charlotte, Mich. 1909, College Prep., Fort Collins, Colo. 1913, ›lorado School of Mines, E. M. 1913, Classifier Man, Ray Cons. Copper Co., ayden, Ariz.; Cyanide Millman and Assayer, Churchill Milling Co., Wonder, Nev. '14–16, Cottrell Foreman, Testing Dept., Asst. Smoke Investigation Dept., ıstom Ore Sampler, International Smelting Co., Tooele, Utah.
Present position: Millman, Independent Tungsten Co.

**John Creighton Garvin,** Spokane, Wash.
Proposed by L. K. Armstrong, J. McD. Porter, A. W. Paterson.
Born 1840, County Londonderry, Ireland. 1848–69, Common and Grammar Schools, 1873, Queens prize in Geology in Marghafalt National School, Dublin. 1873, Examinations Eastern States Mineral Belt, Colombia, S. Amer. 1881, 6 years gold and silver reduction works, Philadelphia. 1881–97, 10 years in Leadville; balance at Puzzler Mine, Boulder Co., in mining and metallurgical practice. 1897 to date, at Spokane in professional work.
Present position: Consulting Engineer.

**Augustus Brunner Hardie,** Philadelphia, Pa.
Proposed by A. W. Hudson, H. H. Colley, Utley Wedge.
Born 1886, Philadelphia, Pa. 1895–1903, Girard College, Philadelphia. 1903–04, Franklin Institute, Philadelphia. 1908, Univ. of Penn. 1910–12, Univ. of Wisconsin. 1903–05, Shop practice, etc., Queen & Co., Inc., Philadelphia. 1905–07, Draughtsman, Standard Steel Works, Pa. 1907–08, Draughtsman, General Electric Co., Philadelphia. 1909, Asst. Plant Engr., Stanley G. Glagg, Pa.; Die Designer, Pressed Steel Car Co., Pittsburgh. 1910–11, Const. Supt., Tennessee Copper Co., Tenn. 1912–14, Const. Supt., Braden Copper Co., Chile. 1914, Design, supervision of erection, metallurgical plants with Utley Wedge.
Present position: Supt. of erection, Sulphuric Acid Plant, for Utley Wedge at Calumet & Arizona Copper Co..

**Fredrik Jörgen Ording Hurum,** Pittsburgh, Pa.
Proposed by Stephen L. Goodale, Robert M. Black, Horatio C. Ray.
Born 1893, Christiania, Norway. 1911, Examination Artium, Christiania. 1911–14, Royal German Inst. of Tech., Aachen. 1914, Cand. rer. Met., Aachen, Germany. 1915–16, Univ. of Pittsburgh, Met. Engrg.
Present position: Met. Engr., Univ. of Pittsburgh.

**William Wallace Inglis,** Scranton, Pa.
Proposed by H. M. Warren, J. M. Humphrey, Paul Sterling.
Born 1871, Scranton, Pa. High School, Scranton, Pa. Hillside Coal & Iron Co. Pennsylvania Coal, N. Y., Susquehanna & Western Coal Co.
Present position: Genl. Mgr., Coal Min. Dept., Delaware, Lackawanna & Western R. R. Co.

**Junzo Inouye,** Kenjiho, Koshin, Korea.
Proposed by Kenroku Ide, Shinji Harada, Takeshi Kawamura.
Born 1888, Osaka, Japan. 1913, Grad., Min. & Met. Institute, Imperial Univ., Kyoto, Japan. 1913, Mitsubishi Co., Tokyo, Japan.
Present position: Met. Engr., Iron & Steel Dept., Mitsubishi Co.

**Phillips F. Jarvis,** St. Louis, Mo.
Proposed by Frederick K. Copeland, P. N. Moore, E. J. Rossbach, O. M. Bilharz, H. T. Walsh.
Born 1878, Claremont, N. H. 1897, Stephens High School, Claremont, N. H. 1897, Sullivan Machinery Co., Chicago office. Shop practice, sale of mining machinery, metal mines and quarries, and specializing considerably in coal mining. Have been connected with metal mines and was president, Columbia Zinc Co., Joplin, Mo.
Present position: Mgr., Sullivan Machinery Co.

**Edward Charles King,** Humboldt, Ariz.
Proposed by G. M. Colvocoresses, J. N. D. Gray, A. F. Bassett.
Born 1875, Rockland, Mich. 1880–90, Public School. 1893, Bryant & Stratton Business College, Buffalo, N. Y. 1905, Special course in metallurgy, Michigan College of Mines. 1890–1900, Calumet & Hecla Smelteries, Lake Linden, Mich. & Buffalo, N. Y. 1900–04, In charge of the Casting Dept. and later Night Supt. Washoe Smeltery, Mont. 1904–05, Asst. Supt., Arizona Smelt. Co. 1905–07, Supt. of Smelter, Orogrande, N. M. 1907–10, Asst. Supt. of Smelter, Cananea Cons. Copper Co., Cananea, Mex. 1910–13, Met. on staff of southern division, Amer. Smelt. & Ref. Co.
Present position: 1914 to date; Supt. of Smelters, Cons. Arizona Smelt. Co.

**Victor E. Lieb,** Elephant Butte, N. Mex.
Proposed by L. J. Charles, S. H. Worrell, William B. Phillips.
Born 1883, Brenham, Tex. 1900–02, Academic course, Polytechnic Inst. of Brooklyn. 1903–07, Min. Engrg., Univ. of Tex., worked for W. Waldo and F. A. Jones of Houston, Tex., general engineering practice. 1907–08, Supt., Las Machos

Co., Tude Durango, Mex. 1908–09, Mine examination, prospecting and de-
ment of own property, Chih., Mex.; chem. and engr., Southern Min. Co.
npo, Chile. 1909–10, Engr., Candelaria Min. Co., San Pedro, Chih., Mex.
to present, Engr. and foreman, U. S. Reclamation Service.
resent position: Foreman, U. S. Reclamation Service.

**lexander Watts McCoy,** Ponca City, Okla.
roposed by E. De Golyer, R. A. Conkling, Dorsey Hager.
orn 1889, Independnce, Mo. 1912, Univ. of Missouri, C. E., 1914, M. A.
–13, Engrg. Experiment Station, Univ. of Missouri. 1914–16, Instructor,
., Univ. of Oklahoma.
resent position: Geol., Morland Oil Company.

**ichard Southgate McIntyre,** Namtu, North Shan States, Burma.
roposed by Andrew C. Lawson, Walter S. Weeks, T. A. Rickard.
orn 1895, Pittsburgh, Pa. 1910–12, Preparatory work, College of the City of
York, New York, N. Y. 1912–16, Univ. of California, B. S. 1915–16, Asst.
stician, U. S. Geological Survey, San Francisco.
resent position: Asst. Min. Engr., Bowdwin Mine, Burma. Corpn., Ltd., Namtu,
h Shan States, Burma.

**harles Mayott,** Aurora, Nev.
roposed by W. M. Dake, Jr., R. A. Hardy, F. A. Downes.
orn 1891, Salt Lake City, Utah. Grade Schools, etc., Salt Lake City. 1908–09,
ker, Red Top Mine, Goldfield, Nev. 1909–11, Machine man, Timber man, Red
Mine. 1911–13, Engrg. Dept., Goldfield Cons. Mines Co., Goldfield, Nev.
–14, Surface Boss, Aurora Cons. Mines Co., Aurora, Nev. 1914–15, Shift Boss,
ra Cons. Mines Co., Aurora, Nev.
resent position: 1915 to date; Mine Supt., Aurora Cons. Mines Co.

**hitney Player Mee,** Santa Rita, N. Mex.
roposed by J. M. Sully, Horace Moses, Frank R. Wicks.
orn 1891, Salt Lake City, Utah. 1900–09, Public Schools, El Paso, Tex. 1909–
niv. of Cal., Berkeley, Cal., B. S. 1911, summer, Surveying, Mesilla Valley, U. S.
. 1912, summer, Mining, Copper Queen Mines, Bisbee, Ariz. 1913–14, Effi-
y Engr., Oakland Lamp Works, of General Electric Co. 1914, Chem. and Asst.
., Shannon Copper Co., Gleeson, Ariz. 1914, Solution man, Simonds and Latham
nide Plant, Melones, Cal. 1914–15, Assayer, A. S. & R. Co., smelter, Asarco,
, Mex. 1915, Car checker and jigger boss, Ray Copper Co., Ray, Ariz. 1915,
yer, Asarco, Dgo., Mex.
resent position: 1915 to date; Chem., Chino Copper Co.

**Villiam Bruce Montgomery, Jr.,** Cleveland, O.
roposed by J. A. Ede, Thomas T. Read, William H. Shearman.
orn 1880, Lynchburg, Va. 1900, Virginia Military Institute, Lexington, Va.,
. 1909, School of Mines, Columbia Univ., E. M. 1901–06, Engr., Arlington
C. Co., Arlington, W. Va., and Gilliam C. & C. Co., Gilliam, W. Va. Shawnee
C. Co. and Glen Alum Coal Co., Glen Alum, W. Va. 1909–15, Engr., examina-
and report work, The Grasselli Chemical Co., of Tenn., New Market, Tenn. Mgr.
year of operation. Since 1915, The Grasselli Chemical Co., Cleveland, O. Prin-
ly examination and report work, on zinc propositions.
resent position: Min. Engr., The Grasselli Chemical Co.

**dgar L. Newhouse, Jr.,** Salt Lake City, Utah.
roposed by P. A. Mosman, C. W. Whitley, C. W. Stimpson.
orn 1890, Moncton, N. B., Canada. 1906–11, Columbia Univ., Min. & Met.
–14, Chem., and Assayer, East Helena Plant, American Smelt. & Ref. Co., Mon-
. 1914–16, Asst. Met., Garfield Smelt. Co., Utah.
resent position: Mgr., Garfield Chemical & Mfg. Corpn.

**hase S. Osborn,** Sault Ste. Marie, Mich.
roposed by O. C. Davidson, C. H. Baxter, George J. Eisele.
orn 1860, Huntington Co., Ind. LL.D. from several universities. Former
ernor of Michigan, etc. 1883–1916, Chairman, Engineering Committee Board
egents, Univ. of Mich. Field work in Lake Superior districts of Michigan, Wis-
in, Minnesota, Ontario, in Russia, Germany, Sweden, Lapland, Africa, Asia,
agascar and South America, Cuba, China, etc., etc.
resent position: Explorer and Prospector.

**Dale L. Pitt,** Tacoma, Wash.
Proposed by Willard V. Morse, Brent N. Rickard, Roger E. Chase.
Born 1884, Salt Lake City, Utah. 1899–1903, Salt Lake High School. 1903–07, Utah School of Mines, B. S. 1907–09, Asst., J. W. Wade, M. E., Salt Lake City. 1909–12, Field Supt., B. F. Tibby Co., Salt Lake City. 1912–15, Chief Engr., Federal Lead Co., Flat River, Mo. 1915–16, Min. Eng., Salt Lake City.
Present position: Min. Eng., Tacoma Smelting Co.

**Rastus S. Ransom, Jr.,** Golden, Colo.
Proposed by Charles F. Rand, Harry J. Wolf, W. G. Haldane.
Born 1890, New York, N. Y. 1907–09, Columbia Univ., School of Applied Science. 1910–13, Colorado School of Mines, E. M. 1913–14, Assayer, Bingham-New Haven Copper and Gold Min. Co., Bingham, Utah. 1914, Leasing and working underground, Central City and Idaho Springs Min. Districts, Colo.
Present position: Min. Engr., Golden, Colo.

**Charles Henderson Rich,** Conshohocken, Pa.
Proposed by John W. Logan, R. M. Bird, R. J. Wysor.
Born 1868, Yardley, Pa. Common school followed by technical training and special study. 1893–98, Analyst, Chemical Laboratory, Duquesne Works, Carnegie Steel Co. 1898–1901, Chem., Aliquippa Steel Co. 1901–02, Asst. Chem., Duquesne Works, Carnegie Steel Co. 1902–12, Chief Chem., Clairton Works, Carnegie Steel Co.
Present position: 1912 to date; Met., Alan Wood Iron & Steel Co.

**Herman F. Schlundt,** St. Francois, Mo.
Proposed by Waldo H. Comins, Philip H. Pipkin, James A. Caselton.
Born 1878, Rahway, N. J. 1899–13, Stevens Institute of Technology. 1903–04; 1905–06, Union Pacific R. R. 1906–07, Rock Island Lines. 1907–09, Taylor Gold Min. Co. 1909–11, Northwestern States Portland Cement Co. 1911–12, Cia. Metallurgica de Torreon. 1912–13, Amer. Smelt. & Ref. Co.
Present position: 1913 to date; St. Louis Smelt. & Ref. Co.

**Ellwood Spencer Smith,** Humboldt, Ariz.
Proposed by G. M. Colvocoresses, J. N. D. Gray, A. F. Bassett.
Born 1890, Brooklyn, N. Y. 1895–1902, Grade Schools, New York, N. Y. 1902–06, High School, New York, N. Y. 1906–10, U. S. Naval Academy, Annapolis. 1911, Post Graduate School, Univ. of Illinois. 1912, Clerk, International State Bank, Trinidad, Colo. 1914, Millman, Cons. Arizona Smelt. Co. 1915, Engrg. Staff, Cons. Arizona Smelt. Co.
Present position: Asst. to Genl. Mgr., Cons. Arizona Smelt. Co.

**Frank J. Staral, Jr.,** Cleveland, O.
Proposed by S. H. Pitkin, Franklin Moeller, Joseph Struthers.
Born 1878, Akron, O. Akron and Cleveland Grammar Schools. 1899, Grad., Cleveland Central High School. 1902–05, Case School of Applied Science. 1904, Grad., Columbia School of Mines, E. M. 1904–06, Surveying Engr., The Boston and Montana Cons. Copper and Silver Min. Co., Butte, Mont., also experimental work at Great Falls Smelter. 1906–07, Draftsman and Surveying Engr., Cia Real del Monte y Pachuca, Pachuca, Hdg., Mex. 1907–13, Cashier, Min. Engr. and Supt., American Smelt. & Ref. Co., and allied interests in Mexico. 1913–14, Examination work and Mgr., The Assets Gold Min. Co., Jefferson City, Mont. 1914–16, Gaylord W. Feogo Co., Cleveland, O.
Present position: Supt., The Investors Co.

**William Carl Strohbach,** Hayden, Ariz.
Proposed by David D. Moffat, William T. MacDonald, David M. Kay.
Born 1891, San Francisco, Cal. 1896–1904, Public School, Oakland, Cal. 1904–08, Polytechnic High School, Oakland, Cal. 1909–14, Univ. of Cal., Berkeley, Cal., B. S. 1910–14, Summers, underground and cyanide plant, North Star Mine and Empire Mine, Grass Valley, Cal. 1915, Attendant at Bureau of Mines Exhibit, P. P. I. E., San Francisco, Cal.
Present position: 1915 to date; Chemist, Ray Cons. Copper Co.

**Clarke Sullivan,** Oakland, Cal.
Proposed by H. M. Alley, Robert G. Davies, E. E. Carpenter.
Born 1882, Atlanta, Ga. 1902, Oakland High School. 1907, Univ. of Cal., B. S. 1907–11 Assayer and Mill Supt., Progreso Min. Co. 1911–14, Mill Supt. and

l. Supt., Zopitoto Min. Co. 1914–15, Const. Supt., Mill work, Arrondo Min. Co. i. Mill work, Churchill Mill. Co.,
Present position: Mill Supt., Eden Min. Co., Nicaragua, C. A.

**E. L. Talbot,** Salt Lake City, Utah.
Proposed by Alfred Frank, George H. Dern, Ernest Gayford.
Born 1862, Central City, Colo. Public grade school also High School. Daly t Min. Co., Park City, Utah. Ontario Silver Min. Co., Park City, Utah.
Present position: Genl. Supt., Daly West Min. Co., also Ontario Silver Min. Co.

**Floyd Alpine Warren, Sr.,** Collinsville, Ill.
Proposed by Jesse O. Betterton, A. T. March, W. E. Newnam.
Born 1886, Hiawatha, Kans. 1904, Academy, Hiawatha, Kans. 1910, Highland : College, Des Moines, Ia. 1910–15, Shift Boss, Foreman, Supt., Pattinson it Dept., Experimental work and Assaying, American Smelt. & Ref. Co., Omaha, r.
Present position: Supt., Blast Furnace, Roasting Depts. and Newnam Hearths, Louis Smelt. & Ref. Co.

**Sidney E. Werthan,** Bisbee, Ariz.
Proposed by Gerald Sherman, Arthur Notman, H. C. Henrie.
Born 1891, Denver, Colo. 1910, Public Schools and High School, Denver, Colo. ;, Univ. of Denver, B. A.; 1915, M. A. 1913–14, Instr., Dept. of Chem., Univ. Denver. 1915, Research work on Flour, Great Western Mill. & Granary Co. i, Mine Laboratory, Copper Queen Cons. Min. Co.
Present position: Chemist, Copper Queen Cons. Min. Co.

**George Hazzard West,** Globe, Ariz.
Proposed by William B. Cramer, L. O. Howard, Walter Harris.
Born 1882, Philadelphia, Pa. 1896–1900, Central High School, Philadelphia, B. A. 1900–04, Univ. of Penn., Philadelphia, Pa., B. S. 1904–06, Assayer, on Mines Co., Congress Junction, Ariz. 1906–07, Chem., Cons. Arizona Smelt. Humboldt, Ariz. 1907–11, Chem. and Engr., Orogrande Smelt. Co., Orode, N. M. 1911 to date; Chem. and Smelter Foreman, Old Dominion Copper . & Smelt. Co., Globe Ariz.
Present position: Smelter Foreman, Old Dominion Copper Min. & Smelt. Co.

**Henry Palmer Westcott,** Pittsburgh, Pa.
Proposed by R. B. Woodworth, Thomas T. Read, William H. Shearman.
Born 1878, Seneca Falls, N. Y. High School at Seneca Falls and Rochester, Y. Five years, Alden Bateria National Gas Co., Alden, N. Y. 1908 to date; ric Metals Works, Erie, Pa. Author, Hand Book of Natural Gas; Measuret of Gases Where Density Changes. Now in press, Hand Book of Casinghead

Present position: Engr., National Gas Construction.

**Garratt S. Wilkin,** Moscow, Utah.
Proposed by Edward R. Zalinski, Earl R. Pembrooke, C. W. Stimpson, W. A. on, Ernest Gayford.
Born 1882, New Philadelphia, Pa. 1895–99, New Philadelphia High School. –02, Western Reserve Univ. 1904, Miner, Katherine Mine, Stockton, Utah. i, Foot nipper, Bingham and New Haven Min. Co., Bingham, Utah. 1905–07, pler and Foreman, Boston Cons., Bingham, Utah. 1907–09, Supt., Utah Copper Bingham, Utah. 1909–11, Supt., Wild Bill Min. Co., Milford, Utah.
Present position: 1911 to date; Supt. and Asst. Mgr., Moscow Min. & Mill. Co.

**Charles F. Willis,** Tucson, Ariz.
Proposed by Arthur Notman, Roger T. Pelton, Gerald F. G. Sherman.
Born 1885, Boston, Mass. 1902–06, Mass. Inst. of Tech., B. S. 1906–09, ayer to Supt., Enterprise Min. Co., Cooney, N. M. 1909–10, Mgr., Pelican Min. Mill. Co., Lake City, Colo. 1910–11, Mgr., Granite State Mica Co., Alstead, N. H. l, Prof. of Min. Engrg. and Director, Arizona State Bureau of Mines, Univ. of ona, Tucson, Ariz.
Present position: Director, Arizona State Bureau of Mines.

**Harold Edwin Willson,** Urbana, Ill.
Proposed by H. H. Stoek, C. M. Young, C. E. Krebs.
Born 1881, Lykens, Pa. 1900, Grad., Baltimore City College. 1916, Univ. of B. S. 1900–01, U. S. Fidelity and Guaranty Co., Baltimore, Md. 1901–13, inman, rodman, levellor, transitman, office man and chief engineer, McKell Coal

& Coke Co., Glen Jean, W. Va. 1904–13, Engr., McKell Heirs. 1905–13, Engr., Nichol Colliery Co. 1904–13, Engr., Laura Min. Co.
Present position: Student.

**Frank A. Woodward,** Cooper Hill, Ariz.
Proposed by B. Britton Gottsberger, P. G. Beckett, C. E. Mills.
Born 1852, Worcester, Mass. General education only. 1906 to date, Iron Cap Copper Co.
Present position: Supt., Iron Cap Copper Co.

**Herbert Watson Woodward,** Copper Hill, Ariz.
Proposed by W. B. Cramer, C. Mendelsohn, Walter Harris.
Born 1883, So. Framingham, Mass. 1900, Superior, Wis., Grammar and High Schools. 1900–02, High school, Somerville, Mass. 1903–06, Tufts College. 1906–08, Surveying mining claims. 1908, Assayer, Mine Surveyor, Copper Basin Min. Co. (defunct), Nev. 1908, Cyanide Mill, North Star Mine, Grass Valley, Cal. 1909–16, Engr., Iron Cap Copper Co., Copper Hill, Ariz.
Present position: Asst. Supt., Iron Cap Copper Co.

**Yeh-Tze Yen,** Hanyang, China.
Proposed by Chin Tao Huang, Cho Yang, C. F. Wang.
Born 1886, Kiangsu, China. 1901–03, St. John's College, Shanghai. 1904–07, Kyoto High School, Japan. 1907–10, Min. course, Kyoto Univ., Japan. 1911–12, Engr., Pinchuho Iron & Coal Co.
Present position: 1912 to date; Engr., Blast Furnace Dept., Hanyang Iron & Steel Works.

**Stuart M. Young,** Bingham, Utah.
Proposed by A. M. Day, J. A. Rule, A. B. Young.
Born 1890, Salt Lake City, Utah. 1896–03, Salt Lake Public schools. 1903–07, Salt Lake High School. 1907–11, Univ. of Utah, B. S. 1911–12, Chem., Grasselli Chemical Co., Park City, Utah.
Present position: 1912 to date; Assayer, Min. Dept., United States Smelt. Co.

## *Associate Members*

**William Wright Armstrong,** Salt Lake City, Utah.
Proposed by C. W. Stimpson, Ernest Gayford, Duncan MacVichie.
Born 1865, Darlington, Wis. 1885, Baker Univ., Baldwin, Kans., B. S. 1887, Univ. of Wisconsin, Madison, Wis., LL. B.
Present position: Pres., National Copper Bank.

**Henry Ayman Bomberger,** Philadelphia. Pa.
Proposed by Arthur J. Hoskin, F. W. Smith, J. W. Finch.
Born 1866, Philadelphia, Pa. 1884, Ursinus College, Pa., A. B.; 1886, A. M. 1900, Vice-Prest., Temple Univ., Philadelphia. 1905, Prest. & Genl. Mgr., Southwestern Dev. Co. 1912, Genl. Mgr., Minnesota-Connor Min. & Mill, Co. Have had no special technical training in the schools as a mining engineer. Have been a student all my life and for nearly fifteen years have made a special study of mining in all its branches, about one-third of which time has been spent in practical field work associated with some of the leading mining engineers and mine managers of the country, none of whom will know that I have ever made any claims at all as an engineer for I have never made any such claims. I personally organized the three companies above mentioned, and my services for mine reports, etc., are constantly sought without solicitation on my part.
Present position: Prest. & Genl. Mgr., Mohave Minerals Co.

**William Paul Gage,** Fort Worth, Tex.
Proposed by Samuel S. Wyer, Harry C. Reeser, David T. Day.
Born 1875, Denver, Colo. 1894, Grad., Riverside (Cal.) High School, scientific course. 1895–96, Finished Sophomore year at Marietta (Ohio) College; Latin, scientific course. 1895, summer, and from spring, 1896 to spring, 1903, Eureka Pipe Line Co. and Buckeye Pipe Line Co. in W. Va. and Ohio. Worked in all the pipe line departments on construction of lines and stations, office and field work of all kinds. 1903–09, Security Oil Co., Beaumont, Tex.; Inventory Clerk; Chief Dispatcher; Asst. to Genl. Supt., Pipe Line Dept. 1909–16, Vice-Prest. and Genl. Supt., Lone Star Gas Co., Fort Worth, Tex., active executive officer. Practical management of construction and operation almost 500 miles pipe line, supplying about

towns in North Texas in addition to development of the gas field, installation of npressor station, etc.
Present position: First Vice-Prest. and Genl. Supt., Lone Star Gas Co.

**Charles A. Gillette,** Salt Lake City, Utah.
Proposed by C. W. Stimpson, Ernest Gayford, Walker J. Boudwin.
Born 1869, Sioux City, Iowa. 1891, B. S., Washington & Jefferson. 1893, Law, iversity of Iowa.
Present position: Attorney at Law.

**William Joseph Quigly,** El Paso, Texas.
Proposed by William D. Gordon, William S. Noyes, W. N. Small.
Born 1872, Carbondale, Pa. 1896–1900, Ibex Mining Co., Leadville, Colo. 00–02, Penn M. & L. Co., Leadville, Colo. 1902–06, Chihuahua Min. Co. 1906– Genl. Agent, Chihuahua Min. Co. and El Potosi Min. Co., Chihuahua, Mex.
Present position: 1912 to date; Genl. Mgr., El Potosi Min. Co., and Chihuahua n. Co. and Director of both companies.

**Henry Denison Randall,** Salt Lake City, Utah.
Proposed by C. W. Whitley, Ernest Gayford, C. W. Stimpson.
Born 1881, Ledyard, Conn. 1902, Amherst College, B. S. 1902–03, Lawyer, ınn Electric Co. 1903–08, Westinghouse Electric Mfg. Co. 1908–12, Allis-almers Mfg. Co.
Present position: 1912 to date, Mgr., General Electric Co.

**James B. Walker, Jr.,** Salt Lake City, Utah.
Proposed by C. W. Stimpson, O. J. Salisbury, Ernest Gayford.
Born 1889, Helena, Mont. 1911, Cornell Univ., B. L., Mech. Engrg.
Present position: Mgr., Kelly Filter Press Co.

## CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at e Secretary's office during the period June 10, 1916 to July 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 115, bruary to July, 1916, and the foregoing list of new members, therefore, pplements the annual list of members corrected to Jan. 1, 1916 and ıngs it up to the date of July 10, 1916.

DERSON, VICTOR C., Cons. Engr., Winnemucca Mountain Min. Co., Winnemucca, Nev.
LEN, ARTHUR P. .......... 233 College Ave., Houghton, Mich.
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| ame. | Last Address of Record from which Mail has been Returned. |
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| s, Blakeslee | Arrow Engineering Co., Palmyra, Mo. |
| D. A. S. | 136 McLaren St., Ottawa, Canada. |

Mineral Statistics, from 1911 to 1915. He was also a member of the committee on Smelter Rates, and chairman of the committee appointed by the association to prepare the report on "The Vertical Side Line Law." At the meeting of the Congress held in Salt Lake City, a paper on "The Mining Industries of Utah" was presented by him.

For the American Institute of Mining Engineers, Mr. Riter served as a member of the Committee on Mining Laws, 1914 to 1916. He became greatly interested in the matter of the revision of laws regarding mineral lands and mining, writing a great many articles on this subject for various publications. He was also a frequent contributor to the various magazines and technical publications of the mining and metallurgical world, writing articles on: "The Elastic Hydro Carbons of Utah;" "The Oil Flotation Process;" "The Making of Mine Survey Notes;" "Mineral Land Laws;" "Character of Title to Mineral Land that should be granted by the Government;" "Cost of Silver-Lead Smelting;" "Selective Flotation."

In addition to his active engineering work, in field as well as office, he spent a great deal of time conducting extensive laboratory experiments for the treatment of various classes of ores, together with experiments into the technology of asphalts and related bitumens.

At the time of his death, Aug. 20, 1916, Mr. Riter was developing a large tungsten property near Lucin, Box Elder County, Utah. He was also consulting engineer for the Emma Copper Mine at Alta.

## F. McMILLAN STANTON

F. McMillan Stanton was born in New York City on May 23, 1865, the son of an English father and of a mother whose family was linked with that of Peter Stuyvesant.

Mr. Stanton took his degree from the School of Mines of Columbia University in 1887, worked for two years as an assayer and surveyor and then went into the mining field on his own account. After a few years, he was offered a position with the Atlantic Mining Co., and began the work which was to continue with marked success for twenty-three years.

In 1910, Mr. Stanton retired and went to Europe for a rest. Upon his return in 1914, he was elected treasurer and director of the Mohawk Mining Co., the Wolverine Copper Mining Co. and the Michigan Copper Mining Co., to which group was later added the White Pine Extension Copper Co., and continued to serve these companies until the time of his death, Sept. 12, 1916.

Mr. Stanton was also president and director of the Ft. Mountain Talc Co., a director of the First National Bank of Houghton, Mich., of the Ohio and Kentucky Railroad, and of the Copper Range Co. He was on the membership list of the following organizations: American Society of Civil Engineers, American Chemical Society, American Forestry Association, American Society of Mechanical Engineers, American Mining Congress, Navy League of the United States, American Association for the Advancement of Science, Sons of the Revolution, Lake Superior Mining Institute, National Security League, National Rifle Association, and many others.

He became a member of this Institute in 1889.

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The Ingersoll-Rand Co. has recently issued three new bulletins:

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Form 3026, 40 page, 6 × 9 catalog on Ingersoll-Rogler Class "PRE" Duplex Direct-Connected Electrically Driven Air Compressors. Among the principal features of design may be mentioned the "Ingersoll-Rogler" Valve, the "Clearance Controller," Direct-Connected Drive, Improved Intercooler and Auxiliary Water Separator. Catalog is printed in two colors and profusely illustrated to show construction details. Four pages are devoted to pressure charts and tables showing sizes and capacities.

Form 3312, 20 page, 6 × 9 catalog on Imperial "XB" Duplex Power-Driven Air Compressors. Illustrated.

Electric "Industrial Klaxons," made by the Lovell-McConnell Mfg. Co. and distributed by the Western Electric Co., are pictured and described in a pamphlet advocating their use as fire alarms and call signals. They are now being used on moving cranes to warn the workmen to stand from under, and are suggested for use in mines and quarries as time, blast, and elevator signals; for use on electric locomotives, underground telephones, etc.

From the Western Electric Co., also, come three bulletins telling of "Square D" switches and motor starters made by the Detroit Fuse and Mfg. Co. These articles bear the Fire and Accident Label of the Underwriters' Laboratories, Inc.

Bulletin No. 37 deals with Steel Enclosed Switches; Bulletin No. 40 with Ironclad Fused Switches, Plunger Type; and Bulletin No. 41 with Induction Motor Starters. Specifications, list prices and dimensions are given.

 

# INDUSTRIAL SECTION

## The Jeffrey Company Opens Branch in Seattle

The Jeffrey Manufacturing Co. of Columbus, Ohio, announces the re-opening of its Northwestern Branch Office at Seattle, Wash., and the appointment of Mr. Percy E. Wright, Consulting Mechanical Engineer, as District Manager for Oregon, Washington, Alaska, British Columbia and Alberta. Mr. Wright, who has been in the Northwest since 1910, and whose connection with the Jeffrey company dates back to 1902, has had a wide and varied experience and training in the handling of the Jeffrey line which will enable him to be of great assistance to customers in solving their elevating, conveying and transmission problems.

## Trade Catalogs

To a majority of the public, the name DuPont suggests military powder—munitions of war. The *DuPont Products Book* A-430 (April, 1916) lists 251 distinct commodities made and sold by this company, many of which are far removed from destructive work.

Mining men are fully cognizant of the need for explosives in constructive work, but they may be surprised to know that side by side with the powder and dynamite this company manufactures such peaceful articles as substitute leather, combs and collar buttons.

The substitute for leather is *Fabrikoid*, varying in weight from a quality light enough for bookbinding to one heavy enough for trunks and automobiles, and the combs and collar buttons (with scores of other articles) are made from *Pyralin*.

The book also contains a long list of chemicals, and a shorter list of special products and miscellaneous materials such as charcoal, wood flour and saltpeter.

The Jeffrey Manufacturing Co. of Columbus, Ohio, making elevating, conveying, crushing, screening and power transmission machinery, has just issued its latest Bulletin, No. 177, featuring self-propelling wagon and truck loaders for handling crushed stone, sand, gravel, clinker, coke, etc.

These machines are described as having a capacity of 1 to 1½ cu. yd. of material per minute, with power to load crushed stone maximum size pieces through 3½-in. ring and to load trucks or wagons in 3 or 4 min.

The self-propelling device is designed to enable one man to operate the loader as it is fed into the pile, and to move the machine about under its own power.

 

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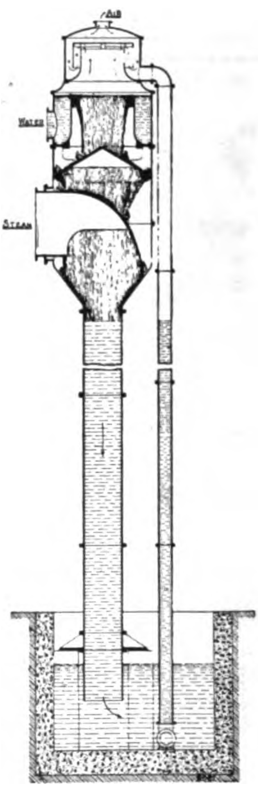
it mixes with the steam. This, the manufacturer points out, not only facilitates the mixing process but permits the removal of air and vapor at a comparatively low temperature, a distinct advantage, as the reduced volume saves in vacuum pumpage horsepower.

The steam inlet is of large diameter to secure low velocity and is hooded in such a way as to discharge the steam into the center of the condensing vessel. The air-removal opening is also of ample area and is protected by a self-draining baffle and trap. This, it is said, positively prevents water being carried over into the vacuum pump.



SECTION. BEYER BAROMETRIC CONDENSER.

The hot waste water is discharged through the self-draining tail pipe. This pipe straddles the hot well and rigidly supports the condenser.

The Imperial and Ingersoll-Rogler vacuum pumps are of the manufacturers' standard type. They are high-speed reciprocating machines, wholly inclosed, automatically lubricated and are said to effect a floor space saving, approximating 50 per cent. over the more common slower-speed vacuum pumps.

When a water pump is required to elevate cooling water to the condenser head, Cameron pumps are provided. These may be either reciprocating or centrifugal, as desired. The Ingersoll-Rand Co., however,

emphasizes the fact that, where the level of the cold well is of sufficient height above the hot well, the condenser will lift its own cooling water, dispensing entirely with a water pump.

## Permissible Explosives and Electric Firing

E. I. DuPont de Nemours & Co. have issued a circular directing the attention of coal-mine operators to the subject of permissible explosives and electric firing, from which the following paragraphs are quoted:

According to statistics compiled by the U. S. Bureau of Mines, there were slightly more than 288,000 lb. of permissible explosives used in American mines in the year 1903, and the fatalities during that year, directly traceable to explosives, were 0.339 per 1,000 men employed.

In 1914 over 15,000,000 lb. of permissible explosives were used and the fatality rate dropped to 0.096 per 1,000 men employed. A reduction of 72 per cent. Again, in 1870 there were 13.47 fatal accidents for every million tons of coal mined in this country and in 1914 only 4.78 for every million tons mined.

There is still room for an even better showing than the above, because, compared to European statistics, we are still behind in the matter of safety and protection in coal mining.

The Bureau of Mines Statistics also show that of all the accidents due to explosives in coal mines 25 per cent. are caused by premature blasts caused by miners using short fuse or defective squibs.

When explosives are detonated electrically, the maximum execution is always assured. The blast cannot occur until the miner desires it and when everyone is in a safe place. As many holes as desired can be fired at one time, so that it is possible to use less explosive per ton of coal than when firing by fuse and blasting caps.

The cost of installation is low. A blasting machine suitable for use in coal mines costs about $10 and will last for years if properly cared for. Electric blasting caps cost little, if any, more than fuse and blasting caps, and the cost of leading wire is small. The advantages and benefits gained greatly outweigh the small first cost of installation.

## Jeffrey Manufacturing Co. Opens Branch Offices at Dallas and St. Louis

The Dallas office will be located in the Commonwealth National Bank Building, and will be in charge of Mr. J. U. Jones. Mr. Jones has 20 years of successful salesmanship to his credit, and a thorough personal knowledge of the State of Texas and the requirements of its buyers.

Mr. W. V. Cullen has been placed in charge of the St. Louis office, with headquarters in Room A-21 of the Railway Exchange Building. Long experience in sales engineering work, and training in the application of Jeffrey products, qualify Mr. Cullen for handling this work.

# INDUSTRIAL SECTION

## Some Recent Inventions in Blasting

The Du Pont Magazine for August and September gives the following interesting information regarding recent progress in the use of explosives.

*Blasting Clay.*—"The place where I tried blasting down clay for a brick yard had been worked for some years by old-time pick and shovel method. There was a large shallow pit, irregular in shape, with a perpendicular face at one end. I put down a line of holes 3 ft. deep and 6 ft. back from this face, loading each hole with one and one-half cartridges of Red Cross Extra 40 per cent. dynamite. The charges were connected up to a blasting machine circuit and fired electrically.

"Two of these shots kept the brick yard men very busy for two days taking out the loose clay.

"I am convinced that if any brick yard man will try this method he will always have a blasting machine and dynamite on hand thereafter, because it cuts labor cost almost in two to get down the clay in this way."

In some clay pits the practice is to blast a large amount of clay in the fall, allowing it to freeze and thaw through the winter. Such clay is much easier to puddle in the spring than freshly dug or blasted clay.

When the clay contains stones, the blasting loosens them and makes their separation easy.

*Well-Drill Holes in Quarrying.*—The well drill or cable drill method of working a quarry face is now in quite common use. The use of large drills is constantly increasing among quarrymen. In fact the growing tendency is for large machinery of all kinds such as mammoth crushers, steam shovels, etc. Volume is what is wanted nowadays.

The advantages of the big drill over the piston driven drill are numerous. By carrying the holes the full depth of the face the stone is loaded and treated on one working floor. It eliminates the cost of bench cleaning. Loading operations are less frequent, thus lessening the danger. Stone can be shot down in greater volume and in better shape, making it economical to use a steam shovel. No bench cleaning makes it safer for the men. It is cheaper and can be used in outlying and temporary operations where the cost of installing a power plant would be out of the question. It permits a greater spacing of bore holes and eliminates the necessity of "squibbing" or springing.

In this method usually holes from 4 in. to 6 in. in diameter are drilled the full depth of the face from 15 ft. to 20 ft. apart and as far back and the whole face shot down at one time.

The cost of well drilling may vary all the way from 20 cents to 75 cents per foot, and in extremely hard rocks up to $1.50 per foot.

*The Unsafe Combination Shot.*—In some of the coal mines of this country, especially those in the anthracite field, the practice of making what is known as "the combination shot" is common. It is so called because a charge of dynamite and a charge of blasting powder are used in the same bore hole. Laws have been passed and are still in the Mining Laws prohibiting the making of combination shots, but the reprehensible practice is still employed by the miners to a considerable extent, with or without the assent of the operators. According to the best authorities and theory, the combination shot is both dangerous and inefficient. It is very probable that in such shots, the dynamite is set on fire by the blasting powder, burns for a while, then upon reaching the critical temperature, explodes.

*Blasting Methods to Decrease the Proportion of Fine Material.*—In the winning of ores from their natural beds by blasting, physically fine flaky material is not desired and is a source of serious loss to mine operators.

High grade, quick acting explosives break down the mineral into minute particles, especially in the zone adjacent to the explosive charged surfaces in the drilled hole. There is also no doubt a fusing effect exerted on zinc ores and other easily reducible ores from the softer rocks. This action is decided, especially where holes are drilled and charged in the ore bodies proper.

Explosives exert the disruptive, crushing and disintegrating effect throughout the burden in direct ratio to the action of the explosive. Low grade, slower acting explosives will not be as severe, or produce proportionately the resultant physically fine ore from the blasted burden, as will the higher grades and quicker acting explosives. It will be necessary to do more boulder blasting when using low grade explosives if the explosive charge in the drilled hole is not sufficiently tamped with "stemming." Dynamite in a drilled hole without "stemming," when exploded, will exert a great deal of its force on the material which holds it, but as the effect of the explosive is in line of the least resistance, the natural result is, that the charge in an untamped hole will expend a great part of its energy is blowing back out of the hole. Explosive charges should, therefore, be "stemmed." The reason for this is set forth in the formula:

$$PV = RT$$

in which $P$ = Pressure and $V$ = Volume; $R$ a constant for each Gas and $T$ = Absolute Temperature. If Temperature remains the same, no matter how Pressure or Volume vary individually, their product will be equal to the constant $RT$. If Temperature increases and remains constant, $R$ being already constant, nothing is left but for Pressure to increase in the same ratio as Temperature, and the resultant efficiency of the explosive is increased, as will be shown by the quality of the work done. The quality of mine air will also be much better after the blast.

ports. This deflection should be maintained, and to this end the rope should be pulled in or out as the temperature rises or falls during the course of the year.

"To find the tension in a cableway rope, the following simple method is very convenient: For a deflection equal to 5 per cent. of the span (the load being at the center of the span) the tension in the rope is equal to five times the load added to two and one-half times the weight of the suspended rope."

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## Trade Catalogs

Bulletin 183, issued by The Jeffrey Manufacturing Co., deals with mine ventilation. It illustrates and describes the various kinds of Jeffrey ventilating fans, fan installations and fan casings. It also contains ventilating formulæ for Jeffrey fans, and tables compiled to give a comprehensive idea of the results to be obtained from the various sizes of the Jeffrey Stepped Multi-Bladed Fans.

The Ingersoll-Rand Co. has recently issued the following bulletins:

Form 9024, on Steam Condensing Plants—Beyer Barometric Type. This bulletin describes the fundamental principles of steam-condensing plants and makes comparisons between the Beyer Barometric Condenser and Low Level Jet Condensers and Surface Condensers. Auxiliary apparatus is illustrated and described in detail.

Form 4122 describes the IR Model Leyner Drill Sharpener. It explains and illustrates the sharpener in detail and shows the various styles of bits.

Form 3033 deals with the Imperial "XPV" Duplex Steam Driven Compressors, designed and constructed to operate satisfactorily under "high pressures" and "superheat" as well as under "ordinary steam" conditions. It shows the various sizes and capacities and explains in detail the operation of the Imperial Piston Valves.

Form 8112 describes the 56-H "Crown" Pick, a pneumatic hammer said to be especially adapted to stripping and picking coal in coal mining, breaking cores in foundry work, tearing up asphalt pavements and loosening and wedging out bricks and stones when demolishing old structures.

Catalog D, 1916, issued by the Denver Fire Clay Co., gives concise information regarding the Case Metallurgical Furnaces. The oil-burning furnaces are featured, particularly the Tilting Crucible Furnace, but many that are fired by gas, gasoline coal, wood or coke are also described. The last half-dozen pages of the pamphlet are devoted to crushers and pulverizers.

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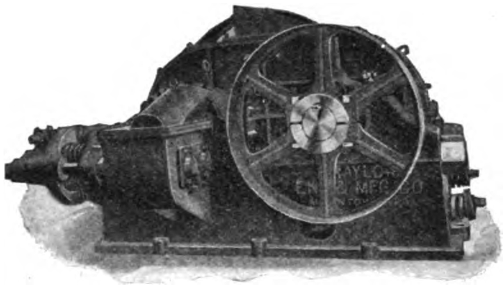

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## An Agency of the Jeffrey Manufacturing Co. in Kansas City

The Jeffrey Manufacturing Co. of Columbus, Ohio, has opened an agency with the Gustin-Bacon Manufacturing Co., a large and well known firm of Kansas City, in order to establish closer coöperation with its customers in Kansas, Missouri and Oklahoma, and improve its service there.

The Gustin-Bacon company will carry a line of Jeffrey transmission machinery, chains, buckets, etc., in stock. Mr. J. S. Davidson, who has spent several years in the sales and engineering departments of the Jeffrey company, has associated himself with the Gustin-Bacon company and will handle all inquiries for Jeffrey products. His experience and training in the application of these products will enable him to be of great assistance to customers in solving their problems in elevating, conveying and transmission machinery, also crushing, pulverizing and screening equipment.

## Trade Catalogs

The General Naval Stores Co. has issued a bulletin giving information about GNS Flotation Oils, with data on their flotation value and suggestions for their use. This bulletin is addressed to the man in the mill, rather than to the metallurgist, the object being to give information of value to those actually engaged in the treatment of ores.

Bulletin No. 12, from the Berger Manufacturing Co., Canton, Ohio, is on the subject of "Metal Lumber." In addition to a full description of the Berger metal lumber, with specifications, the bulletin gives tables showing safe load in pounds when metal lumber is used.

A folder issued by the Empire Concentrator Co., of Denver, Col., gives construction and operating details of the Empire Concentrating Table. The following advantages are claimed for the table by the manufacturers: Large capacity, clean concentrates, clean separation of different minerals, clean tailings, takes the place of the jig, small floor space, simplicity, low water and power consumption, inexpensive installation, low operating cost.

Two bulletins have been received from the Jeffrey Manufacturing Co. Bulletin No. 191 contains interesting installation views and describes the latest improvments in the Jeffrey 29-B "Arcwall" type of coal cutter for "overcutting" system of mining. This cutter is self-contained and self-propelled, is adjustable in all directions, and can be built to cut anywhere above the top of rail.

Bulletin No. 192 contains full instructions for the care and operation of the Jeffrey 35-B Shortwall coal cutter, which has been designed to meet the demand for a simple, rugged, medium-weight machine having features which make it quickly and easily handled. This bulletin also features the Jeffrey Self-Propelling Handitruck, which permits of unloading the machine at any angle to the truck and on a spot which will require the least amount of time to place it in the proper position for cutting.

 

Like the No. 1, the No. 11 Little Tugger is built for operation both by compressed air and steam. The standard clamp fits a 4½ in. diameter column or pipe, but by removing the clamp the hoist can be readily bolted directly to any convenient support—timber, flooring, etc.

Although designed primarily for underground work, it is recommended by the manufacturer for all-around hoisting, hauling and handling in mines, tunnels, quarries and industrial plants.

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## Trade Catalogs

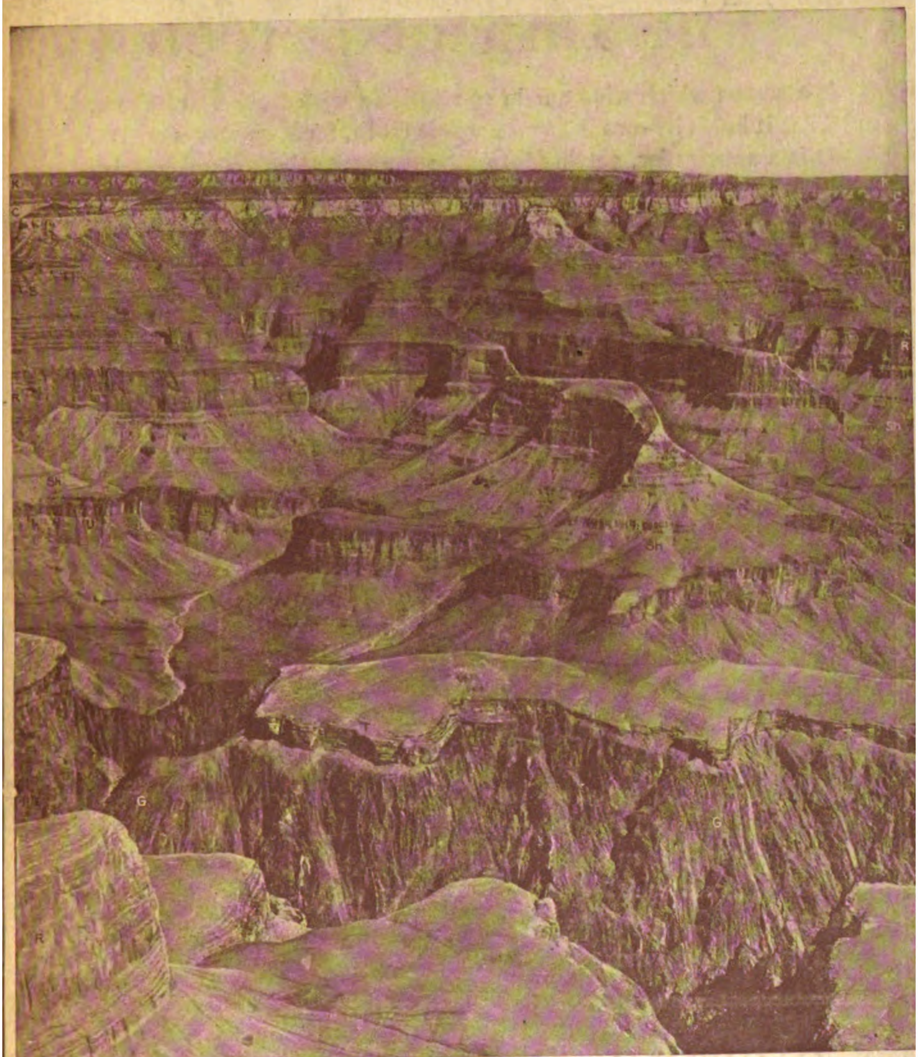
Pamphlet No. 31, issued by the Colorado Iron Works Co., deals with grinding machinery. Full descriptions are given of ball and tube mills and of grinding pans manufactured by this company. The grinding pan is said to be a close second to the tube mill for fine grinding in cyanide and similar work. The pamphlet cites one pan as having finished 16½ tons from + 18 to − 80-mesh per 24 per. In another case, the average of seven pans was 14.8 tons per day of 20-mesh feed reduced so that all passed 100 mesh.

The pamphlet also contains a short article with two tables, reprinted from *Metallurgical and Chemical Engineering* for June, 1912, on "Slime Density Relations."

On the last page, an Impact Screen and a new diaphragm pump are pictured and described.

Bulletin No. 7150 describes the Cameron Centrifugal Pumps, manufactured by the A. S. Cameron Steam Pump Works. The bulletin is illustrated, and contains, in addition to a full description of the pumps, tables showing friction of water, theoretical horsepower required to raise water to different heights, and pressure of water in pounds per square inch.

AUGUST, 1916



FOR EXPLANATION OF REFERENCE LETTERS SEE PAGE XI

SEPTEMBER, 1916



# Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

## SECTION I



## SECTION II.—PAPERS

GEOLOGY MINING

METALLURGY

VHA

OCTOBER, 1916

# Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

SECTION I



SECTION II.—PAPERS

Tech. Div.

NOVEMBER, 1916

# Bulletin of the American Institute of Mining Engineers

## PUBLISHED MONTHLY

V F A

### SECTION I



### SECTION II.—PAPERS

GEOLOGY MINING

METALLURGY

DECEMBER, 1916

# Bulletin of the American Institute of Mining Engineers

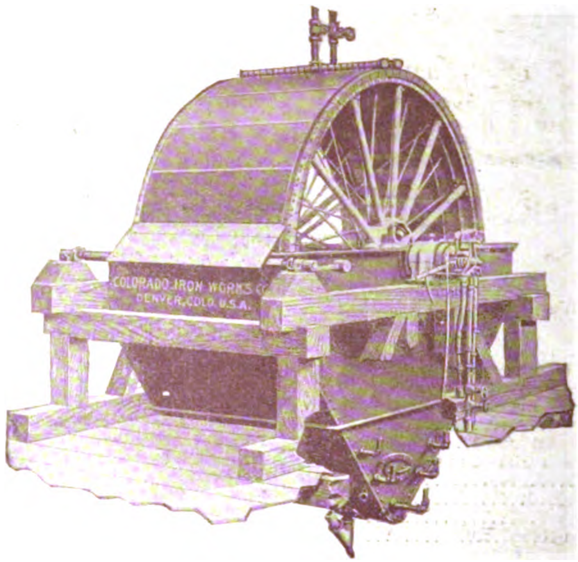
VHA

## PUBLISHED MONTHLY

### SECTION I



### SECTION II.—PAPERS



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